USBM Contract Number H0133037

CSMRI
Colorado School of Mines Research Institute

FEASIBILITY OF HYDRAULIC TRANSPORTATION IN UNDERGROUND COAL MINES

By

James M. Link
Andrew Allan, Jr.
Robert R. Faddick

United States
Department of the Interior
Bureau of Mines

USBM Contract Report H0133037

May 30, 1975

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U. S. Department of the Interior
Bureau of Mines
Washington, D. C.

FOREWORD

This report was prepared by the Colorado School of Mines Research Institute, Golden, Colorado, under USBM Contract Number HO133037. The contract was initiated under the Coal Mine Health and Safety Program. It was administered under the technical direction of PMSRC, with Mr. A. J. Miscoe acting as the Technical Project Officer. Mr. John Blum was the contract administrator for the Bureau of Mines.

This report is a summary of the work recently completed as part of this contract during the period June 27, 1973, to May 30, 1975. This report was submitted by the authors on May 30, 1975.

The successful completion of the work covered in this report was made possible by the willing cooperation and assistance of many individuals, coal and manufacturing companies, and government agencies. The work itself was conducted under contract by the Colorado School of Mines Research Institute.

Special thanks are due Mr. C. W. Schulties, Vice President Underground Operations, Peabody Coal Company, and Mr. William Laird, Vice President Research and Development, Eastern Associated Coal Corporation. These gentlemen reviewed the mining and equipment portions of the report, and provided much helpful comment.

The Colorado School of Mines Research Institute wishes to acknowledge and thank the following coal mining companies for time study data and other valuable information on their mining operations which were furnished for use in this report:

Colowyo Coal Company
Consolidation Coal Company
Mr. Frank Mink
Eastern Associated Coal Company
Mr. William Laird
Mr. Lawrence Jackson
Inland Steel Company
Mr. E. M. Pace
Mr. William S. Tate
Monterey Coal Company
Old Ben Coal Company
Peabody Coal Company
Mr. C. W. Schulties
Mr. Kelly Summers
Pittston Coal Company
Mr. Irven C. Spotte
Swisher Coal Company
U. S. Steel Corporation
Zapata Coal Company
Mr. Robert L. Purswell

The Colorado School of Mines Research Institute also wishes to acknowledge the following manufacturing companies for their cooperation in supplying valuable technical and/or price information on their equipment. Since it would be difficult to distinguish the relative values of their contributions, the companies have been simply listed in alphabetical order:

Abex Corporation
Acme Machinery Company
American Petroleum Institute
American Standard Industrial Products Division
Armco Steel Corporation
A-S-H Pump
Auto Quip Corporation
Beebe Bros.
Black Diamond Enterprises, Inc.
CF&I Steel Corporation
Coeur d'Alenes Company
Denver Equipment Company
The Dixie Dredge Corporation
Dorr-Oliver-Long Limited
Dover Corporation, Elevator Division
Dravo Corporation

Eaton Corporation, Fluid Power Division
Elgood Mayo Corporation
Ellicott Machine Corporation
Envirex Inc.
Formsprag
Galis Manufacturing Division, FMC Corporation
Gates Rubber Company
General Electric
Georgia Iron Works Company
Goodman Equipment Corporation
Hammermills, Inc.
Herold Manufacturing Co., Inc.
Hewitt-Robins
Hitachi (Canadian) Ltd.
Honeywell, Inc.
Iowa Manufacturing Company
Jeffrey Mining Machinery Company
Joy Manufacturing Company
Kennedy Van Saun
Koppers Company, Inc., Hardinge Operation
Lee-Norse Company
Lone Star Steel Company
Long-Airdox Company
McNally Pittsburgh Manufacturing Corporation
MSI Industries
Naylor Pipe Company
Nordberg, Division of Rex Chainbelt, Inc.
Pennsylvania Crusher Corporation
Powered Equipment, Inc.
Reliance Electric Company
Serpentix General Corporation
Smith Engineering Works
The W. R. Stamler Corporation
Thomas Foundries, Inc.
Valley Steel Products Company
Warman Equipment (International), Inc.
Wemco, Division of Envirotech Corporation
Westinghouse Electric Corporation
West Virginia Armature Company, Inc.
A. R. Wilfley and Sons, Inc.
Yuba Heat Transfer Corporation

Considerable assistance and information was also received from various Federal agencies. Particular thanks are due to:

Messrs. N. J. Greenaway, U.S.B.M. and O. B. Nair, MESA, for providing coal mine accident statistics for 1972 and 1973.

Mr. G. R. Schottler, Mr. T. B. Johnson, and Mrs. J. S. Gerg of the U.S.B.M., Division of Mine Systems Engineering in Denver, for providing current U.S.B.M. mine systems information, including the UGMHS programs.

Mr. A. J. Miscoe, U.S.B.M., Pittsburgh, for providing previously unpublished hydraulic data relating to coarse coal.

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EXECUTIVE SUMMARY

Underground coal mining is the most hazardous industrial occupation in the United States. Haulage accidents in underground coal mines contribute substantially to the occupational hazards of this type of work. In an effort to improve mine safety, a completely hydraulic transport system has been designed to move coal from the mine face to the preparation plant.

The capital investment for the hydraulic haulage system is great, and the size of the mine developed for this study is large. Mine production for the conventional haulage mine is 1.1 million tons of clean coal per year, while the hydraulic haulage mine will produce over 1.6 million tons of clean coal per year. Safety statistics for a number of mines of this size are shown.

Two hydraulic systems based on commercially available equipment were designed for this study. One of these, the surge feed hydraulic system, allows the continuous miner to operate at peak efficiencies. The second system, the direct feed system, requires the continuous miner to work at a slower, more uniform rate.

Hydraulic designs are based upon the best data available, but they are believed to be extremely conservative. Pump-motor efficiencies of 45% and maximum slurry concentrations by weight of 33% were used. Higher efficiencies and slurry concentrations may be possible, but further research is required to demonstrate this.

Each of the hydraulic systems offers greater coal productivity and lower mine mouth coal costs. The surge feed system provides up to a 15% reduction in mine mouth coal costs and the direct feed system offers up to a

12% reduction. Mine mouth costs for each of the systems are summarized below.

Summary of Mine Mouth Coal Costs ($/T Clean Coal) for 300 foot Shaft Depth

Length of Main Haulage	Conventional Haulage	Hydraulic Haulage	
		Surge Feed System	Direct Feed System
1 mile	9.133	7.726	8.079
2 miles	9.185	7.801	8.151
4 miles	9.422	8.021	8.365
8 miles	9.879	8.382	8.713

Substantial cost reductions in the hydraulic systems may be possible as the result of further research. Coal slurry preparation equipment and system controls may be less complex then envisioned. The number of face haulage pumps, per section, might be reduced to one or two. The separate feeders at the secondary haulage system might be eliminated. The main haulage pipeline size and number could be optimized. And, the dewatering step could be eliminated at the preparation plant.

Hydraulic hoisting using the Hitachi system is very expensive. A cheaper system based upon staged centrifugal pumps offers a 4% cost reduction for mine mouth cost. That such a system can be made to work at greater depths needs to be demonstrated.

Ultimately, a full-scale demonstration project will be required to establish the cost effectiveness of the hydraulic transport system. This study indicates that a 15% cost reduction is possible, and, in all probability, an equivalent percentage improvement in mine safety will be achieved. This should provide the incentive for further development of a hydraulic transportation system.

1.0 DESCRIPTION OF THE PROBLEM

Underground coal mining is the most dangerous industrial occupation in the United States, according to the National Safety Council[1]. Both government and industry are striving to improve this situation, but current mining technology may not be adequate to provide an acceptably safe environment for the miner. Therefore, major efforts are being sponsored by the U. S. Bureau of Mines to develop safe mining systems.

The purpose of this study was to design a total mine hydraulic pipeline coal transportation system that would be fail-safe. The system consists of hydraulic subsystems for conveying run-of-mine coal, including normal quantities of rock, from continuous mining machines to a surface loading point or preparation plant. An additional objective of this study was to determine the economic potential of the hydraulic system by making capital, operating, and depreciation cost estimates for this system and comparing them to similar costs for conventional haulage systems.

The study was begun in June of 1973, and the first draft of the report was submitted to the Bureau in May of 1974. All costs shown in the report are based on vendor quotes effective at January 1, 1974. No effort has been made to revise the costs to a more current date because of the rapid and irregular rise in prices during 1974.

The transportation of solids by pipeline is quite old. In 1891, a United States patent was granted to Mr. W. C. Andrews for the transport of coal in water. His process was later applied in Pennsylvania for the transport of washery waste back into worked-out portions of coal mines.

A recent and well-publicized application of hydraulic coal transport described by Singhal[2], is the Mitsui-Sunagawa Colliery in Japan. In this and other Japanese mines, the flow of water into the mine may exceed the weight of coal mined by a factor of 10. Therefore, both hydraulic transport of the coal and dewatering of the mine may be accomplished by the same equipment.

Other applications of hydraulic coal transport may be found at the Michel Colliery of Kaiser Resources Limited, Sparwood, British Columbia[3], and the Robinson Run Mine of Consolidation Coal Company, near Shinnston, West Virginia[4]. In the former mine, the coal is flumed in semicircular steel flumes laid along side the entry while in the latter, the coal is pumped through a 10-inch-diameter steel pipe.

Applications of hydraulic coal transport in underground mines are also reported in eastern Europe and in China. All of the foreign applications encountered by the authors appear to be accompanied by major mine dewatering and/or hydraulic mining activities.

Detailed cost data for current hydraulic transport systems are not available. Furthermore, the problems of mining methods or mine dewatering will affect the transportation system in a variety of ways. Therefore, this study presents a detailed comparison of a conventional and a hydraulic transport system equipped mine in an effort to determine the economic impact of the transport system on the cost of coal produced.

In an effort to determine actual accident experience with hydraulic mines, several operators were contacted. Only one, Sunagawa Colliery, Japan, responded, and they reported no injuries with hydraulic coal transport systems. Figures relating to the number of man hours worked were not made available.

2.0 APPROACH TO THE PROBLEM

2.1 BACKGROUND DISCUSSION

After rock falls from roof or rib, haulage accidents are the most serious hazard in underground coal mines. Unlike rockfalls, which generally occur near the work face, haulage accidents may occur throughout the mine. Of the many fatal and the thousands of nonfatal accidents directly attributable to the materials handling function each year, the most severe accidents are those that involve personnel being struck, run over, or squeezed by moving equipment.

A survey of haulage accident statistics for 1973 was made for a group of mines similar to the one described in the study. Similarities were seam height, use of continuous miners, and annual coal production. The mines for which figures are available are shown in Table 2.1-1.

The 1973 statistics for these mines showed that for haulage accidents, the five most dangerous job classifications and number of accidents were:

Motorman	(55)
Shuttle Car Operator	(39)
Laborer	(16)
Section Foreman	(14)
Roof Bolter	(13)

On a percentage basis, these five categories account for 60% of all haulage injuries in the mines examined. Motormen and shuttle car operators alone account for 40% of all of the haulage accidents. A complete breakdown for 233 haulage accidents, including eight fatalities, is shown in Table 2.1-2.

Table 2.1-1

Underground Coal Mines Similar to the Hypothetical Hydraulic Transport Mine

Mine	Production* 1973 Million Tons	Seam* Thickness Inches	Company	State
Ellsworth No. 51	1.060	65	Bethlehem Mines Corporation	Pennsylvania
Marianna No. 58	1.384	65	Bethlehem Mines Corporation	Pennsylvania
Somerset No. 60	1.227	65	Bethlehem Mines Corporation	Pennsylvania
Montour No. 4	1.728	68	Consolidation Coal Company	Pennsylvania
Warwick	1.652	52 & 84	Duquesne Light Company	Pennsylvania
No. 1	2.353	60	Florence Mining Company (Subsidiary of North American Coal Corporation)	Pennsylvania
Jane	1.615	54	Rochester & Pittsburgh Coal Co.	Pennsylvania
Gateway	1.430	78	Gateway Coal Company	Pennsylvania
Lucerne No. 6	1.241	45 to 72	Helvetia Coal Company	Pennsylvania
Mathies	-----	--	Consolidation	Pennsylvania
Clyde	0.627	72	Republic Steel Company	Pennsylvania
Russellton	0.576	72	Republic Steel Company	Pennsylvania
Maple Creek	2.090	96	U. S. Steel Corporation	Pennsylvania
Robin Hood No. 8	0.856	72	Armco Steel Corporation	West Virginia
No.'s 13, 14 & 15	1.107	45 to 60	Badger Coal Company (Subsidiary of Pittston)	West Virginia
Ireland	2.342	66	Consolidation Coal Company	West Virginia
Shoemaker	1.681	66	Consolidation Coal Company	West Virginia
McElroy	1.427	66	Consolidation Coal Company	West Virginia
Keystone	0.971	58	Eastern Associated Coal Corp.	West Virginia
Hillsboro	1.887	66 to 72	Consolidation Coal Company	Illinois
Monterey No. 1	2.695	--	Monterey Coal Company	Illinois
Leatherwood No. 1	1.028	60	Blue Diamond Coal Company	Kentucky
Providence No. 1	1.339	60	Island Creek (West Kentucky Division)	Kentucky

*Source: Keystone Coal Industry Manual, 1974 Edition, McGraw Hill Publishing Company, New York, New York

Table 2.1-2

Haulage Accident Statistics by Job Type, Selected Mines[1], 1973

Section Workers (Face)

Job Code[1]	Job Type	Degree[2] Fatal (1)	Nonfatal Disabling (2-4)	Nondisabling (5-6)	Total
00	Not Coded	1	-	-	1
01	Belt/Conveyor Man	-	2	1	3
04	Mechanic	-	2	6	8
06	Rock Duster	-	-	2	2
07	Shotfirer	-	1	-	1
09	Supply Man	-	1	5	6
10	Timberman	-	1	2	3
11	Wireman	-	-	2	2
16	Laborer	-	5	3	8
32	Brattice Man	-	-	2	2
39	Coal Drill Operator	-	1	-	1
35	Continuous Miner Helper	-	2	-	2
36	Continuous Miner Operator	1	-	1	2
41	Jack Setter	-	-	1	1
46	Roof Bolter	-	3	10	13
49	Section Foreman	-	7	7	14
50	Shuttle Car Operator	1	16	22	39
53	Utility Man	-	1	2	3
54	Scoop Car Operator	-	1	1	2

1 Based on USBM Code of Coal Mining Occupations

2 Based on USBM Degree Code of Injury

- 1 Fatal
- 2 Permanent Total
- 3 Permanent Partial
- 4 Temporary Total
- 5 No lost time no lost workdays
- 6 No lost time with lost workdays

Table 2.1-2 (Cont)

Haulage Accident Statistics by Job Type, Selected Mines[1], 1973

General Underground (Nonface)

Job Code[1]	Job Type	Degree[2] Fatal (1)	Nonfatal Disabling (2-4)	Nondisabling (5-6)	Total
101	Belt/Conveyor Man	-	-	1	1
102	Electrician	-	1	2	3
104	Mechanic	-	2	5	7
105	Mechanic Helper	-	-	1	1
106	Rock Duster	-	2	4	6
108	Stopping Builder	-	1	-	1
109	Supply Man	-	1	3	4
110	Timberman	-	-	1	1
116	Laborer	1	8	7	16
122	Coal Dump Operator	-	-	1	1
149	Labor Foreman	-	-	1	1
154	Belt Cleaner	-	1	-	1
157	Pumper	-	-	2	2

1 Based on USBM Code of Coal Mining Occupations
2 Based on USBM Degree Code of Injury
  1 Fatal
  2 Permanent Total
  3 Permanent Partial
  4 Temporary Total
  5 No lost time no lost workdays
  6 No lost time with lost workdays

Table 2.1-2 (Cont)

Haulage Accident Statistics by Job Type, Selected Mines[1], 1973

Underground Transportation (Nonface)

Job Code[1]	Job Type	Degree[2]: Fatal (1)	Degree[2]: Nonfatal Disabling (2-4)	Degree[2]: Nondisabling (5-6)	Total
216	Trackman	-	1	1	2
262	Brakeman	-	1	3	4
269	Motorman	3	9	43	55
276	Driver	-	1	1	2
	Above Ground				
321	Hoist Operator	-	1	-	1
373	Car Dropper	-	1	-	1
	Supervisory and Staff				
418	Maintenance Foreman	1	-	3	4
462	Fire Boss	-	2	-	2
464	Inspector	-	-	2	2
	Total	8	76	149	233

1 Based on USBM Code of Coal Mining Occupations
2 Based on USBM Degree Code of Injury
- 1 Fatal
- 2 Permanent Total
- 3 Permanent Partial
- 4 Temporary Total
- 5 No lost time no lost workdays
- 6 No lost time with lost workdays

During the period 1966-1970, Barry[5] reports that shuttle car related fatalities accounted for 11.6% of all fatal accidents in underground coal mines. The proportion of nonfatal accidents was equally high. Furthermore, the trend in haulage accident frequency was upward.

Underground coal mining would be much safer if the rail, conveyor belt, and shuttle car haulage systems could be replaced by a less hazardous haulage system. If such a system were composed of coal slurry pipelines, further benefits would be gained because the spillage of coal and the generation of airborne dust would be eliminated by the enclosure of the coal in a water-filled pipe. Also, the clean-up of coal along tracks and conveyor belts would no longer be necessary, thereby eliminating exposure to accident of miners performing this clean-up.

2.2 COAL MINE LAYOUT

The first task undertaken in this study was the development of a typical coal mine layout. Many of the specifications relating to number of entries, seam thickness, entry widths, etc. were specified in the contract. Other specifications were worked out as the project progressed.

Early in the study, it became apparent that any cost comparisons for hydraulic and conventional systems would have to be made on a current and hypothetical basis. This was because there were no cost data for the proposed system, and only general costs for the conventional systems were available. Further, the rapid and steady increases in equipment prices made it desirable to price all capital equipment at the same date so that comparisons could be made.

The most meaningful comparison is the impact of hydraulic haulage on mine mouth costs. This is also the easiest comparison to develop because it is almost impossible to segregate coal haulage from other mine costs. Furthermore, variations in mine production will affect mine mouth coal costs because of the changes in investment charges per ton.

The two transport systems compared in this study are radically different, and the investment costs reflect this difference. The delivery of coal slurry to a preparation plant may increase coal preparation costs because of increased fine coal, and the coal slurry dewatering system will be required whenever the hydraulic transport system is used in conjunction with a mine which ships raw coal.

2.3 OPERATIONS RESEARCH

The hydraulic haulage system outlined in this report is a highly complex network, the successful operation of which requires close control of water and coal input, slurry velocities, and system shutdown and start-up procedures. When this system is coupled with an already complex underground continuous coal mining operation, it becomes almost impossible for an individual to visualize the operation of the entire system, let alone optimize it. For this reason, analysis techniques, including statistical analysis and computer simulation of the entire system, were attempted.

The mine simulation programs available for this study yielded disappointing results, and these were not used in the final report. Continuing work at Pennsylvania State University and elsewhere should provide simulation programs capable of analyzing the hydraulic transport system.

3.0 MINE LAYOUT

3.1 COMMON FEATURES

As mentioned earlier, the contract specifications stipulated certain conditions for both the mine layout and operation. Some of these were later modified by the authors, with the Bureau's consent. The basic criteria were:

1. The face haulage system shall be operable in a five-entry room-and-pillar panel system during development and retreat mining.

2. Seam height shall not exceed 5 ft., 6 in.

3. Entries and crosscuts shall not exceed 16 ft. in width.

4. Section tonnage from a continuous miner shall be up to 10 tons per minute.

5. Panels shall be 5,000 ft. in length (later modified to 2,500 feet by mutual agreement).

6. Operations shall be 600 shifts per year.

7. All portions of the haulage system shall be easily extendable and retractable while in operation to following mining progress.

8. All portions of the haulage system shall be capable of accepting feed from multiple input locations.

9. The final design of all portions of the haulage system shall be suitable for installation in an underground coal mine.

In addition to the specifications provided in the contract, the following assumptions have been made to aid in making a generalized study while maintaining a close relationship to actual mining conditions:

1. The coal seam is level for all practical purposes.

2. Roof and floor conditions present no unusual problems.

3. The coal weighs 85 lb/cu ft in place (1.36 sp.gr.).

4. Three inches of top or bottom are mined, with the coal, at 172 lb/cu ft (2.75 sp.gr.); haulage equipment will not require more than 5.5 feet of seam height to operate.

5. Seam depths of 300, 500, 1,000, 1,500, and 2,000 ft are considered.

6. Capital and operating costs shown are those pertaining to full capacity production of the mine at the time work has progressed to the extent shown in Figure 1.

7. All costs, including labor, are those in effect in January, 1974.

8. Mine life is assumed to be 20 years.

9. The mine will work three shifts per day, 200 days per year.

For the purpose of this report, the overall general mine layout shown in Figure 1 is the same for both conventional and hydraulic systems. The main shaft "A" is located at the center of the property with initial main entries "B" advanced in one direction from the shaft bottom to the property boundary. Submain entries "C" are turned at right angles from the return side of these main entries on approximately 5,600 foot centers and advanced to the property boundary in that direction. Mining advances on this side of the mains to the boundary, then retreats on the other side to the shaft bottom. Main and submain entries are driven in two sets of five entries each and are separated by a wide, solid pillar between the sets. Pillar widths are determined by the depth of the coal seam below the ground surface and the natural conditions of the strata. Sixty-foot by 40-foot pillars are shown in the drawings. Connections between sets of main entries are made at each submain location and halfway between submain turnoffs.

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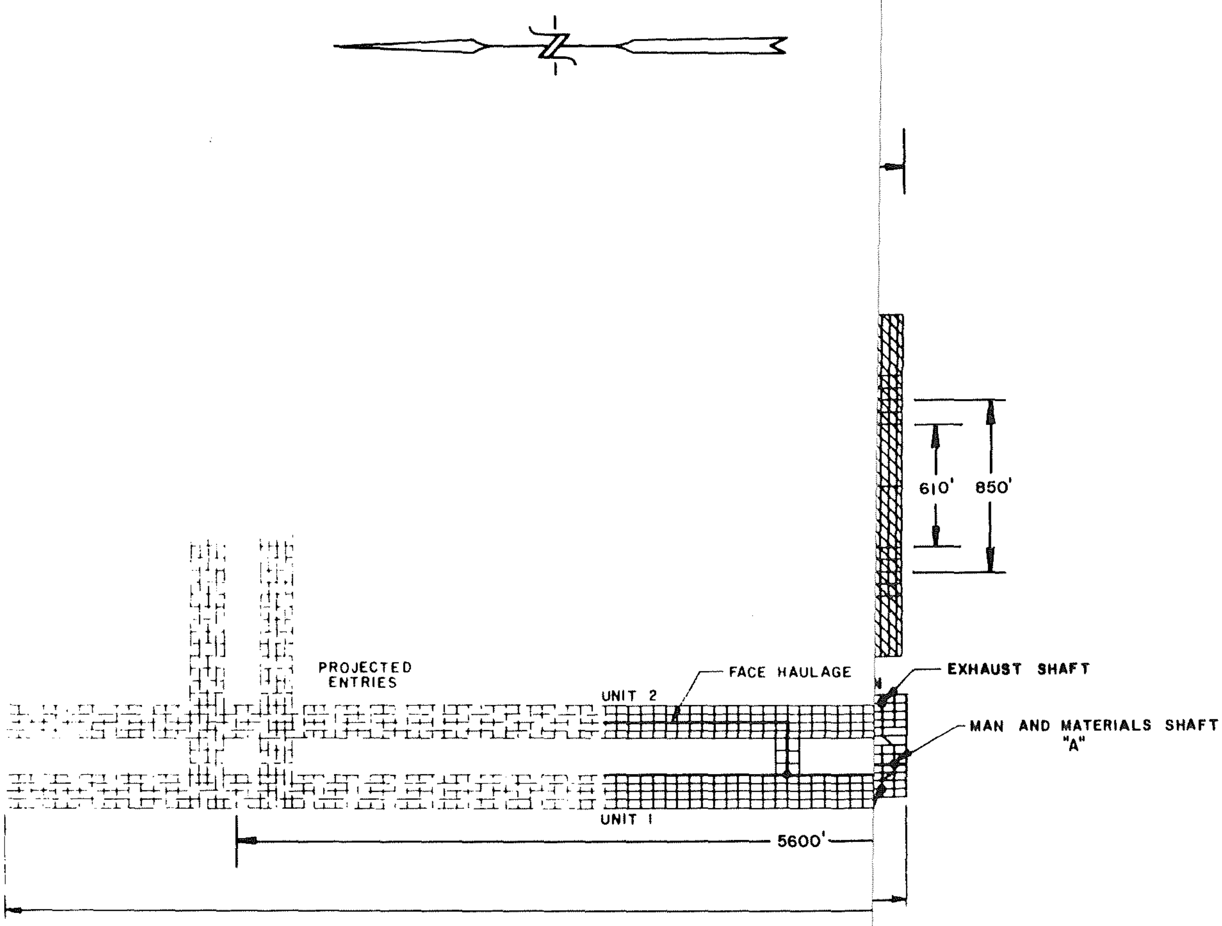


FIGURE 1-GENERAL A

Panel entries "D" in sets of five are turned at right angles from the return air side of the submain entries to outline 610-foot-wide blocks of coal 2,500 feet long. Mining of these panels advances on the return air side of the submains, to the property boundary, and retreats to the main entries on the intake side. This places all worked-out areas behind active workings on the ventilating circuit and drains gases from them directly into the return air courses. Interference with mining operations and danger to personnel is thus reduced.

Although five operating units are required to obtain production, seven are purchased and installed underground. By schedule rotation of the five operating sections, among the seven available, routine maintenance can be accomplished regularly on each unit during an idle shift. In addition, two mining units are always available for use to maintain production if one of the operating units is unable to work for any reason.

Each producing section consists of five main faces plus crosscut faces as shown in Figure 2. Each face is 16 feet wide with room and crosscuts spaced on 50-foot centers, leaving 34-foot by 34-foot pillars. The coal is cut from the seam by a continuous mining machine which moves from one face to another in sequence. Each face is mined to a depth of 20 feet in two cuts by the machine. The first cut is the full 10-foot width of the machine; the second is a six-foot-wide cut.

All mining is accomplished with continuous miners. Working in the section with the miner is a roof bolter which provides safe working places without delaying the miner. These machines drill holes in the roof on

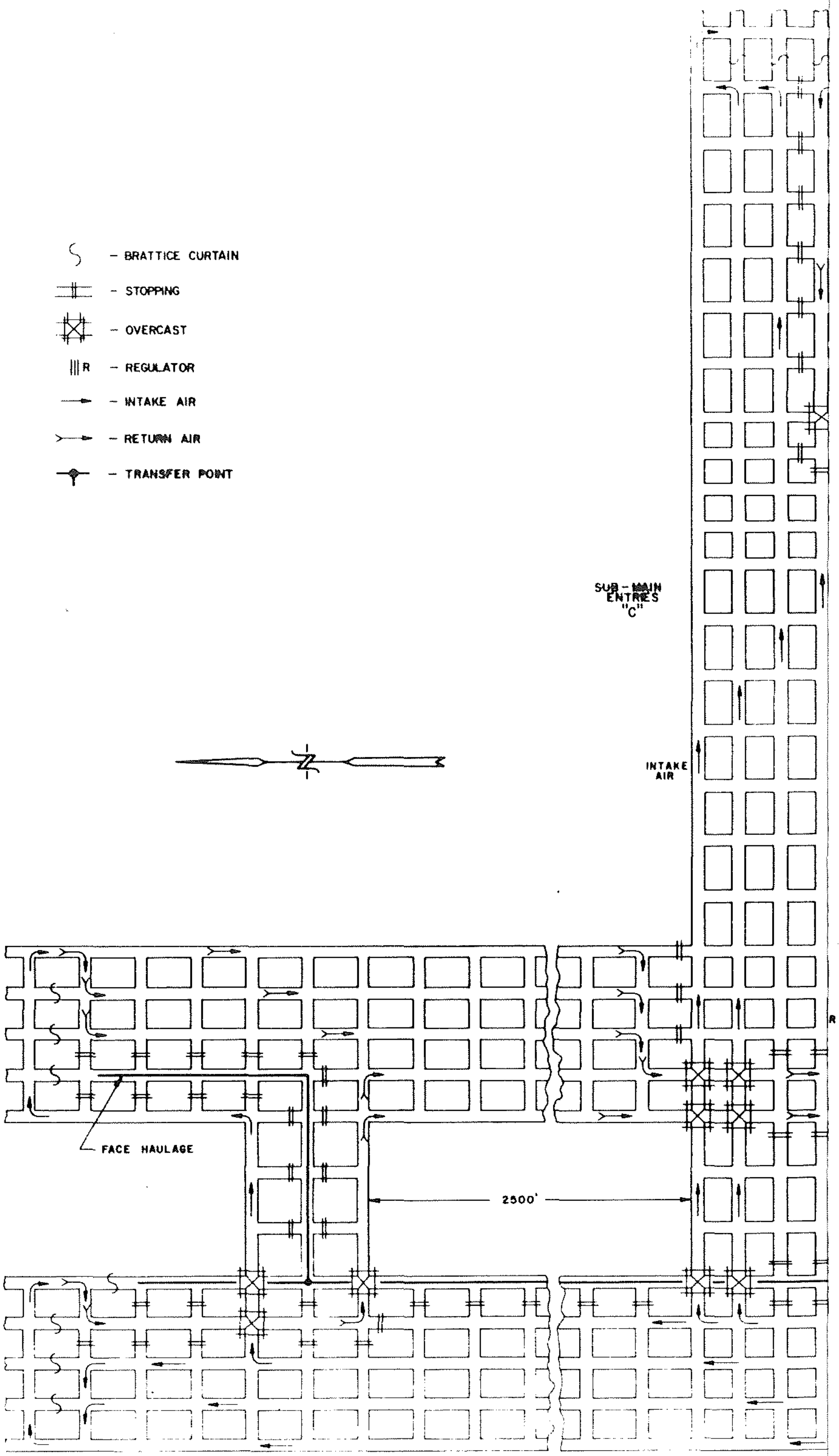


FIGURE 2 - VENTILA
COAL MI

four- or five-foot spacings, usually to a depth of four to six feet. Steel bolts with expansion shells are inserted into these holes and tightened against the roof surface for support. Fifteen to 20 such bolts are installed for each cut.

Another machine in use in the face area is the "Scooptram". This unit serves a dual purpose. The scoop is used to clean up loose coal at the face left there by the miner which is a rather poor clean-up machine. This must be done for compliance with the 1969 Federal Coal Mine Health and Safety Act. The scoop is also used to move supplies in the face area between the end of track and the point of use. These machines are usually battery-powered to eliminate the use of trailing cables.

A rock-dusting machine is also included in the face equipment in order to reduce the explosiveness of the coal dust. These machines are wheel-mounted, AC-powered, and are usually pulled by the supply unit. They generally have a storage capacity of about four tons of rock dust which they spread by blower over the roof, sides, and bottom of the mine openings.

Another piece of machinery is that used to transfer the coal from the shuttle car to the panel belt conveyor. It is called a feeder and receives the coal carried in the shuttle car at a high rate but feeds it onto the conveyor at a lower, more uniform rate to prevent spillage and overloading of the conveyor. If a breaker is incorporated into the machine to reduce oversize material, the unit is called a "feeder-breaker". These machines are AC-powered, and would reduce all material to less than nine inches for discharge onto a 36-inch-wide conveyor.

Coal is moved from the face area to the mine shaft by conveyor belt, and is then hoisted to the surface by a conventional hoisting arrangement. Men and materials are transported underground from the shaft bottom to the working areas by electric locomotive and cars on track laid parallel to the belt conveyor system. Service track for this study is 60-lb. rail in the mains and submains and 40-lb. in the panel entries. All track is laid with a 42-inch gauge. Locomotives vary from eight to 15 tons for this service, and are powered by overhead trolley wire, battery, or combinations of both. Personnel transportation is accomplished with single self-powered man-trip units capable of carrying the men required to work one section. The units are then kept on the section during the shift for use as necessary. Mechanics and supervisors move throughout the mine on smaller, self-powered personnel carriers which will carry from one to six men.

Fresh air is supplied to the underground openings and working areas by a 350,000 SCFM main ventilating fan mounted on the surface over one of two ventilating shafts and operated on a separate electric circuit from the mine circuit. In this hypothetical mine, the fan is operated as an exhaust fan pulling air into the second shaft and through the mine as shown in Figures 1 and 2. The fan capacity has been based on the assumption that seven operating sections will be ventilated and that each will need 50,000 cubic feet of air per minute at the fan, allowing for leakage losses in transit.

One of the entries contains the supply track, belt conveyor, and electric distribution equipment. It is isolated, as required by law, from the fresh air current by a line of stoppings in the adjacent crosscuts and by the solid pillar which separates the intake from the return entries.

Electric power is supplied to the underground area through bore holes located at the 7,500-kva substation on the surface. The substation consists of two 3,750-kva transformers so that one can be moved to a new location as the mine workings advance toward the property boundary. Figure 3 shows the location and type of electric equipment used underground for the conventional haulage system, while Figure 4 shows the electrical layout for hydraulic haulage.

The main distribution line carries 7,200 volts in an 8,000-volt, 250,000 CM, SHGD U.S.B.M.-approved cable suspended from the mine roof on insulators. This voltage is reduced to 440 volts for use by machines through portable 150-, 300-, 500-, or 1,000-kva transformers. These units usually have outlets for attaching the trailing cables supplying power to each machine; they are called power centers and are portable for ease in moving with the faces. Switchgear ncessary to isolate and control equipment or sections of the mine is also included in the power center in the mine distribution system.

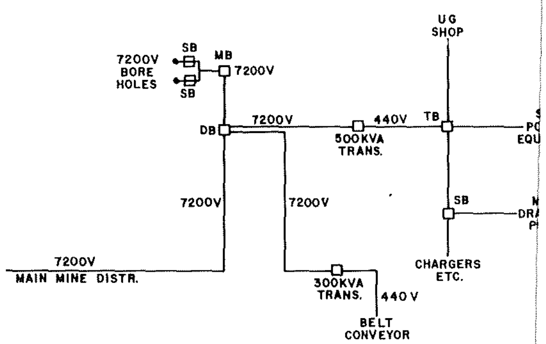
Surface facilities include the shaft head frame, hoists, building, substation, fan, preparation plant, and miscellaneous shop and service functions.

ELECTRICAL MATERIALS LIST

300KVA TRANS.--------- 10 + 1 SPARE
1000 KVA TRANS. (COMB.)---- 7
DOUBLE BREAKERS-------6 + 1 SPARE
SINGLE BREAKERS------- 6 + 1 SPARE
500KW RECTIFIER--------1
8000V MAIN BREAKER-----1
600V TRIPLE BREAKER----1
7200/440V 500KVA TRANS.--1
250000 CM ELECTRIC CABLE -11250 FEET
1/0 ELEC. CABLE 8000V--3000 X 7 PANELS=21000 FEET
TROLLEY WIRE----------11250 FEET

SHAFT BOTTOM ELECTRICS

1-MAIN BREAKER 8000V
1-TRIPLE BREAKER 600V (SHOP, PUMP, RECT., SKIP POCKET, CHARGERS)
1-500KVA TRANS. (7200/440V)
1-DOUBLE BREAKER 8000V (M7200-BOT. TRANS., ME CONV. TRANS.
1-300KVA TRANS. (7200/440V)
1-SINGLE BREAKER 440V
2-SINGLE BREAKER 8000V



——— 7200V LINE
— — — 440V LINE
□5 500KVA TRANS. 7200/440V
TB TRIPLE BREAKER
SB SINGLE BREAKER
■ 500 KW RECTIFIER

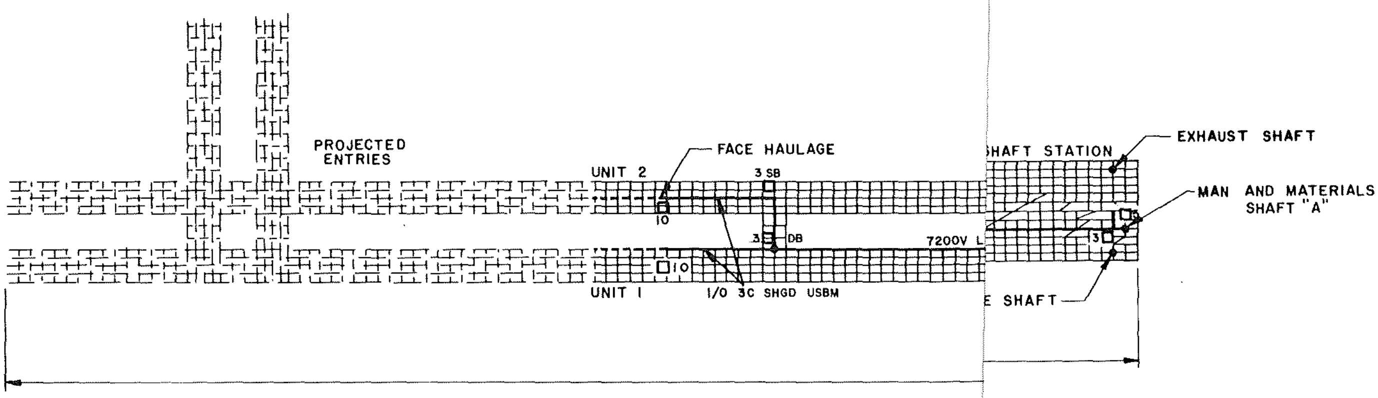


FIGURE 3— CONVENTI
PILLAR C

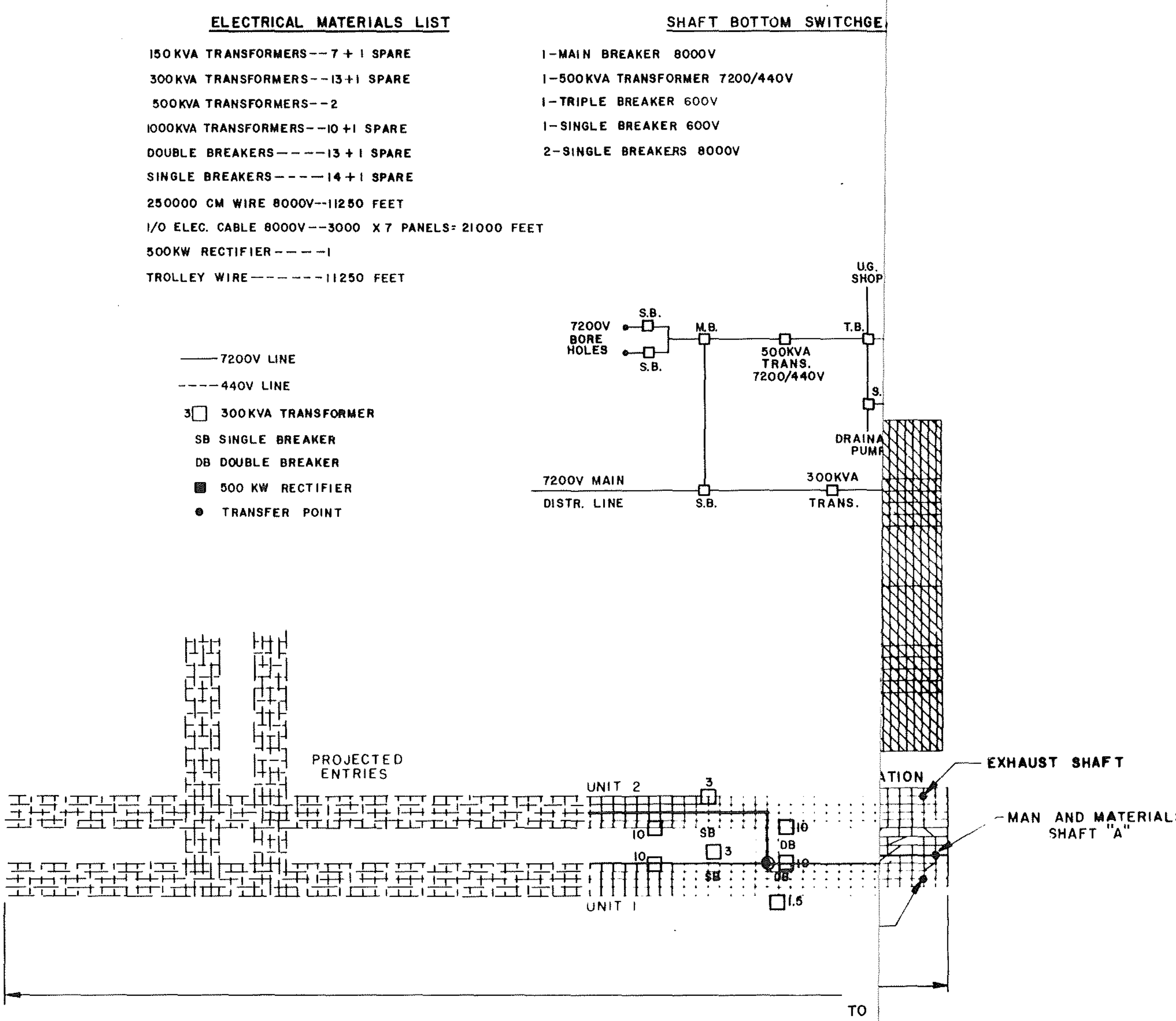


FIGURE 4 – HYDRAULIC
PILLAR COA

3.2 DESCRIPTION OF CONVENTIONAL HAULAGE SYSTEM

Both development and room work is accomplished by continuous mining machines at the face using shuttle car haulage into belt conveyor feeders which discharge onto the panel entry conveyor system. In the simulated mine of this study, panel entry conveyors discharge onto sub-main entry conveyors, which, in turn, feed the main entry conveyor system. It is thus an all-conveyor haulage system, but rail haulage could be substituted at any of the transfer points given above.

Panel entry or secondary haulage generally consists of a 36-inch wide belt conveyor of wire rope suspension construction which discharges onto the submain conveyor or other haulage system. Maximum length of this conveyor in the mine layout described is 2,800 feet. It is extended in increments of 150 to 250 feet as the entries are advanced, and correspondingly shortened as work retreats to the submain entries. Although 100-hp motors are sufficient to drive these units, some mines use 200-hp motors to allow interchangeability with main haulage conveyors and reduce spares inventories. Some mines also use 42-inch conveyor belting and structure for the same reason.

Submain and main entry haulage systems are essentially the same. The former may use 42-inch wire rope suspended belt conveyors, while the latter may use 48- or, in some cases, 60-inch conveyors. The simulated mine uses 48-inch units in both mains and submains. The length of each conveyor will be in the vicinity of 4,000 feet, with units added as necessary as the entries are advanced to the property lines. Drives for these conveyors are 200 hp with provision for use of a second 200-hp motor (dual drive) if needed.

Under existing Federal and State laws, no coal dust or foreign material is permitted to accumulate underneath any portion of the conveyors. The conveyor must have a start-stop control throughout its entire length so that it can be stopped from any point, and it must have automatic fire sensing and suppression systems along its length. The first stipulation requires frequent and continual inspection and clean-up of the entire conveyor by personnel assigned to this work. The other two require equipment expenditures and maintenance. Complete screening or guarding of the head and tail mechanism is also required.

If a slope is used to open the mine, belt conveyor haulage will probably be used to convey the coal to the surface. If a shaft is used, the coal will probably be hoisted in skips using either friction or drum-type hoists with manual, semiautomatic, or fully automatic control, depending on conditions. Surface transportation from the shaft collar to the washing plant or storage area is also accomplished by belt conveyor.

Whether hoisting is accomplished by conveyor or skips, provision must be made for access to the underground area by men and materials. In either case, a hoist with man-trip or supply cars is required, and, in most instances, a hoistman must be in attendance on the surface.

Overland transportation from the mine to the preparation plant would be accomplished by belt conveyors similar to those used in the main haulage system. The belts might require rigid steel frames and would probably be covered to prevent wetting and freezing of the coal in inclement weather. Coal from the overland belt system would be stored in a 3,000-ton raw coal storage silo. From here, the coal would be fed directly into the preparation plant at the rate of 350 tons per hour.

3.3 DESCRIPTION OF HYDRAULIC HAULAGE SYSTEM

In addition to the stipulations common to both the conventional and hydraulic transportation mines, the contract added the following requirements for the hydraulic transport system mine:

1. The hydraulic haulage system shall meet the current Federal mining regulations.

2. The system shall be "fail-safe", that is, the failure of any part must occur in such a manner to prevent hazard to underground personnel and operations and to permit easy restarting when service is restored.

3. Automatic control of solids concentration at feed points and throughout the pipeline system is required.

The mine layouts for the hydraulic haulage system are similar to those for the conventional haulage system. In the study, two face haulage systems were devised. One of these utilizes a shuttle car and a feeder-breaker directly behind the miner. Coal is discharged from the miner to the shuttle car and into the feeder-breaker. This allows the hydraulic system to operate at a more uniform rate as coal surges from the miner are stored in the shuttle car and feeder-breaker. Discharge from the feeder-breaker will be reduced to two-inch maximum size and fed at a controlled rate of 4.6 tons per minute into the hopper of the pump feed system. This system supplies a coal and water mixture to a 200-hp, 10-inch slurry pump which has two suction intakes. Coal and water enter one opening, and fresh water enters the other at a regulated rate to make a 33% solids mixture by weight. In this report, the miner-shuttle car/feeder-breaker system is called the "surge feed" hydraulic system. The hopper, pump feed, and pump are called the "slurry module".

The fresh water line and the slurry discharge line are each made up of a 100-foot length of flexible 10-inch hose coiled in the opening behind the pump and connected to a standard 10-inch steel pipe leading to the secondary haulage system. This arrangement permits 46 feet of forward movement of the machines for mining and enables work in crosscuts at any angle.

The slurry and fresh water hoses are clamped together and suspended, near the floor, from a monorail in the center of the entry. The hoses are fitted with metal guides at bend points to prevent pinching. The metal guides also have shoes to prevent the hose from abrading when it is dragged across the floor.

At the completion of each cut, the miner trams to a new work place. The shuttle car and feeder-breaker follow the miner as soon as they are empty of coal. The feeder-breaker couples to the idle slurry module in the entry serving the new work place and the cutting cycle resumes.

Meanwhile, the slurry module at the work place just vacated by the miner has shut down. The outbye ends of the hoses are disconnected and they are pulled forward on the overhead monorail into their coiled position by a small hoist mounted on the module structure. Following this operation, new sections of steel pipe are inserted using the Scooptram and the hoses are reconnected for the next cycle. Hoses, pumps, and pipelines are installed in four of the five openings of each mining section; the center one is free of this equipment to permit unrestricted travel of supply and other machines. Actual operations could probably be conducted just as easily with pipe and hoses laid only in entries 2 and 4. The miner could reach entries 1, 3, and 5 with the 46-foot extension of the flexible hoses which would enable the entries to be advanced from the adjoining side openings.

A second system has been devised to provide hydraulic transportation directly from the miner with no intermediate equipment, and it is called the "direct feed system". Although this system limits miner output to 4.6 tons per minute, the increased operating time and shorter moving time between faces raises overall shift production over conventional haulage systems. This system will require the use of a coal breaker built into the conveying system of the continuous miner to size the coal for pipeline transportation. Construction of the breaker as an integral part of the miner has been assumed to be practical. Coal from the miner will be discharged directly into the one- or two-ton capacity hopper of the pump feed system described earlier, at which point hydraulic transportation becomes identical to that of the other system, although a longer hose may be required.

The hydraulic secondary haulage subsystem is designed to receive coal from a single face haulage subsystem and the hydraulics are similar. This system is much less complex because it is moved only when not pumping slurry. The slurry pump will be moved as necessary into or out of the panel entry as the working faces advance or retreat. The system would be shut down and pipe additions or deletions would be made as needed on an idle shift.

The slurry preparation and feed system developed for the face haulage subsystem would be used whereever introduction of coal from a conventional haulage subsystem, such as a belt or a shuttle car, is necessary. A slurry module could be positioned anywhere in the hydraulic transport system, and is incorporated in the design at each pumping station. By positioning a feeder-breaker to introduce coal into the module, the conventional and hydraulic systems may be readily joined. When coal slurry is being received from a hydraulic transport subsystem, it would be introduced directly into the feed hopper of the slurry module.

The main haulage subsystem is the most complex of all of the hydraulic subsystems. It must receive coal from one to five producing sections at any time and in any combination. At the same time, this material must be transported in the system at or near the maximum gravimetric concentration to maintain reasonable efficiency.

In a slurry transport system, slurry velocity is generally controlled in a very narrow range. Velocity must not fall below that needed to suspend the solids or the pipe will plug. As velocity is increased above that needed to suspend the solids, power requirements increase. Therefore, the hydraulic main haulage system was designed to operate as efficiently as possible with the minimum number of pipes. The system selected utilizes two pipelines, one of which is capable of carrying the production of one or two sections, while the other can carry production from two or three sections. This allows efficient operation with automatic valving in the subsystem directing the material into the appropriate lines.

A 14-inch- and a 16-inch-diameter pipeline will provide the most suitable combination for the entire main haulage system. The pumps are sized accordingly. Pump station spacing should be determined by the distance between submains along the main entry.

Pump spacing in the working area of the submain entries (Figure 2) is governed by the distance between the haulage pipelines of adjacent panel entries. In the mine layouts, this distance would be 850 feet.

Each pumping station will be a transfer point for material in transit, and will constitute the end of the inbye pipeline. The free, open-end flow from this line will be directed into a hopper which will feed the succeeding set of pumps by gravity. This will minimize the effect of water hammer in the slurry lines and permit transfer of slurry between pipelines as necessary.

A hydraulically more efficient system could be constructed utilizing varispeed drives on the pumps and linking the pipeline directly to the suction of the next pump downstream[6]. This would increase the efficiency of the system and would be easier to operate, but it would prevent the introduction of coal into the system as specified by the Bureau contract for this study.

Use of the Hitachi Hydrohoist is contemplated for hoisting the material to the surface. This device is commercially available from a Japanese manufacturer. It has been used in Japan for several years in applications similar to the one contemplated in this study, and may be applied to North American mines in the near future.

In this study, coal and water are discharged from the main entry pipeline into the 460-ton-capacity shaft pocket. This pocket is constructed to intersect an opening, containing the hydrohoist, beneath the coal seam. The intersection must be made water-tight with a chute on the lower end connected to the feed hopper of the hydrohoist. Two dams must be constructed across the entry at the top of the pocket to prevent water from flowing back into the mine.

Coarse coal will settle to the bottom of the pocket with some water and constitutes the feed to the hoist. The excess water, partially clarified, will be taken off at the top of the dams and pumped through a 20-inch pipe to a holding tank where pumps will recirculate it back into the mine. Provision for periodic cleaning of this tank and pumping of the fine material to the surface for reclamation must be provided.

Slurry makeup water is available from a 20-inch waterline located in the main and submain entries. A 10-inch waterline provides water to the panel entries and face haulage systems. A pump station at the shaft transfers the makeup water from the tank into the waterline.

Bypass lines connect the slurry lines to the makeup waterline at each pump. Valving arrangements in this line permit water in the slurry lines, when there is no coal, to be pumped directly into the makeup waterline, thus maintaining efficient concentrations of materials and eliminating the cost of pumping this water all the way to the shaft bottom and back. The process is controlled by automatic valve and switch operations.

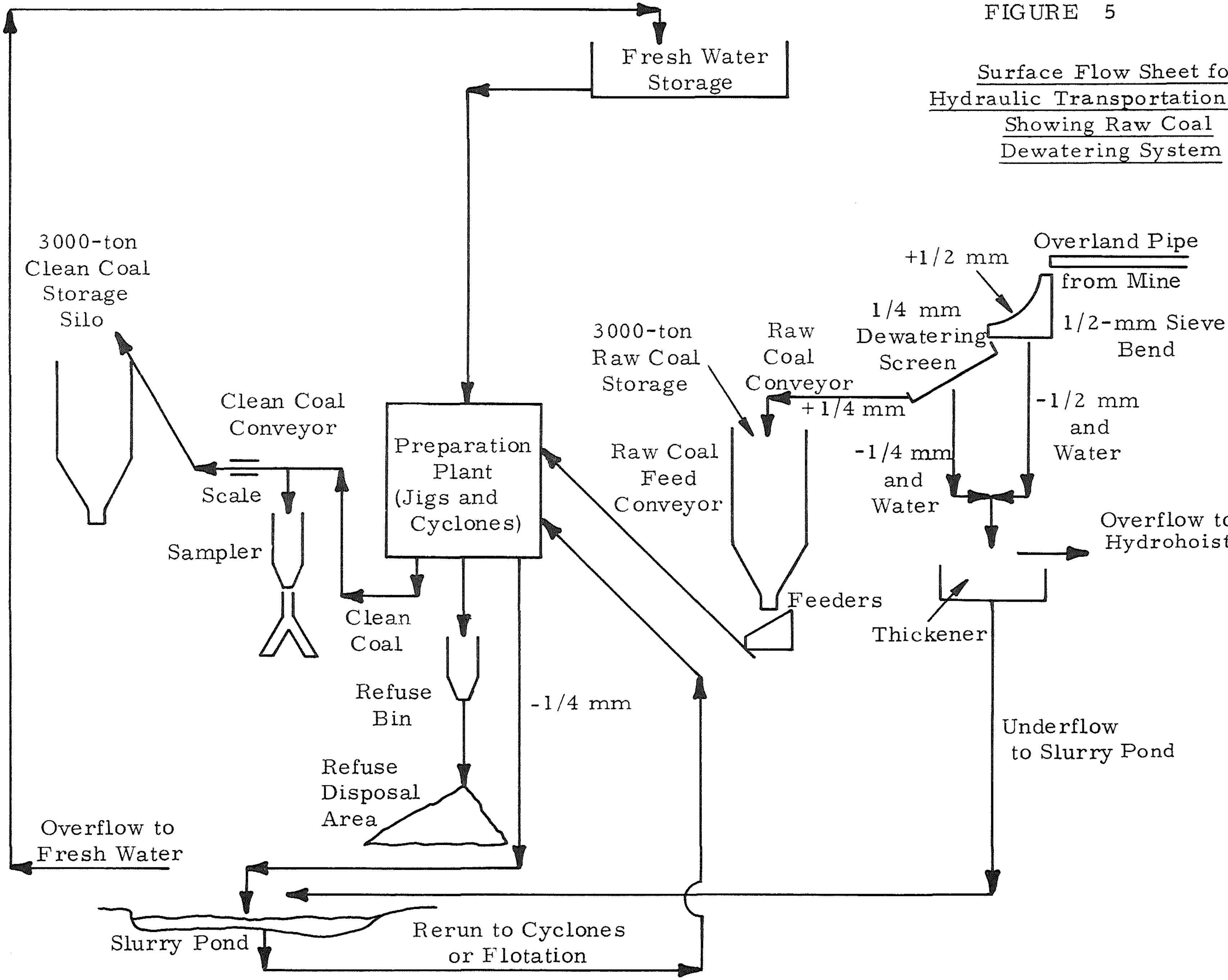
The hydrohoist is designed to deliver the slurry directly to the surface coal preparation plant, shown in Figure 5, through 3,000 feet of horizontal surface pipelines. Discharge of the overland pipe will be directed over four-foot (DSM sieve bend) screens and then onto a bank of four-foot by eight-foot dewatering screens to remove the water. One-half-millimeter slot openings in the DSM screens and one-quarter-millimeter slots in the vibrating screens will remove -60 mesh material with the water. Subsequently, the underflow will go to a 60-foot-diameter thickener tank from which the overflow will be returned underground via the hydrohoist and the underflow will go to the slurry pond. This dewatering step could probably be eliminated by appropriate preparation plant design, but it is included here to allow comparison with conventional mines.

Dewatered coal from the screens will be accumulated on a 36-inch belt conveyor discharging into a 3,000-ton raw coal storage silo, similar to the one for the conventional haulage mine. From here, a second similar conveyor will feed it to the preparation plant at the rate of 400 tons per hour.

The preparation plant has been assumed to contain the necessary jigs and cyclones to prepare raw coal with an overall reject of 15%. Minus 60-mesh material will go to the slurry pond and be periodically recirculated through the fine coal circuits of the plant. Coarse refuse will be collected in a 50-ton bin and hauled to a disposal area by truck or pipeline. Clean coal will be removed on a 36-inch belt conveyor along which will be an automatic sampler and a belt scale to determine the quality and quantity of the salable product. Clean coal will be stored in a 3,000-ton silo capable of loading the required transportation facilities.

FIGURE 5

Surface Flow Sheet for Hydraulic Transportation Mine Showing Raw Coal Dewatering System



Water for the preparation plant, surface use, and underground transportation, etc. will be obtained from a surface runoff impounding dam constructed near the mine site. Clarified water from the slurry pond will be recirculated to this dam or to the preparation plant.

4.0 CONVENTIONAL HAULAGE SYSTEM

4.1 COAL PRODUCTION RATE

One of the first tasks undertaken in the haulage system study was the determination of probable mining rates by continuous miners under the conditions postulated for the study. Cutting rates of 10 to 14 tons per minute are reported for continuous miners, but average daily production rates are from one to two tons per minute.

In order to develop a realistic cycle time and cutting rate for the mining machine, a number of coal mine operators were contacted. Six companies responded with substantial amounts of time study data for continuous miners. Additional time study data were obtained from Gates Engineers, a subcontractor, and various other manufacturers. These time studies were used to develop the average work cycle for each miner, shown in Table 4.1-1.

Table 4.1-1

Average Miner Cycle Time/Work Place

Miner Operating Time/Place	25.5 Minutes
Waiting Time/Place	17.7 Minutes
Moving Time/Place	7.9 Minutes
Other Delays/Place	27.3 Minutes
Total Time/Place	78.4 Minutes

The miner operating time includes all time when the drum is rotating. Waiting time is generally the time associated with shuttle car delays. Moving time includes both the maneuvering required to make the second cut and the time required to tram the miner to a new work place. Other delays include safety delays, ventilation, ground support, minor maintenance, operator travel time, lunch time, and other needs.

Discussions with manufacturers and coal mine operators revealed that machines are now available for driving entries 16 feet wide in one pass. However, the industry prefers the smaller machines which require two passes to create the 16-foot-wide entry. This preference appears to be based on greater familiarity with the smaller machines by operating companies, coupled with apparently higher maintenance costs for the larger machines.

Reference to specific brand names in the following discussions is made for identification only, and does not imply endorsement by either the Colorado School of Mines Research Institute or the Bureau of Mines. The mine layouts developed for this project are based on a Joy 12 CM or equivalent miner. On the first pass, the miner will make an opening 10 feet wide and 20 feet deep. The machine will then back out and make a second pass to widen the room to 16 feet.

A total of 81.7 tons of raw coal will be produced from each work place in the 78.4 minutes. Assuming an eight-hour work shift with 65 minutes of portal-to-portal travel, each miner will produce 433 tons per shift. This is very close to the production rates reported by the industry as a whole.

4.2 FACE HAULAGE SUBSYSTEM

The broken coal is picked up by the gathering arms of the miner and loaded into Joy 10 SC shuttle cars which perform the face haulage function. In the height of seam (5.5 feet) specified for this mine, each car has a rated capacity of nine tons of coal. When the car is filled, it carries the coal to the belt conveyor loading station, a distance which normally varies between 100 and 550 feet. As the first car leaves the miner and clears the area, a second one moves in to be loaded. This delay consumes one-half minute to more than one minute when the empty car is standing by. When no car is available, the delay may be much longer.

4.3 SECONDARY HAULAGE SUBSYSTEM

The secondary haulage subsystem is composed of a 36-inch-wide belt conveyor and the associated equipment located in the middle panel entry. For the purpose of this study, the belt has a maximum length of 2,800 feet. Coal is loaded onto the inbye end through a Stammler Model BF-14 feeder-breaker which controls the feed rate and maximum size of the material loaded onto the belt. Overloading of the conveyor by surges of coal from the shuttle cars is thus minimized, as is belt damage and down-time caused by oversized material. A 36-inch conveyor moving at 400 to 500 feet per minute is adequate for this application, although it is not uncommon for a 42-inch structure and, sometimes, belting to be used in order to standardize belts throughout the mine. In either case, 100-hp drives are adequate for the panel belts.

Each drive unit is installed at the junction of the panel entry and the submain entry. A 300-kva, 7200/440-v transformer and 8000-v double-circuit breaker is installed at the drive unit. One branch of the breaker feeds the conveyor drive, and the other supplies the face equipment by means of 1/0, 3 conductor, electrical cable suspended in the panel entry.

Substantial upgrading of safety standards for conveyor installations were mandated by the 1969 Federal Coal Mine Health and Safety Act. Mines are now required to install waterlines parallel to the conveyors and to provide fire hoses at 300-foot intervals. Automatic fire sensing devices and deluge-type water sprays or acceptable substitutes are required at the conveyor drive ends. These devices must be capable of automatically detecting and extinguishing fires at the drive and the adjacent 50 to 150 feet of conveyor.

Belts must be equipped with automatic belt motion and sequence controlling switches. These will automatically shut down belts when an outbye unit slips or stops, thus preventing coal spills at transfer points.

All belts must be equipped with emergency shutdown switches. These are actuated by pull cords strung along the conveyor support ropes and may be actuated from any point along the belt.

4.4 MAIN HAULAGE SUBSYSTEM

The main haulage subsystem is located in the submain and main entries. Coal from the panel belts discharges onto the submain haulage belts by means of simple transfer points where the belts meet. The submain belts then discharge onto the main entry belts in a similar fashion.

The submain entry belts may be equipped with 42-inch belts, while the main entries are often equipped with 48- or 60-inch-wide belts. This facilitates the movement of the ever-increasing volume of coal nearer the shaft. However, in the mine design for this study, 48-inch belts were selected for both segments of the main haulage subsystem.

Each conveyor will be 4,000 feet long with new units added as entries are extended. These units are recovered as mining retreats from a set of submains and are then available for installation in a new area of the mine. Rope belt supports are used to facilitate installation and removal. Conveyor drives for the hypothetical mine are 200 hp, with the provision for adding another 200-hp motor if the dual drive is needed.

As with the panel belts, all Bureau of Mines safety requirements are met. These requirements also include a continual clean-up effort to prevent coal and dust buildup at transfer points or along the belts.

4.5 HOISTING SUBSYSTEM

In shallow coal seams, adits or slope entries are frequently employed. If a belt conveyor is used in these entries for hoisting coal to the surface, it should be at least the same size as the main haulage conveyor underground. The drive horsepower requirements will be greater due to the elevating factor. It is often possible to discharge the hoisting conveyor directly into the raw coal storage facility on the surface.

For the purpose of this study, shaft hoisting is assumed. Modern shaft hoisting employs skips, usually two in balance, operating from shaft loading pockets. The size and capacity of these skips is calculated from the shaft size and depth, mine production, and hoist characteristics. Fully automatic systems eliminate the need for a hoistman in constant attendance at the controls.

Figure 6 shows the shaft bottom layout for the conventional belt haulage system developed in this study. Coal from the main haulage belt discharges into a 360-ton loading pocket detailed in Section A-A. From the loading pocket the coal is transferred into skips for hoisting to the surface. The coal hoisting operation will be fully automatic through a two-compartment, 16-foot-diameter circular shaft.

The contract calls for the evaluation of hoisting from a number of depths. The Nordberg Division of the Rex ChainBelt Company provided the data for hoists of their manufacture shown in Table 4.5-1. It is of interest to note that the production hoist changes from a drum-type to a friction, or Koepe, type at shaft depths of 1,000 feet or more.

A service hoist is also included in the hoisting installation. This hoist is used to transport men and materials in a separate compartment of the main shaft. Details for this hoist are also shown in Table 4.5-1.

In addition to the hoists described above, a small emergency hoist is included in the installation. This hoist is installed in the ventilation shaft to hoist personnel in the event the main shaft is disabled.

In order to facilitate compliance with the Federal Coal Mine ventilation requirements, 12-foot-diameter intake and exhaust shafts have been provided in addition to the hoisting shaft. In order to reduce air velocity and fan horsepower, two intake and two return shafts will be necessary for depths greater than 1,000 feet. They are not shown near the main shaft bottom because they may be drilled away from the main shaft after mine production reaches full capacity and increased fan capacity is required.

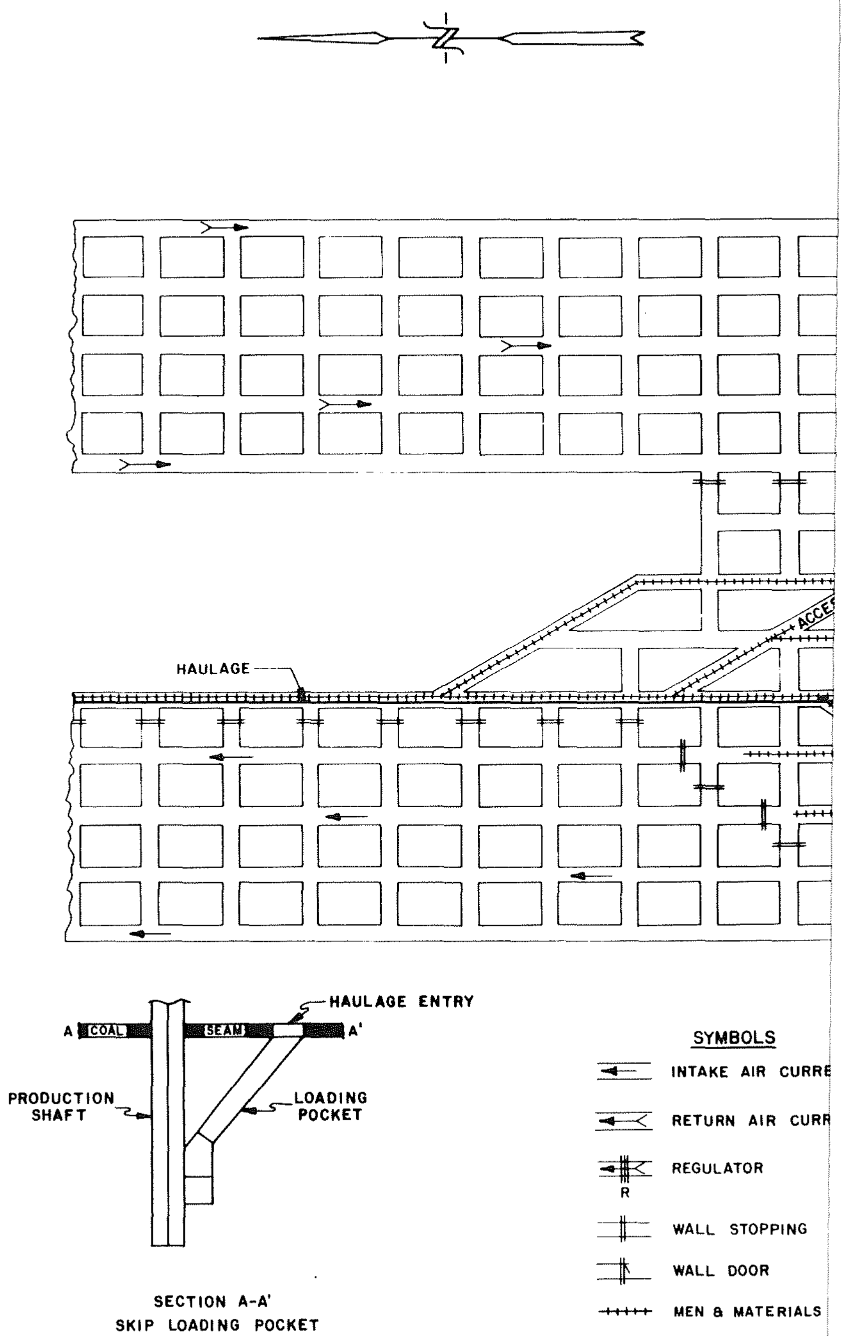


FIGURE 6 - SHAFT BOTTOM LAYO

TABLE 4.5-1

Production Hoist Data Based on 300 tph from a Vertical Shaft

Type Hoist	Depth ft	Drum Diameter in.	Motor hp	Skip Load tons	Velocity fps	Skip Weight tons	Cost[1,2,3] thousands of dollars
Single Drum	300	90	300	3.21	19.53	2.41	241
Single Drum	500	100	300	4.24	17.54	3.18	261
Single Drum	1,000	100	600	5.41	25.25	4.05	352
Koepe	1,000	70	700	5.03	29.94	7.88	266
Koepe	1,500	80	900	6.61	28.34	9.04	331.5
Koepe	2,000	80	1,000	8.00	28.82	9.45	341

Single Drum Service Hoist Data Based on 15-ton Man and Material Maximum Load from a Vertical Shaft

Depth ft	Drum Diameter in.	Motor hp	Velocity fps	Skip Weight tons	Counter Weight tons	Cost[1,2,4] thousands of dollars
300	180	700	10	11.25	18.75	491
500	170	700	10	11.25	18.75	466
1,000	160	700	10	11.25	18.75	456
1,500	170	700	10	11.25	18.75	476
2,000	160	700	10	11.25	18.75	466

[1] The indicated costs include mechanics and electrics as of January, 1974.

[2] Add $50,000 for installation.

[3] Add $3,000 for rope on the skip hoist.

[4] Add $2,000 for rope on the service hoist.

Underground mine openings in the shaft area are designed so that a sufficient number of them can be isolated from the intake and return airflow to permit the haulage and loading of coal at the bottom of the shaft, supply loading and storage, personnel movement, construction and operation of an underground shop, and installation of electrical switchgear in neutral air. Several smaller diameter bore holes connect the shaft bottom with the surface. These allow access for power cables, and lubricating oil and rock dust in bulk quantities.

4.6 SURFACE FACILITIES

4.6.1 Surface Transport System

Coal from the hoisting system is dumped by the skips into a 50-ton hopper located in the head frame. Coal from this hopper is discharged onto a 48-inch conveyor belt for transfer to the preparation plant 3,000 feet away.

The conveyor belt is constructed on steel framework and elevated at least 20 feet off the ground to facilitate the passage of mobil equipment. The belt is also covered by a light steel sectional cover to keep moisture off the coal. Walkways and appropriate safe guards for maintenance and service employees are also included.

4.6.2 Raw Coal Storage for Preparation Plant

The belt rises about 100 feet near its terminus and discharges into the top of a 40-foot by 100-foot concrete raw coal storage silo. Coal from this bin is then fed into the preparation plant.

A detailed design for the coal preparation plant was not prepared. However, McNally Pittsburgh reported that a 300-ton-per-hour plant constructed in late 1973 was erected for a total cost of $1,585,000.

4.7 REJECTED CONCEPTS

Conveyor haulage is often not recommended for distances of more than a few miles. However, the contract required comparison of conventional and hydraulic main haulage systems at distances of one, two, four, and eight miles.

In the early phases of the work, a mixed rail-conveyor haulage system was considered in order to provide a more authentic transport system. However, the cost variations introduced by this approach tended to obscure the objective of this study which was to compare a conventional haulage system with a hydraulic system. Inasmuch as the belt system appeared to offer the most direct comparison with the hydraulic system, it was selected. For those interested in other comparisons, a recent article by Laird[7] contrasts rail and conveyor haulage costs. Several other studies of haulage systems have been published and will be found in the literature[8,9,10].

5.0 HYDRAULIC HAULAGE SYSTEM

5.1 COAL PRODUCTION RATE

Early discussions relating to the hydraulic transport concept resulted in general agreement that any system to transport the coal should not require the mining machine to operate at less than full capacity. A continuous haulage system based upon a reduced cutting rate for the miner would probably not be accepted by the industry. Therefore, it became necessary to develop a theoretical maximum sustained cutting rate for a continuous miner operating under the mining conditions postulated.

In order to develop such a cutting rate, the available time study data were examined. These data provided considerable information regarding delays, but did not provide adequate information relating to the cutting process.

Subsequent discussions with manufacturers and coal mine operators provided additional information which made the estimation of cutting cycle times possible. According to those persons contacted, the cutting operation consists of three distinct steps. In the first step, the rotating cutter head is forced into the coal seam by tramming the machine forward. This operation, called sumping, is considerably slower than the straight cutting, or shearing, operation because, in sumping, the cutting head is forced into the coal by tramming the miner forward. Upon completion of the sumping cut, the drum is lowered with the miner in a stationary position. Coal cutting during this step is at or near the maximum machine cutting rate. When the cutting head reaches the floor, the cutting operation is complete and the third step is the raising of the head prior to repeating the cutting operation.

The sustained cutting rate cycle developed for this study is shown in Table 5.1-1. This cycle is based on an assumed cutting rate during sumping of about 60% of the theoretical maximum machine cutting rate.

Table 5.1-1

Continuous Miner Sustained Cutting Rate

Operation	Time (min)	Cutting Rate (tons/min)	Volume Cut (ft^3)	Production (tons)
Sump in	0.3	6.3	2 x 2 x 10	1.9
Shear	0.3	10.7	3.75 x 2 x 10	3.2
Raise Head	0.1	0	0	0
Total	0.7	7.3	5.75 x 2 x 10	5.1

Development of the total cycle time per work place was based upon time study data summarized in Table 4.1-1 and the cutting cycle data shown in Table 5.1-1. Assuming 81.7 tons of coal will be cut, at a rate of 7.3 tons per minute, 11.2 minutes of cutting time will be required. Since shuttle car delays are eliminated, waiting time has been reduced from 17.7 to 5.5 minutes per work place. The 5.5 minutes should be adequate for safety and other related delays such as maneuvering to maintain direction. Moving time has been increased from 7.9 minutes to 18.0 minutes for the hydraulic system. This additional time is required to move the equipment from one work place to another. This has been assumed to be an average moving time, as many of the moves will not require the machine to leave the entry in which it is working. Other delay time includes three minutes for the miner to back out of the cut and position itself for the second pass. Other delays would also include cable movements, minor mechanical problems, and so forth. The revised cycle time for the hydraulic surge feed system is summarized in Table 5.1-2.

Table 5.1-2

Continuous Miner Time per Work Place
Hydraulic Surge Feed System

Operation	Time (min)
Miner Cutting Time/Place	11.2
Waiting Time/Place (Safety)	5.5
Moving Time/Place	18.0
Other Delays/Place	17.0
Total	51.7

On the basis of the work place cycle time shown in Table 5.1-2, the daily unit production rate was computed. As with the conventional system, 81.7 tons of raw coal would be produced from each work place in the 51.7 minutes. Assuming an eight-hour work day with 65 minutes of portal-to-portal travel, each miner could produce 656 tons of coal with a continuous transport system. This is a substantial increase over the conventional haulage system.

A similar rationale was used to develop a cycle time for the direct feed hydraulic system shown in Table 5.1-3. The cutting rate was reduced to 4.6 tons per minute maximum with a resultant increase in cutting time from 11.2 minutes to 17.8 minutes. Other portions of the cycle remained the same as for the surge feed system even though some reduction in moving time might be expected. This system would produce 582 tons per continuous miner per shift.

Table 5.1-3

Continuous Miner Time per Work Place
Direct Feed System

Operation	Time (min)
Miner Cutting Time/Place	17.8
Waiting Time/Place (Safety)	5.5
Moving Time/Place	18.0
Other Delays/Place	17.0
Total	58.3

5.2 HYDRAULIC FACE HAULAGE SUBSYSTEM

The hydraulic face haulage subsystem receives coal from the discharge boom of the continuous miner and transfers it hydraulically to a hydraulic secondary haulage subsystem. The face haulage subsystem replaces the conventional shuttle car haulage system in the normal mining cycle, including the transport of mine refuse and minor roof falls. It is capable of transporting coal up to 1,300 feet, but it will probably move the coal only a few hundred feet under most operating conditions.

5.2.1 Coal Particle Size and Reduction

As with conveyor belt systems, coal and rock particles must not exceed some maximum dimension which is generally one-third to one-fourth of the belt width or pipe diameter. However, the pipe diameter required for hydraulic haulage will be much smaller than the 36-inch conveyor width used in the conventional haulage system. Therefore, the degree of size reduction required for hydraulic transport may be assumed to be greater.

In order to determine the sizes of material to be expected from a continuous miner, several coal mine operators were contacted. Data received from these and other sources were tabulated and are summarized in Table 5.2-1.

Table 5.2-1

Typical Coal Size Distributions from Continuous Miners

Coal Size	A	B	C	D	E
Plus 2 in.	14.4	9.9	8.7	11.7	22.0
2 in. x 1 1/4 in.	7.8	8.4	9.8	8.1	22.0
1 1/4 in. x 3/4 in.	12.7	12.5	17.6	12.1	
3/4 in. x 1/4 in.	29.8	22.6	35.7	30.0	20.0
1/4 in. x 20M	29.8	38.5	21.8	28.8	28.5
28M x 0	5.5	8.1	6.4	9.3	7.5

The data shown in Table 5.2-1 are from three different types of continuous miners, four coal seams, and five mines. These data should give a reasonable idea of the sizes of materials which could reasonably be expected to come from a continuous miner.

Approximately 80% of the coal coming from the miner is less than two-inch lumps. While occasional pieces may range up to more than a foot in diameter, and large slabs several feet in diameter could be delivered, they would appear to be infrequent problems. On the basis of this information, an arbitrary decision to reduce all coal to minus two-inch material was made.

Several types of crushers were examined for the face haulage application. Of these, the large mobile feeder-breakers, such as those manufacturered by Long Airdox or Stammler, appear to be best suited for reduction of the coal to a size suitable for pipeline transport. Such devices also provide a surge capacity of about nine tons which would help to reduce some of the production surges at the miner.

5.2.2 Slurry Mixing and Introduction into the Pump

One of the most difficult tasks in the design of the face haulage subsystem is the introduction of the coal into the pipeline. The low head room available in the coal seam makes most conventional slurry mixing devices impractical because the majority of these devices are quite large in relation to their capacity. The mixing requirements postulated for this application are a minimum of four to eight tons of coal per minute and up to twice this weight of water.

The simplest mixer would be a large tank which would receive lump coal and water. A high-pressure water stream would wet the coal and agitate the slurry in the tank so that no settling would occur. Such a tank would have to provide an adequate suction head and a large volume of slurry would have to be available at the pump. The tank would also require a conical bottom so that all solids would enter the pump.

In order to meet the requirements of low head room, high volumetric capacity, and positive head at the suction intake of the pump, several means of transferring the slurry from the mixing tank to the pump were examined. An eductor placed at the bottom of the conical tank was considered. In this configuration, the coal would be swept out of the bin and into the suction of the pump by the high-pressure water jet. This system appeared to have considerable merit, but it suffered from the inability to control the proportions of coal and water delivered to the pump.

The system finally selected for use is the hopper and screw feeder illustrated in Figure 7. This system receives the dry coal from the

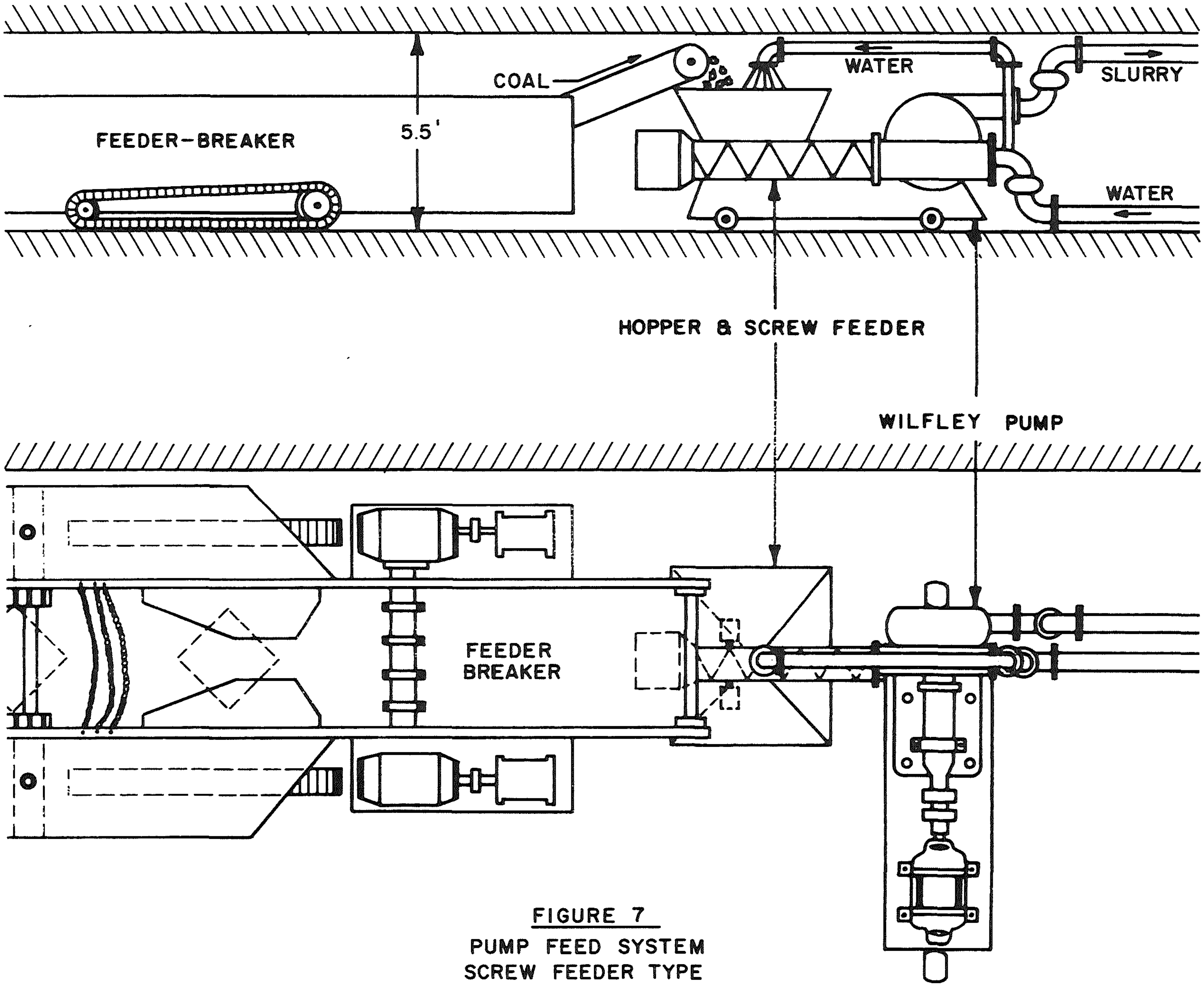


FIGURE 7
PUMP FEED SYSTEM
SCREW FEEDER TYPE

feeder-breaker in a hopper to which water is continuously added. The mixture then drops into a screw-feeder conveyor which forces it through an enclosed tube at the bottom of the hopper and into one suction opening of a double suction opening pump. The other suction opening is supplied with additional fresh water. The pump combines the two feed systems and discharges a 33% by weight solids mixture of water and coal into the transportation line. This system will require close control to prevent spillage while high maintenance on the auger is a distinct possibility. However, a feed system constructed on this principal has reportedly been used successfully at Mine 15 of the Selidovugal Trust in Russia[11]. This is the only unit discussed in this study which is not available from an established manufacturer.

5.2.3 Pump Selection

From the inception of this study, the hydraulic transport system has been based upon centrifugal pumps. This is because of their ability to handle large solids particles and their relatively small size and high volumetric capacities. Their obvious shortcoming is their inability to produce high heads, but this may be overcome by additional pump stations at appropriate spacings.

The examination of various manufacturers' pump catalogs revealed that the number of pump styles available is great. However, the requirements of a pump having sufficient capacity, small enough to fit the available opening, with a double suction, and a mechanical seal simplified the selection process. The pump selected was a Wilfley Model 10K pump. This pump is manufacturered with a vertical discharge leg, but discussions with the manufacturers indicated that this pump could readily be converted to a horizontal discharge. The pump has a double suction, requires no seal water, and has excellent slurry head characteristics. This was also the largest capacity pump which could be accommodated in the specified seam height.

5.2.4 Hydraulic Characteristics of the Face Haulage Subsystem

A complete discussion of the theory and results of the hydraulic analyses conducted as a part of this study are included in the appendix as Exhibit 1. This discussion presents the facts and problems associated with the transport of coarse coal slurries and is much too detailed to include here. However, a summary of hydraulic data used for the face haulage subsystem design is shown in Table 5.2.4-1.

Table 5.2.4-1

Data Used for Hydraulic Design of Face Haulage System*

Velocity fps	Slurry Friction Losses psi/1000 ft	Power Required To Overcome Friction kwh/ton/1000 ft	Throughput Tons Per Hour
10**	33.5	0.1572	228.1
12	35.2	0.1652	273.7
14	38.7	0.1816	319.4
16	43.6	0.2047	365.0
18	49.8	0.2337	410.6
20	57.2	0.2680	456.2

*Absolute Pipe Roughness	0.00015 feet
Pipe Inside Diameter	10.020 inches (Schedule 40)
Slurry Concentration	33.3% Solids by Weight
Slurry Specific Gravity	1.108
Sliding Bed Flow Regime	
Maximum Particle Size	two inches

**Velocity near critical - below recommended minimum

From the data shown in Table 5.2.4-1, it will be seend that at the lowest recommended velocity, 273.7 tons per hour, or 4.6 tons per minute, of coal would be removed from the continuous miner. This was the maximum cutting rate selected for the direct hydraulic feed system.

5.2.5 Face Haulage System Production Rates

In an effort to devise a system capable of taking coal away from the mining machine as fast as it was produced, the surge feed system was developed. Table 5.2.5-1 summarizes surge requirements for the cutting rate developed in Table 5.1-1 and assuming 4.6 tons per minute are removed by the hydraulic haulage system.

Table 5.2.5-1

Surge Requirements for Maximum Sustained Continuous Miner Cutting Rate (Assume No Delays)

Operation	Time (min)	Coal Cut (tons)	Coal Removed (tons)	Cumulative Surge (tons)
Make first 10 ft wide cut 20 ft deep	7.0	51.1	32.2	18.9
Back out and position for 6 ft wide cut	3.0	----	13.8	5.1
Make second cut	4.2	30.6	19.3	16.4

An examination of Table 5.2.5-1 reveals that the nine tons of storage capacity available in the feeder-breaker are not adequate to allow the mining machine to operate without interruption caused by the haulage system. An additional nine tons of surge capacity is obtained by inserting a shuttle car between the miner and the feeder-breaker. This arrangement is the one finally selected and is shown in Figure 8. By utilizing this system, an 18-ton surge capacity is provided. The greatest probable surge requirement may exceed the capacity slightly, but this is unlikely as numerous minor delays for safety, etc. will reduce the cutting rate somewhat. If desired, an eight-foot mining machine may be used in place of the 10-foot unit, and in this case, there is adequate surge capacity in the two available units.

5.2.6 Face Haulage System Extensibility

In order for the face haulage subsystem to work, it must be able to follow the miner on both the advance and retreat. While some flexibility is obtained from the tail boom of the miner, this is not enough to allow a 20-foot advance. The shuttle car and feeder-breaker are both mobile and could easily follow the miner. The slurry module would be attached to the feeder-breaker and would move with it. This leaves only the problem of extending the pipes. One method was suggested by Dahl[4], and two alternatives are discussed here.

Figure 9 illustrates the use of Chicsan couplings to construct a section of pipe capable of folding like a pantograph. This coupling permits 360° of rotation in a horizontal plane between adjacent sections of pipe. Use of these devices in conjunction with eight-foot lengths of pipe would permit pipelines to be folded accordion-wise in the mine opening behind the miner and unfolded as the miner advances. Figure 9-a is a vertical section showing this arrangement. Figure 9-b is a plan view showing the pipes extended, while Figure 9-c is a plan view showing the pipes contracted to allow forward movement.

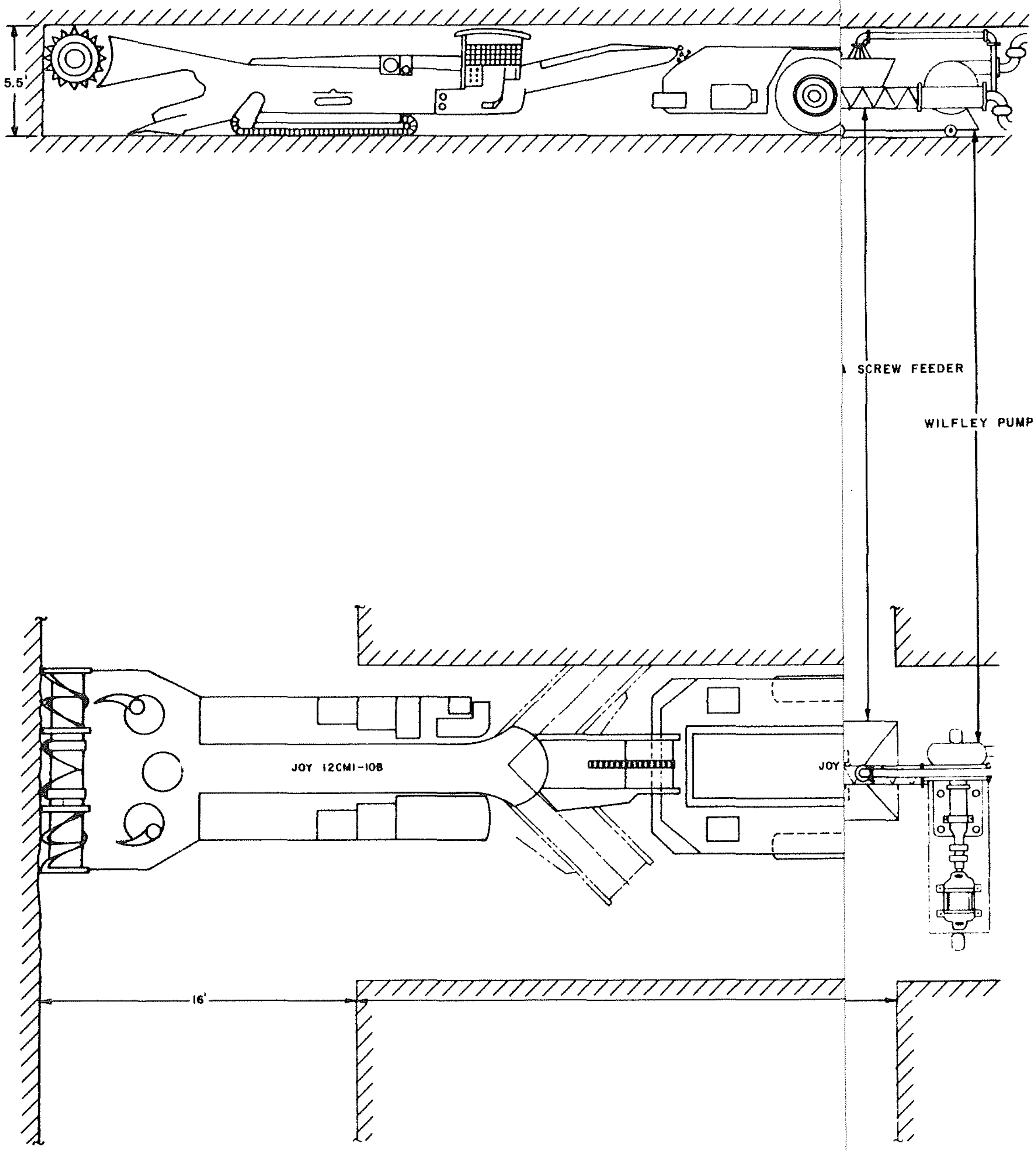


FIGURE 8 - FACE ARR

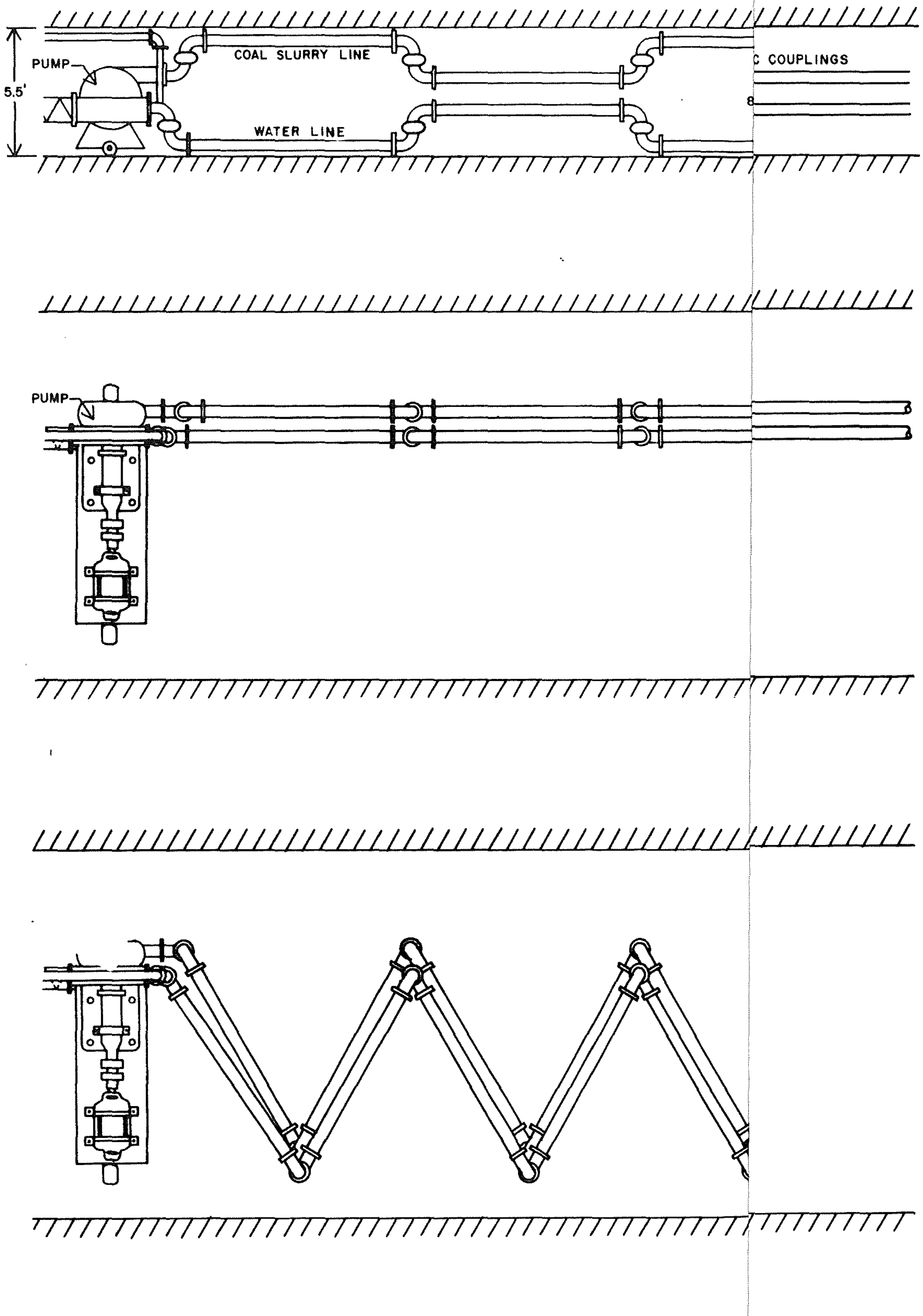


FIGURE 9-

Approximately 40 feet of such line would be used per face. The idea has not been entirely rejected, but machine travel through the section could be difficult and problems can be expected when vertical deviations in the coal seam require vertical movement in pipe joints. In addition, assembly and disassembly of these joints could prove to be cumbersome and time-consuming, while coal degradation, friction losses, and pipe wear at the joints would be severe.

A preferred method of achieving flexibility is the use of flexible hose in 100-foot lengths. The bending radius of such hose was originally thought to be too great to permit its use within the confines of the 16-foot-wide mine openings. However, Gates Rubber Company indicates that their 10-inch hose has a bending radius of 50 inches, and it is quite suitable for use in 16-foot-wide openings. A 100-foot section can be looped back and forth so that it will occupy about 54 feet of entry and permit 46 feet of movement of the miner by extension of the hose. This is enough to permit a 20-foot advance or movement through a crosscut to the adjoining room or entry. In addition, there are no couplings to deteriorate and leak. Steel skids or wheels inserted under the hose at intervals will decrease hose wear resulting from sliding on the rough bottom of the working area. The hose may also be suspended at intervals along the entry from a monorail, like a shower curtain. A small cable hoist located at the pump would be used to retract the hose so that additional joints of pipe could be added. Such an arrangement is shown in Figure 10, and is the most promising of the methods developed in this study. It should be noted in Figure 10

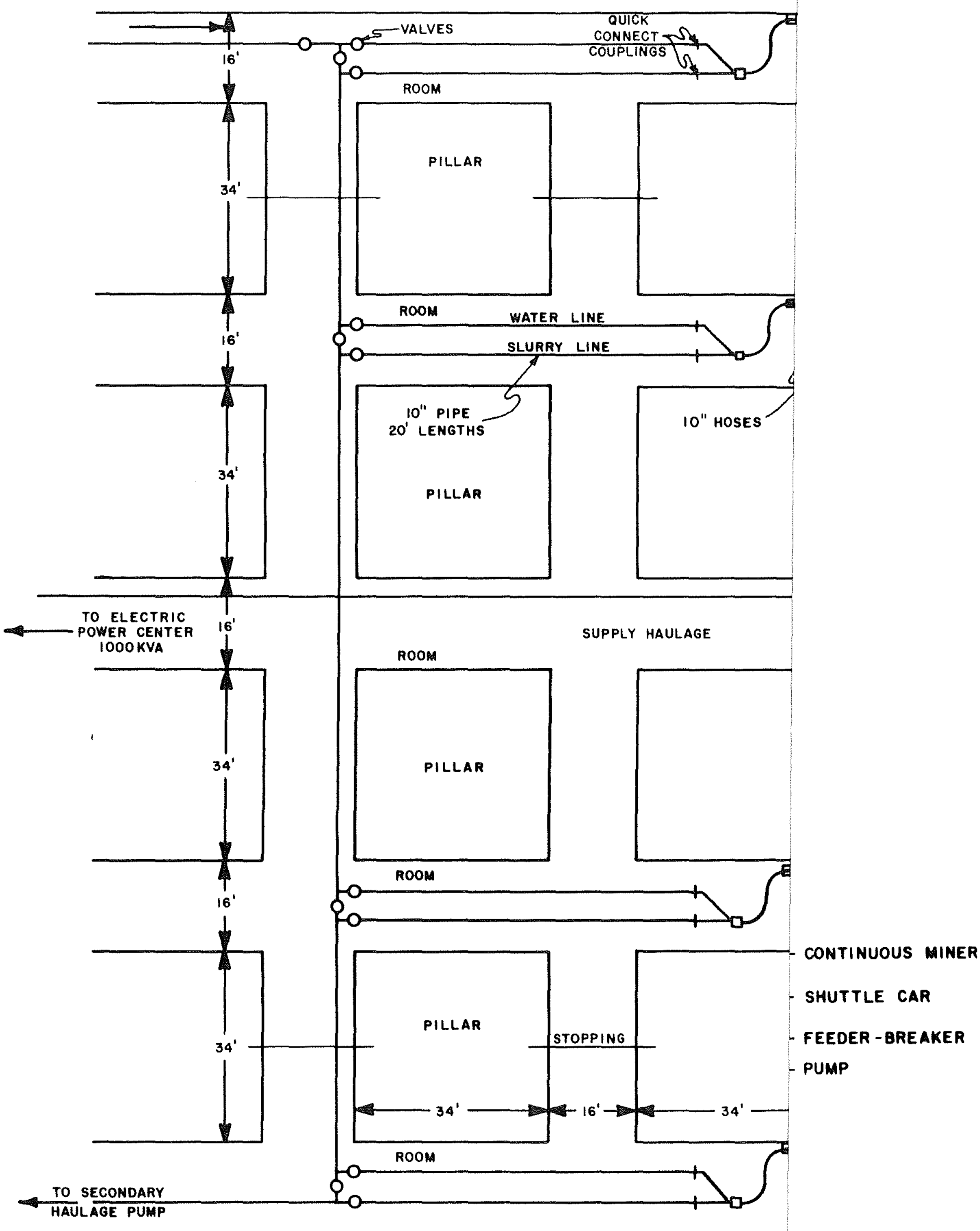


FIGURE 10 - HYDRAULIC TRANS

that the water, slurry lines, hoses, and pump module are located in four of the five room entries so that only the feeder-breaker and shuttle car must move with the miner from entry to entry. This is the layout for which subsequent cost data were developed.

5.2.7 Alternative Face Haulage Subsystem

In an effort to simplify the face equipment for hydraulic haulage, a direct miner to pump feeder configuration was examined. In this system, a breaker is incorporated into the continuous miner to prevent oversize coal from plugging the pump. The slurry module, hose, and piping remain the same as those shown in Figure 10. The shuttle car and feeder-breaker are eliminated, and miner cutting rate is reduced to a maximum of 4.6 tons per minute.

A revised cycle time was developed for this system based upon the data shown in Tables 5.1-1 and 5.1-2. Cutting rates for both the sumping and shearing operations are reduced to 4.6 tons per minute. This increases the cutting time per place from 11.2 to 17.8 minutes. The revised cycle time for this system is 58.3 minutes, assuming no other changes.

Using the same shift time as before (415 minutes), daily production with this system is 582 tons per miner. This is somewhat lower than the production of the shuttle car feeder-breaker system, but it also requires less equipment and may offer greater flexibility. The system will also probably result in less efficient operation of the other hydraulic transport links, because of the irregular flow of coal to the pump. Costs for this system are also included, as an alternative approach, because the simplicity of the system may make it more desirable for coal mine applications.

5.2.8 Face Haulage Subsystem Pump Power Requirements

The hydraulic face haulage subsystem design was based on the computer analysis of slurry flow characteristics discussed in the Appendix, Exhibit 1, and summarized in Table 5.2.4-1. This analysis included various sizes and quantities of material, in various sizes of pipes, and at various velocities, under conditions of sliding flow and saltation flow. A flow of approximately 2,200 gallons per minute of water is needed to transport 4.6 tons per minute of material at a concentration of 33% by weight. The material is assumed to be a combination of coal with a specific gravity of 1.36 and rock having a specific gravity of 2.75, which results in a material with an average specific gravity of 1.42.

The largest pump which would fit into the 5.5-foot-high coal seam is a 10-inch size and this is coupled with a 10-inch pipeline for the slurry, which will move in the sliding bed flow regime, at a velocity of 12 feet per second. Friction losses are predicted to be approximately 35 psi per 1,000 feet of pipeline.

The pump in this system has a water head of 188 feet. The introduction of solids will reduce the efficiency of the pump and its effective head. By increasing the pump speed, the slurry head will be increased, but more power is required.

The art of predicting losses in pump efficiency is not well-developed. However, McElvain[12] has presented a system for estimating the effect of solids on pump efficiency. His method indicates that pump efficiency would be 74% of the clear water pump efficiency under the slurry conditions postulated. Assuming a centrifugal pump efficiency of 70% with water, the slurry efficiency of the pump would be about 50%.

A combined pump-motor horsepower efficiency of 45% was used in this study. This is believed to be quite conservative and greater efficiencies would be experienced in actual practice.

The study assumed that slurry from the face haulage subsystem would be delivered to a surge hopper at the secondary haulage pump. This appears inefficient because any surplus head from the face pump would be lost and an additional slurry module would be required at the secondary haulage system. Therefore, pump motors were sized for a series application as well as the case where separate slurry hoppers are used.

Pump motors were sized for the extreme case where the miner is working at its maximum distance from the next downstream pump. The motor horsepower required to operate the pump is computed by the following equation

$$hp = \frac{QH\gamma}{550\eta} \qquad (1)$$

where Q = Slurry flow rate in ft^3/sec

H = Pressure head in feet of water

η = Combined pump-motor efficiency

γ = Specific weight of water, lbs per cubic foot

In the case where the miner is working 500 feet from the secondary haulage system pump, the required horsepower is:

$$hp = \frac{6.64 \times 500 \times 35 \times 62.4}{550 \times 1000 \times .433 \times .45} = 67.7$$

A small allowance over the computed horsepower is required to allow for minor losses in fittings, etc. For this reason, a 75-hp motor is recommended for the face haulage subsystem.

As indicated earlier, the staging of the face and secondary haulage system pumps in series appears to offer advantages. Assuming the total distance from the face to the main haulage system will not exceed 2,800 feet in the panel entry and 500 feet in the room entry, the face and secondary haulage pumps must move the slurry a maximum of 3,300 feet. Required horsepower for this distance would be:

$$hp = \frac{6.64 \times 3,300 \times 35 \times 62.4}{550 \times 1,000 \times .433 \times .45} = 446.55$$

Allowing for minor losses in fittings, a total of 500 horsepower appears adequate. In this configuration, a 200-hp pump is recommended for the face haulage system and a 300-hp motor for the secondary haulage system.

5.3 HYDRAULIC SECONDARY HAULAGE SUBSYSTEM

The hydraulic secondary haulage subsystem may receive coal from either a conventional shuttle car face haulage system or the hydraulic face haulage system. It must move the coal to a transfer station up to 2800 feet away where it will be delivered into the main hydraulic transport system.

The slurry preparation and feed system developed for the face haulage subsystem would be used wherever introduction of coal from a conventional haulage subsystem, such as a belt or a shuttle car, is necessary. By positioning a feeder-breaker to introduce coal into the module, the conventional and hydraulic systems may be readily joined.

The secondary haulage subsystem is less complex than the face haulage subsystem because it is moved only when not pumping slurry. The slurry module or pump will be moved as necessary into or out of the panel entry as the working faces advance or retreat. The system would be shut down and pipe additions or deletions would be made as needed on an idle shift. The pipe selected for the secondary haulage subsystem is the same 10-inch pipe used in the face haulage subsystem, and slurry velocities would be similar

When operating a slurry module on the secondary haulage subsystem in conjunction with the hydraulic face haulage subsystem, slurry would be introduced directly into the hopper of the slurry module. The water and solids ratio would already be established, and the slurry mixed so that no additional water would be added at the pump. Under these conditions, the screw feeder would be used only to prevent solids from settling out on the bottom of the feed hopper. As feed would enter only one of the suction openings in the pump, the other would be redundant. For this reason, a single suction pump, purchased at a slightly lower cost, might be used.

Because the face haulage and secondary haulage subsystems are not staged in this system, a 400-hp motor would be required to move the slurry through the 2,800 feet of pipe in the secondary haulage subsystem.

As indicated earlier, the most efficient combination of face and secondary haulage systems would stage the pumps so that any surplus pressure head from the first pump would provide positive suction head at the secondary haulage pump. The secondary haulage pump would require a varispeed drive so that the combined pump outputs would match the friction losses in the pipeline and maintain a constant flow. This is necessitated by the variations in solids content of the slurry from the slurry module, and the variation in length of the secondary haulage subsystem pipeline. The varispeed drive recommended for this service was described in a recent publication[6].

Such a configuration should also allow the elimination of the slurry module at the secondary haulage pump. In this study, no cost advantage is assumed because the slurry module is replaced by the varispeed drive. However, the varispeed drive controls and instrumentation would be much simpler than those for the slurry module and the system would be more reliable. Spills would also be eliminated by removing the hopper from the system.

A 200-hp motor was recommended earlier for the face haulage system while a 300-hp motor was suggested for the secondary haulage system. A 300-hp motor for this application would allow interchangeability with main haulage motors and might offer an inventory reduction in spare motors. A similar advantage could be gained by using a 250-hp motor at the face.

5.4 HYDRAULIC MAIN HAULAGE SUBSYSTEM

The hydraulic main haulage subsystem will receive coal from five conventional panel belt conveyors or five secondary hydraulic haulage systems, and move the coal to the mine mouth or shaft bottom where it will be prepared for the hydraulic hoisting subsystem. The hydraulic main haulage subsystem is capable of replacing conventional main line belt or rail haulage systems.

The main haulage subsystem is the most complicated of all of the hydraulic subsystems. It must receive coal from zero to five producing sections at any time and in any combination. At the same time, this material must be transported in the system at or near a gravimetric concentration of 33% for reasonable efficiency. This latter requirement is made extremely complex by the fact that any given panel will be producing coal for about 20 minutes out of each hour or less than three hours out of each shift, with no coordination of production time between panels.

The possibility of providing surge storage for slurry at the end of each panel to smooth out main hydraulic system flow was considered briefly and abandoned. Volumetric requirements for slurry storage would be quite large, and solids would quickly settle out. The limited head room available dictated that the area of a storage facility would be quite large and the reclamation of the settled solids would be extremely difficult.

A second possibility was the use of varispeed drive pumps to vary the slurry velocity depending on the number of sections operating. The assumption was made that one pipe would be used and the minimum velocity for slurry from one section would be above the stationary bed formation velocity. The suggested pipe size and hydraulic data for these conditions are shown in Table 5.4-1.

Table 5.4-1

Transport Velocities for Coal From One to Five Mining Sections in a 12-Inch Pipe at 33% by Weight

No. Sections	Slurry Velocity fps	Friction Loss fps/1,000 ft	Dry tons/hr
1	8.5*	34	274
2	17.0	41	548
3	25.5	55	822
4	34.0	**	1096
5	42.5	**	1370

* Estimated plugging velocity 9.3 fps
**Not Computed

From Table 5.4-1, it may be seen that slurry velocity would have to range from 8.5 to 42.5 feet per second in order for all of the sections to utilize a single pipe at 33% concentration by weight. Furthermore, the velocity for one section would be below the system plugging velocity. Because of the severe problems of power requirements, wear, and water hammer associated with high slurry velocities, this approach was dropped.

Another approach considered was to utilize a single pipe design for optimum flow conditions with all five sections. As coal production decreased, the addition of water would maintain slurry velocity. Slurry concentrations and other hydraulic data for this approach are summarized in Table 5.4-2.

Table 5.4-2

Slurry Concentrations for One to Five Mining Sections Introducing Slurry Into a 24-Inch Pipe at a Constant Velocity of 15 fps

No. Sections	Concentration % by Weight
5	33.3
4	27.3
3	20.9
2	14.2
1	7.3

From Table 5.4-2, it will be seen that slurry concentrations would vary from seven to 33% in a 24-inch pipe. Velocity would be constant and near the allowable minimum.

A simple probability analysis was made to determine the probable number of mining sections producing coal at any one time. This study was based on the coal production time and total cycle time developed in Table 5.1-2 and assumed five producing sections were available.

On the basis of the analysis described above, it appears that one, two, or three of the five sections would be producing coal 84% of the time. The rest of the time, no coal would be produced or four or five sections would be operating. On this basis, the main haulage system would be operating at approximately 14 weight percent concentrations 84% of the time.

Although no economic analysis was made for this type of operation, it was discarded. Power requirements on a dry ton basis would be at least 50% greater for the more dilute slurries than for the 33 weight percent slurries. Much larger pumps, motors, and pipes would also be required, and this would necessitate substantially larger excavations to house the pumps.

A combination of two pipe sizes appeared to offer the greatest flexibility and economy for the main haulage subsystem. The pipe sizes selected for the main haulage system are 14 and 16 inches. Flow conditions are optimized for the maximum throughput. This reduces inefficient operations to those times when one or four sections are operating.

Under normal operating conditions, when only two sections are producing, all flow would be handled by the 14-inch line, at 14 feet per second. Three sections could be handled in the 16-inch line at 16 feet per second. Four sections could be handled in the 16-inch line at 21.5 feet per second, or they could be split between the two lines. Hydraulic data for these conditions are summarized in Table 5.4-3.

Table 5.4-3

Hydraulic Data for One to Five Operating Sections Providing Slurry to 14- and 16-Inch Pipes at 33 Weight Percent Solids

No. Sections	Velocities fps	Friction Loss psi/1,000 ft	Throughput tons/hr
14-Inch Pipe (13.126 in. I.D.)			
1	7.07*	33	273.7
2	14.13	35	547.4
3	21.20	49	821.1
16-Inch Pipe (15.250 I.D.)			
2	10.47*	32	547.4
3	15.70	34	821.1
4	20.94	44	1094.8

*Minimum recommended velocity 12 fps.

From the data in Table 5.4-3, it will be seen that slurry velocities for one section in the 14-inch line and two sections in the 16-inch line are below minimum recommended velocities. For one section operation, slurry concentration would be reduced by adding water until an acceptable velocity is obtained in the 14-inch pipe. Under any other combination of operating sections, the slurry would be directed into the appropriate line or lines. Improved operating efficiency could be obtained by the addition of a 10-inch-diameter line to the 14- and 16-inch-diameter pipelines. However, capital investment, which is already heavy, would be increased considerably. The period of time of inefficient operation is not believed to be great enough to warrant the expenditure. However, simulation studies would allow the optimum system to be designed.

5.4.1 Pump Selection

The selection of pumps for the main haulage subsystem is much simpler than for the other subsystems because the main pump will not require portability. Pump stations will require rock excavation above or below the coal and should be located at the intersections of panel entries with the submains, or at the intersections of the submains and the main entries. In this study, submains are spaced to provide 5,000-foot-wide solid blocks of coal so that the distance between main haulage intersections becomes 5,600 feet. Main haulage pump station spacing should be 5,600 feet in the mains and 850 feet, at the panel entry transfer points, in the submains. Power requirements would depend upon pump characteristics; pumps could be multistaged at one location or distributed along the main entries where friction losses for 5,600 feet exceed available single pump heads.

5.4.2 Motor Selection

As mentioned earlier, any number of large centrifugal pumps are available for the main haulage subsystem. A number of manufacturers provided data for their pumps, and the following power calculations are based on data provided by the Georgia Iron Works.

For the submain entry, the pump spacing is 850 feet. Assuming two sections feeding the 14-inch line, the horsepower required would be:

$$hp = \frac{13.28 \times 35 \times 850 \times 62.4}{550 \times .433 \times 1000 \times .45} = 230.0$$

A similar computation for three sections in the 16-inch line shows:

$$hp = \frac{19.92 \times 850 \times 34 \times 62.4}{550 \times 1000 \times .433 \times .45} = 335.2$$

A 350-hp motor was selected for use with the 16-inch pump, while a 250-hp motor was selected for the 14-inch pump.

In the main entries, the coal must be moved 5,600 feet. This distance is too great for a single pump, so for this study, a pump spacing of 2,800 feet was assumed. An alternative would be to stage the two pumps at the submain intersections. Total cost and power requirements would probably be similar. Power requirements for the 14-inch pumps are:

$$hp = \frac{13.28 \times 35 \times 2800 \times 62.4}{550 \times 1000 \times .433 \times .45} = 757.8$$

Power requirements for the 16-inch pumps are:

$$hp = \frac{19.92 \times 34 \times 2800 \times 62.4}{550 \times 1000 \times .433 \times .45} = 1104.2$$

An 800-hp motor was selected for the 14-inch pump, and a 1,200-hp motor was selected for the 16-inch pump. This small excess capacity would be adequate to overcome minor losses from fittings, etc., and would provide a margin of safety in the event slurry concentrations above 33% occurred.

Each pumping station will be a transfer point for material in transit and will constitute the end of the inbye pipeline. The free, open-end flow from this line will be directed into a hopper which will feed the succeeding set of pumps by gravity. This will minimize the effect of water hammer in the slurry lines and permit transfer of slurry between pipelines as necessary to maintain efficient concentrations. Each hopper must be kept full to prevent the pump from sucking in air and could be fitted with a slurry module to receive coal from a conventional transport system if necessary.

5.5 HYDRAULIC HOISTING SUBSYSTEM

The hydraulic hoisting subsystem receives coal slurry and transports it to the surface. This system must be capable of replacing the conventional hoisting system at depths of 300, 500, 1000, 1500, and 2000 feet.

On the basis of Institute experience, only two hydraulic hoisting systems appeared to have merit. However, in an effort to identify potential systems with which the authors were not familiar, a brief literature survey was undertaken. Several systems from around the world were identified, and a publication[13] from Pennsylvania State University was obtained.

Hydraulic hoisting systems may be easily classified on the basis of the size of solids they are capable of passing. Hoisting systems for fine solids (up to one-fourth inch or 6 mm) generally pass the solids through the pump. Hoisting systems for coarse solids generally require that the solids be introduced into a high-pressure fluid stream downstream from the pump.

Hoisting systems for fine solids are generally based on staged centrifugal or positive displacement pumps. Chu and Li[14] describe a centrifugal pump and chamber-type feeder for use in hydraulic hoisting of coal. The use of Allen-Sherman-Hoff pumps to remove mud and tailings from a depth of 3,500 feet is described by Wildon[15]. It is said that over 40 of these pumps are being used in New Mexico for dewatering deep uranium mines.

Mitsubishi is marketing a piston-type pump with claims that it will pass one-quarter-inch solids. Slurry is isolated from the cylinder by an oil-filled reservoir, although it still must pass through the valves. A scheme for pumping 25,000 dry tons per month of quartzite fines from 7,200 feet by means of Mitsubishi Mars reciprocating pumps is described by Gyngell[16].

A system similar to the Mars pump, called the Jupiter system, relies on water to replace the piston and create a surging effect on oil in the pumping chamber[17]. Hitachi has also created a system for handling fine slurries in vertical cylinders. The driving force is a high-pressure fluid which is separated from the slurry by a floating ball.

Hoisting systems for coarse coal are much more complex because of the inability of high-pressure pumps to pass coarse solids. For the most part, these pumps pass solids or slurry into an isolation chamber at relatively low pressure. The chamber is then sealed and pressurized and then opened to the high-pressure system.

The Hydrolift system for hoisting coarse solids relies on a high-pressure water stream passing through a hopper filled with solids. The stream entrains the solids and then passes up through a vertical pipe in the top of the hopper.

The Hydrolift system for the hydraulic hoisting of coal or ore vertically up a mine shaft has been described in several references. Laubscher[17] and Laubscher and Sauermann[18] point out that the method was developed in a pilot plant at the University of New South Wales, Australia, and is patented in Australia, South Africa, and several other countries. Buchanan and others[19] state that the device was developed for and ores. The potential cost of hoisting 200 tph of minus four-inch coal 1,600 feet was estimated to be five cents per ton in 1962, divided 38% for power, 50% for labor, and 12% for plant maintenance. He also gives equations for calculations of pressure requirements for vertical transportation of coal and ores.

5.5.1 Shaft Bottom Layout for Hydraulic Hoisting

The shaft bottom layout for the hydraulic hoisting system is essentially the same as that for the conventional mine and is shown in Figure 11. The essential difference is the inclusion of the slurry dewatering facility in conjunction with the loading pocket.

Coal slurry from the main haulage subsystem discharges into the loading pocket shown at Section A-A′ in Figure 11. The solids and some water immediately settle to the bottom of the pocket where they are fed into the sand pump which feeds the hydrohoist. The bulk of the water enters the overflow pipe, which is 20 inches in diameter, and is pumped to the recirculating water sump. Additional fine solids will be deposited in this sump before the water is returned to the mine via the 20-inch makeup waterline and pumps.

The rail haulage system and other shaft bottom features are identical to those for the conventional haulage system. The only additional and significant difference is the excavation of the large gallery required by the Hydrohoist or pumps for hoisting.

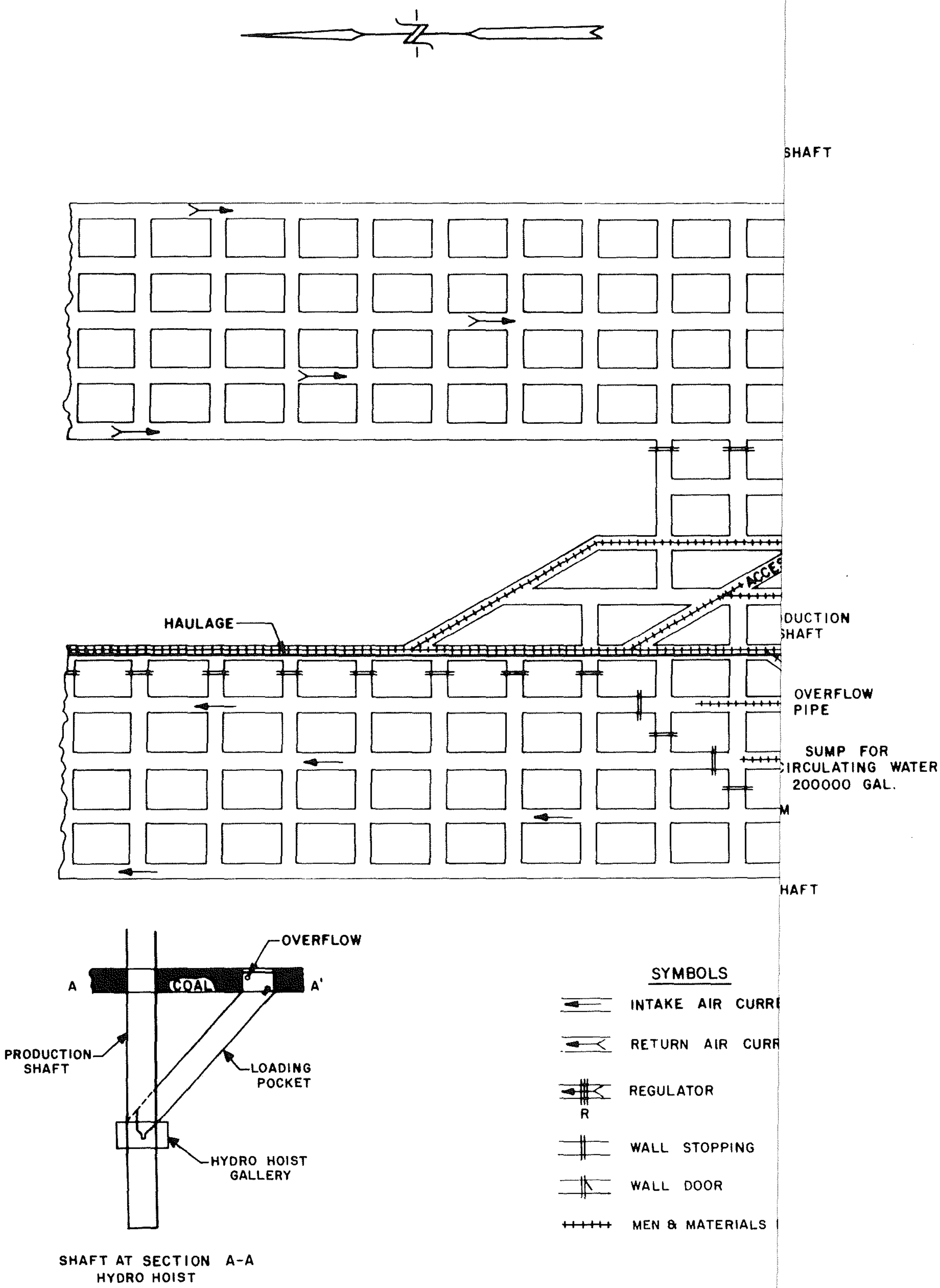


FIGURE II- SHAFT BOTTO

Dilution of the slurry during hydraulic transport would occur whenever a new slug of slurry entered a water-filled segment of the haulage system. This dilution water would be recovered in the sump so that the hoisting operation could not be effected. An additional benefit of the sump would be its capacity to absorb surges in mine production so that a uniform hoisting rate may be maintained.

A further advantage of the slurry dewatering sump occurs in the event of a power failure. If the pumps used to hoist the slurry failed, the slurry would flow back into the sump safely and no flooding would occur. In the event of a total power failure, all flow of water in the mine would cease because the pumps would stop.

5.5.2 Description of the Hydrohoist

The Hitachi Hydrohoist, designed for high-head transportation of relatively coarse granular solids in slurry form, is described in articles by Singhal[2], Sakamoto[31], Terada[32], and Yamaguchi[33], and in an undated brochure from Hitachi, Ltd.[34], explaining the development of this equipment by Showa Denko K.K. and Hitachi, Ltd. The system is covered in U. S. Patent No. 3,449,013, issued June 10, 1969.

Considerable familiarity by the authors with the Hitachi system resulted in its selection as one of the candidate hoisting systems for this study. A detailed discussion of the system will be found in the literature, but a brief explanation of the system is included here for those not familiar with it.

The Hydrohoist is shown in Figure 12 and consists of three parallel horizontal large-diameter pipes, valves, controls, and so forth. Water is

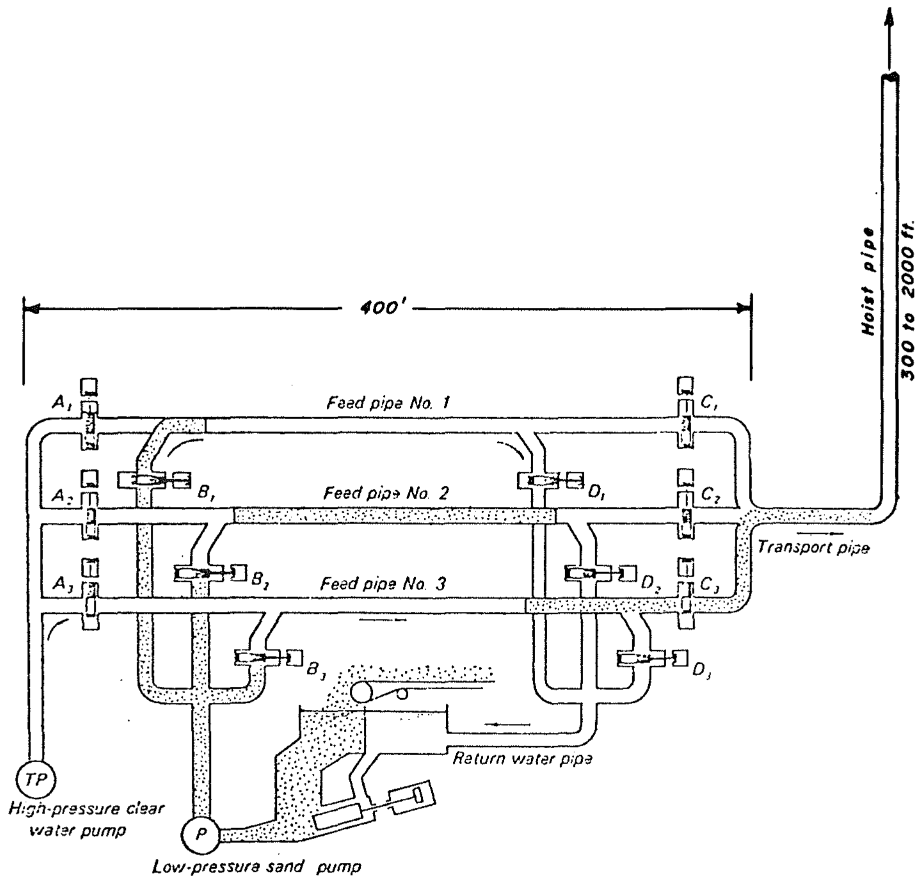


FIGURE 12

PRINCIPLE OF HYDRO-HOIST

delivered to a high-pressure water pump located at the underground hoisting station. Coal slurry is charged into each of the large horizontal pipes in turn by a low-pressure centrifugal pump which is fed from the coal pocket by a small chain feeder. The pipe is then isolated from the low-pressure line by plate valves. Next, the plate valves connect the slurry filled line to a high-pressure waterline and the hoisting pipe. The high-pressure water forces the slurry out of the horizontal pipe and the plate valves close. By utilizing three chambers, the process is essentially continuous.

Considerable correspondence with Hitachi resulted in their providing a quote and specifications for the Hydrohoist installations discussed in this report. This facility will handle surges of up to 15 dry short tons of coal per minute. The maximum particle size at the face is 2.0 inches, but this degrades in hydraulic transport to 1.4 inches (35 mm); maximum slurry concentration is 33% by weight.

A maximum hoisting capacity of 410 tons per hour is required for the surge feed hydraulic system. This is hoisted through a 16-inch pipe at a velocity near 10 feet per second. Pipe wall thickness ranges from 0.75 inches for the 300-foot shaft to 1.2 inches for the 2,000-foot shaft. Shaft collar pressures are adequate to transfer the slurry through 3,000 feet of horizontal pipe, so no additional pumps are required. Data for the Hydrohoist are summarized in Table 5.5-1.

Table 5.5-1

Details of Hydrohoist Installations

Shaft Depth	Motor hp	Installed Cost (Thousands $)
300	1,600	2,629.6
500	2,500	2,914.8
1,000	4,000	3,632.4
1,500	5,000	3,632.4
2,000	7,000	3,922.8

Table 5.5-2

Details of Warman Pump Installation

Shaft Depth	No. Pumps	Service	Motor hp	Installed Cost (Thousands $)
300	4	Hoist	1,000	
	2	3,000' Horiz	500	
Total	6		1,500	161.3
500	5	Hoist	1,500	
	2	3,000' Horiz	500	
Total	7		2,000	205.8

5.5.3 Alternative Hoisting System

An examination of Table 5.5-1 shows that capital costs for the Hydrohoist applications in the 300-foot shaft are about two-thirds of those for the 2,000-foot shaft. This cost appeared excessive for such shallow depths and the possibility of using centrifugal pumps was investigated.

McElvain[12] reported that the Warman pump company is developing large centrifugal pumps with cases capable of operating at pressures of 750 psi. Such case pressures might be achieved in multistage pump hoisting installations. Correspondence with Mr. McElvain revealed that such pumps are now available, and he provided costs and specifications for them.

Table 5.5-2 summarizes cost data for centrifugal pump installations in 300- and 500-foot-deep shafts. Costs are only a fraction of those for the Hydrohoist installations in the same shafts. Centrifugal pump hoisting systems are not suggested for the deeper shafts because the number of pumps required would probably adversely affect the larger coal sizes and produce excessive fines.

In each of the cases presented, two pumps are required to move the slurry from the shaft to the preparation plant 3,000 feet away. These may be staged at the shaft collar, or spaced as in the main haulage system.

5.6 SURFACE FACILITIES

5.6.1 Surface Transportation Facility

Transfer of the coal from the shaft collar to the preparation plant is accomplished through 3,000 feet of horizontal pipe. A 12-inch I.D. pipe is adequate for this service, and slurry velocity is about 12.6 feet per second. Pumping pressure is provided by the Hydrohoist at the shaft bottom or by the staged centrifugal pumps at the shaft collar.

5.6.2 Raw Coal Storage for Preparation Plant

The dewatering step for the coal slurry was described earlier in Section 3. Although the preparation plant would require wet coal, and the hydraulic mine could deliver it directly, the dewatering step has been included to allow a cost comparison for a raw coal mine as well as one using a coal preparation facility. Some additional costs for more fine coal recovery circuits might be expected in the preparation plant sewed by a hydraulic transport system.

In addition to the dewatering screens, a hydraulic thickener, water clarifying pond, and pump station would be included to transfer the clarified water back to the mine. An eight-inch pipe would be adequate for this service.

Raw coal from the dewatering stage would be stored in a silo similar to the one described for the conventional haulage mine. The preparation plant would be 400 tons per hour capacity because of the greater output from the hydraulic system. Such a plant, exclusive of the dewatering and coal storage equipment described above, cost approximately $1,720,000 in late 1973.

5.6.3 Substation

Power requirements for the Hydrohoist and pumps are much greater than for the conventional haulage mine. Substation capacity for the conventional haulage mine was 7,500 kva. The substation for the hydraulic transportation mine is 20,000 kva. Substation sizes will vary somewhat with shaft depth, and the figures given are for the 2,000-foot-deep mine.

5.7 HYDRAULIC SYSTEM INSTRUMENTATION AND CONTROLS

The complexity of the hydraulic transportation system will make manual operation of the system undesirable, except in emergencies. A small process control computer, such as those manufactured by Honeywell, located at the shaft bottom, would be well-suited to controlling the operation.

Input data from various parts of the mine would be transmitted to the computer via telephone cable, and instructions from the computer would be transmitted in the same way. Manual override would be possible at any of the main pump or control stations.

The system would be provided with electrical interlocks so that a failure of a portion of the system would shut down all elements upstream from the failure. This would allow unaffected areas of the mine to continue in operation. Emergency shutdown and start-up controls would also be provided at all pumps and control stations.

In the event of a major failure, such as a power interruption, the entire system would be filled with settled solids. Emergency dump valves in the shaft bottom would allow the vertical pipes to drain back into the shaft bottom slurry pocket, and valves at the shaft collar would close the overland pipe so that slurry could not flow into the shaft piping.

Start-up of the system, after an unscheduled shutdown, would be with clear water and would begin at the segment of the system nearest the shaft. When clear water flow was established, slurry could be reintroduced. Because the hoisting system is not connected to the underground system, hoisting could resume as soon as the shaft and surface pipes were cleared.

The design for the hypothetical mine is based on an integrated hydraulic control system devised by Mr. George Ekert of Johnson Service Company of Denver. The system would be composed of a central panel located at the shaft station which would be connected by telephone cable to the various control points throughout the mine. The panel itself would be a hard-wired analog computer which would send and receive data from the control points continuously. Such a device for the mine described in this study could be installed for about $258,300.

The instruments and controls described in the following discussion are conventional and commercial items. For the most part, these devices have not seen service in underground coal mines, and a major program of systems research will be required before a workable system will be available.

5.7.1 Face and Secondary Haulage Subsystems

The following discussion is based upon the surge feed system. The operation would be similar for the direct feed system. Reference to Figure 13 will assist in following the description of the control system.

Initially, all lines in the face and secondary haulage subsystems are full of water. The slurry module is connected directly to the feeder-breaker in Room 1 of Section 1. Pump (A) would be started by the face operator. This action would also start the panel entry slurry pump (B), and the panel entry fresh water pump (C). It also closes normally open Valve No. 2 in the room occupied by the miner and opens normally closed Valves No. 1, allowing water flow into the operating face pump and slurry away from it. Valves No. 3 and 5 remain closed and Valve No. 4 remains open. In this configuration, the panel entry fresh water pump and the face and secondary haulage pumps simply circulate water in a closed loop through the face and panel areas.

Only two control functions are required at the face area. First, the pump feed hopper must not overflow, and, second, the slurry velocity

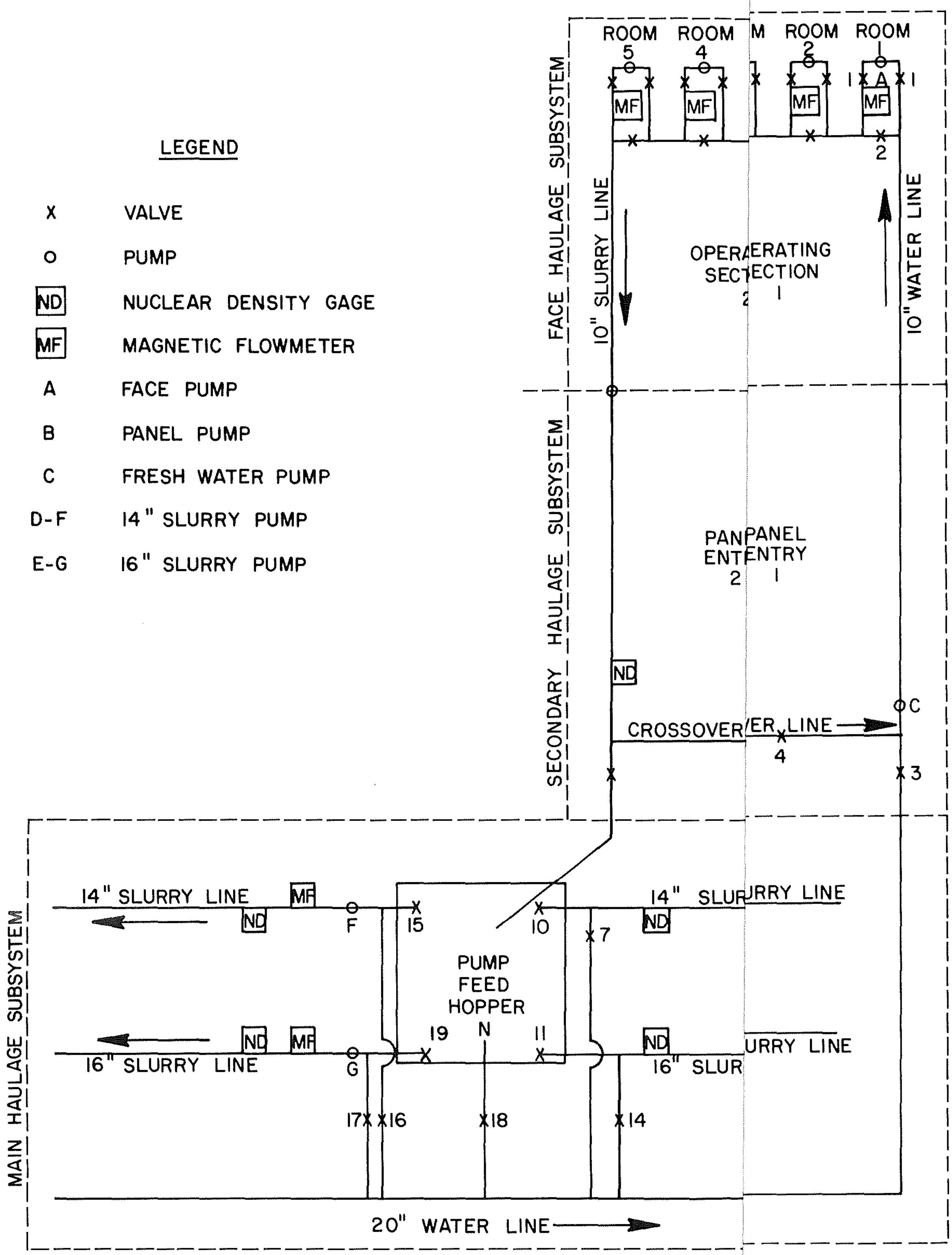


FIGURE 13 HY
DIA

must be controlled at 12 feet per second in the slurry pipe. Water coming into the hopper of the slurry module is modulated by an electrically operated ball valve on the waterline. Control of the valve is accomplished by two level indicators on the side of the hopper. When the water level is below the lower indicator, the valve is full open. When the water reaches the lower indicator, it partially closes, and when the upper level is reached, it fully closes. Normally, the valve would operate in the partially closed position which would be manually preset to deliver a quantity of water slightly less than the amount of water removed with the coal slurry.

A Fischer & Porter or equivalent magnetic flow meter (MF) would be installed just downstream from the outlet of the slurry pump in each slurry module. This device would sense the velocity of the fluid flowing in the line. Output from this meter would regulate another water valve feeding the second suction opening of the slurry pump. This valve would allow enough water into the pump to provide the appropriate velocity in the slurry line.

As coal from the miner enters the face pump through the slurry mixer, it pushes the clear water ahead of it in the slurry line through the panel entry pump toward the submain pipeline feed system. The slurry pump module in the secondary haulage system would be equipped with the same instrumentation as the one at the face. However, the quantity of coal-water slurry delivered would provide the needed velocity in the 10-inch secondary haulage line so not additional water would be required.

As the slurry reaches a point upstream from the intersection of the slurry line with the panel entry bypass line, a Kay-Ray or equivalent nuclear density gauge (ND) senses the increase in concentration and causes Valve No. 4 in the bypass line to close and Valve No. 3 in the fresh water line to open so that the fresh water pump (C) is supplied from the 20-inch-diameter submain entry fresh water line. The computer also causes Valve No. 5 to open, permitting discharge of the slurry into the main haulage system.

There is a possibility that this control function might be more easily controlled by monitoring secondary haulage pump amperage. As slurry flows down the secondary haulage pipe, the motor current would increase steadily until the entire line is full of slurry. The computer might be preset to change the valve settings at some predetermined current load.

When the continuous miner stops and coal is no longer being delivered to the slurry module, slurry flow ceases and clear water again reaches the nuclear density sensor. Valves No. 1 and 2 are returned to their starting positions. The operator may then stop the face pumps. The secondary haulage pump will not cease to operate until there is no coal in the pipeline at the density sensor. When the panel system is empty of coal, Valves No. 3 and 5 close and Valve No. 4 opens and Pumps B and C shut down.

5.7.2 Main Haulage Subsystem

As indicated earlier, the automatic hydraulic control system is the key to the successful operation of the transport system. This device continually monitors section operations and anticipates the demand for haulage capacity before the need arises. This arrangement divorces the operation of the face haulage and secondary haulage subsystems from that of the main haulage subsystem, permitting complete independence and flexibility of operation in the mining sections. The main haulage subsystem is designed to operate at maximum efficiency independently of the operation of individual sections. Slurry from any two operating sections is always directed into the 14-inch-diameter pipeline of the main haulage subsystem. Slurry from any three sections is always directed into the 16-inch slurry line. When only one section is operating, water is added to maintain sufficient velocity in the 14-inch pipe. This situation only occurs when one or four sections are operating.

Figure 13 shows only a small portion of the main haulage subsystem. This segment would be replicated whenever a panel entry or other new feed line enters the main haulage subsystem. In those areas of the subsystem where new feed is not introduced, such as auxiliary pump stations, a similar arrangement would be used, but the secondary haulage slurry lines shown in Figure 13 would not be present.

Instrumentation and operations of the main haulage subsystem are similar to those of the face and secondary haulage systems. The key to the operation is the removal of coal slurry from a common feed hopper for the two slurry pumps which feed the two slurry lines. As before, level control is achieved by two level detectors in the feed hopper. When the fluid level in the bin reaches the lower detector, the pump selected by the controller will start. If the level reaches the second indicator, the other pump will also start or the system will shut down the upstream units to prevent overflow.

Two examples will illustrate the operation of the face haulage subsystem described earlier. In the first example, assume no slurry is coming into Hopper M shown in Figure 13, except from Panel Entry 1. In other words, only one section is operating, and the main haulage subsystem is shut down.

The computer senses the slurry arriving at the nuclear density gauge in the panel entry and opens Valves No. 3 and No. 5 and closes Valve No. 4, allowing the slurry to discharge into Hopper M.

The computer also starts the 14-inch pump (D) and opens Valves No. 6 and 7. This allows water to circulate through the main haulage sector from the fresh water line, through the slurry line, and back into the fresh water line at the next pump station which is 850 to 2,600 feet away. A magnetic flow meter (MF) downstream from the slurry pump (D) senses the flow and informs the computer. At the same time, slurry from Panel Entry 1 enters Hopper M and reaches the lower of the two level devices. The computer than opens Valve 8 and closes Valve 6.

As slurry from Panel 1 enters Hopper M, the computer senses that only one section is supplying slurry to Hopper M and that additional water is necessary to provide adequate transport velocity. Additional water is added by opening preset Valve No. 9 which maintains the slurry level between the high and low level indicators. In the event that the slurry level rose above the upper indicator, the computer would shut down Section 1 and the related haulage systems to prevent overflow.

At the start-up of slurry Pump D, the 14-inch line was full of water. As the slurry enters Pump D from Hopper M, the nuclear density device just downstream from the pump senses this condition and informs the computer. As the slurry flows down the pipe, the nuclear density gauge upstream from Hopper N informs the computer which then closes Valve No. 7 and opens Valve No. 10, allowing the slurry to enter Hopper N. The computer will also start Pump F downstream from Hopper N and set the appropriate valves to keep the slurry moving out of the mine.

When the slurry flow from Panel Entry 1 stops, the slurry level will again drop below the lower sensor in Hopper M, and the computer will open Valve No. 6 full to maintain the slurry velocity in the pipe. The nuclear density gauge at Pump D detects the lack of coal and informs the computer, which then closes Valves 8 and 9. As the clear water reaches the nuclear density gauge at Hopper N, Valve No. 7 is opened and Valve No. 10 is closed. The main haulage segment is now full of water and Pump D shuts down.

A second example assumes that two sections are feeding coal through the 14-inch line at Hopper M and that Panel Entry 1 then begins to deliver coal slurry to the hopper. At this point, Valves No. 6, 7, 9, 11, 12, 13, and 14 are closed. Valves No. 8 and 10 are open, and Pumps D and F are moving the slurry through the 14-inch line.

As coal slurry moves down the panel entry pipe from Section 1, the nuclear density gauge detects the slurry and informs the computer which closes Valve No. 4 and opens Valves No. 3 and 5. The slurry begins to flow into Hopper M, and the level rises to the upper level indicator. The computer senses the increased flow and starts Pump E and opens Valves 12 and 14. With both pumps operating, the slurry level falls rapidly, and the computer senses this through the bin indicators. With three sections operating, the computer wants to shut down the 14-inch line which is full of slurry. To do this, slurry Valve No. 8 closes and water Valve No. 6 opens, forcing fresh water into the 14-inch slurry pipe. The fresh water is then pumped down the 14-inch slurry line until the nuclear density gauges detect that the line contains only water. This line is then shut down as before.

When the computer started up Pump E, it also opened Valves No. 12 and 14. Valves No. 11 and 13 were closed and Valve No. 15 was open. As the slurry flows through the nuclear density gauge at Pump E, the computer realizes that slurry is now flowing down the 16-inch line. When the coal slurry reaches the nuclear density gauge at Hopper N, Valve No. 11 opens, while Valve No. 14 closes, allowing the slurry to enter the hopper. The computer sensing the arrival of the slurry through the level indicators starts the next 16-inch pump and continues the slurry movement down the line. The shutdown sequence is repeated for the 14-inch line downstream from Hopper N and at each subsequent hopper until only the 16-inch line is carrying slurry.

5.8 REJECTED CONCEPTS

Several concepts were developed and then rejected in the course of the project. These are mentioned here, briefly, in the hope that they have merit and that they may present other investigators with a starting point for development of successful solids feeders.

Two types of slurry feeders were considered. The first of these would add the dry coal solids into a moving stream of high-pressure water. The advantage of such a system is that a pump would not be required at the coal face, and the high-pressure water could move the slurry to the end of the panel entry where it would be introduced into the main haulage system.

This approach was abandoned for the face haulage subsystem for two reasons. First, the high pressure hoses would be extremely difficult to handle, and they could be a considerable hazard to personnel in the event of a water hammer or rupture. Second, only two systems, the lock hopper and the Hydrohoist, appear to be capable of injecting the coal into the high-pressure line. Both of these systems are much too large for a face installation.

The second group of slurry feeders would mix the solids and water, and introduce them into the suction side of a centrifugal pump. The pump would pick the slurry up at atmospheric pressures and inject it into the pipe for transport.

In an effort to control the rate of introduction of solids into the pipeline, a rock pump was examined. This system is illustrated in Figure 14. The coal from the miner drops into a small hopper. From this hopper, the dry coal drops into a 14-inch square chamber. A double acting, compressed-air-operated piston forces this coal into a tube connecting the chamber to the pump suction. Fresh water is injected into the side of the tube near the pump in the direction of flow so that the

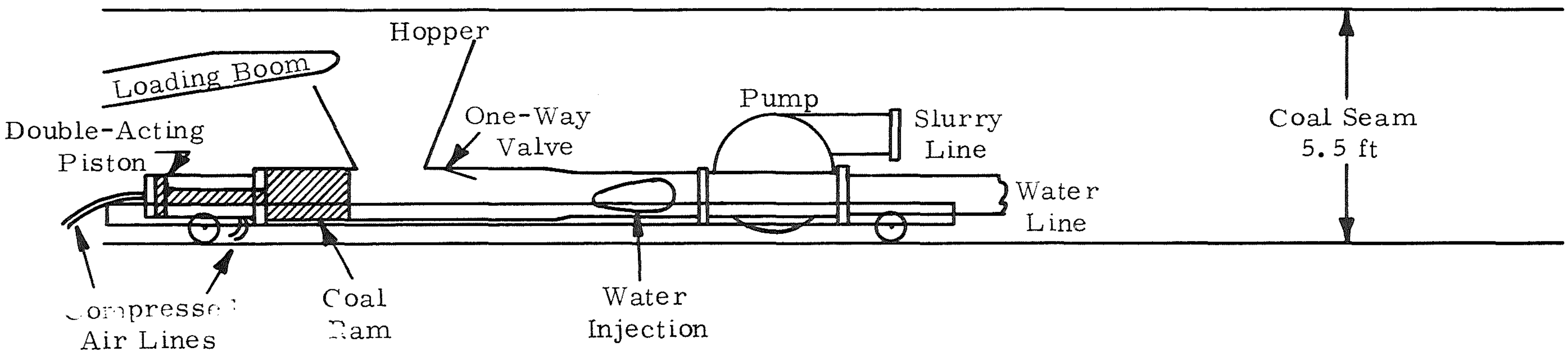


FIGURE 14

Pump Feed System
Reciprocating Piston Type

(1/4 in. = 1 ft)

pump suction is supplied with a metered coal and water mix. This system has the potential advantage of lower maintenance costs and less spillage than the system finally selected. However, the pump must cycle once a second to move four tons of coal per minute, and compressed air would be required at the face. Possible difficulties may arise in this area as a result of coal packing in the pump suction.

The initial concept for face area transportation involved the use of telescoping sections of pipe behind the continuous miner. Movement of the machine would be possible through extension and retraction of the telescoped sections of pipe. Certain apparent disadvantages of the system appeared serious enough to discourage in-depth investigation of the idea. First, it would require rigid pipelines in each opening, making the movement of auxiliary equipment in the area difficult. Second, flexibility of the pipeline to negotiate vertical variations in the coal seam and for lateral movement through crosscuts would be extremely limited. Third, leakage and wear in the telescoping sections appeared to constitute serious operating and cost difficulties. Finally, no domestic manufacturer was prepared to quote costs and specifications for application considered in this study.

6.0 CAPITAL COSTS

Detailed schedules of equipment for the conventional and hydraulic transportation mines will be found in Appendix, Exhibits 2 and 3, respectively. These data are summarized in Tables 6-1, which shows investments for both hydraulic and conventional face haulage equipment, Table 6-2, which summarizes investment for the main and secondary haulage, and Table 6-3, which summarizes costs for shafts and surface facilities.

Surface and shaft costs for equipment and facilities constitute the largest single portion of initial investment expenditures. Table 6-4 summarizes the equipment required and the total investment for each of the five shaft depths specified in the contract for both conventional haulage and hydraulic hoisting. Costs shown cover the delivered equipment and its installation based upon January 1974 prices.

Shaft sizes and hoisting facilities are, of course, not the same for hydraulic hoisting as they are for conventional. The former requires a 14-foot-diameter hoisting shaft to accommodate the man and materials cage, the hydraulic pipeline, the mine drainage line, and the slurry make-up waterline. It will provide sufficient room for movement of machines and equipment without damage to the pipelines. This shaft will also have a head frame for the man and materials hoist.

For the conventional system, the production shaft is 16 feet in diameter and has been compartmented. One compartment will accommodate two skips for balanced hoisting of coal, the other will contain a man and materials cage similar to the one used in the hydraulic mine.

TABLE 6-1

Estimated Face Area Capital Costs

(Thousands of Dollars)

Life in Years	Description	Unit Cost	Hydraulic Systems: Direct Feed System: Total hp	Initial Investment	Number of Units	Surge Feed System: Total hp	Initial Investment	Number of Units	Conventional System (Miner, 2 Shuttle Cars, Feeder-Breaker): Total hp	Initial Investment	Number of Units
8	Continuous Miner	194.0-214.0	4,235[1]	1,498.0[2]	7	3,710	1,358.0	7	3,710	1,358.0	7
8	Roof Bolter	15.0	560	210.0	14	560	210.0	14	280	105.0	7
8	Feeder Breaker	37.2	--	--	--	560	260.4	7	560	260.4	7
8	Shuttle Car	54.0	--	--	--	525	378.0	7	1,050	756.0	14
8	Rockduster	14.0	105	98.0	7	105	98.0	7	105	98.0	7
8	Mobile Shop	0.9	--	6.3	7	--	6.3	7	--	6.3	7
8	Telephones	0.3	--	2.1	7	--	2.1	7	--	2.1	7
8	Spare Equipment	300.2-320.2	--	320.2	--	--	300.2	--	--	300.2	--
8	1,000-kva Transformer (combination)	25.0	--	175.0	7	--	175.0	7	--	175.0	7
8	Pump Feed System	4.0	300	120.0	30	300	120.0	30	--	--	--
8	Scoop Tram	35.0	--	245.0	7	--	245.0	7	--	245.0	7
8	Lube Cars	8.5	--	59.5	7	--	59.5	7	--	59.5	7
8	Battery Chargers	3.4	--	27.2	8	--	27.2	8	--	27.2	8
8	10-in. Centrifugal Pumps	14.6	6,000	438.0	30	6,000	438.0	30	--	--	--
8	Skids for Slurry Hose	0.5	--	84.0	168	--	84.0	168	--	--	--
8	Winch for Moving Hose	1.0	140	28.0	28	140	28.0	28	--	--	--
	Subtotals -- 8-yr Equipment		11,340	3,311.3		11,900	3,789.7		5,705	3,392.7	
	Contingency at 10%			331.1			379.0			339.3	
	Estimated Capital Investment 8-yr Facilities			3,642.4			4,168.7			3,732.0	
5	2-in. Water Lines	0.00108/ft	--	3.8	500 x 7(ft)	--	3.8	500 x 7(ft)	--	3.8	500 x 7(ft)
5	4-in. Drainage Line	0.00150/ft	--	4.2	400 x 7(ft)	--	4.2	400 x 7(ft)	--	4.2	400 x 7(ft)
5	10-in. Hose	0.05181/ft	--	290.1	800 x 7(ft)	--	290.1	800 x 7(ft)	--	--	--
5	10-in. Valves	2.4	--	218.4	91	--	218.4	91	--	--	--
5	10-in. Pipe	0.01200/ft	--	184.8	15,400 ft	--	151.2	12,600 ft	--	--	--
	Subtotals -- 5-yr Equipment			701.3			667.7			8.0	
	Contingency -- 10%			70.1			66.8			0.8	
	Estimated Capital Investment 5-yr Facilities			771.4			734.5			8.8	
	Grand Totals for 8- and 5-yr Life Items Above		11,340	4,413.8		11,900	4,903.2		5,705	3,740.8	

(1) Higher horsepower is with built-in breaker.
(2) Higher cost is with built-in breaker.

Estimated Installed Facility Costs for C

Main and Seconda

Life in Years	Description	Unit Cost	All Hydraulic System Total hp	All Hydraulic System Initial Investment	All Hydraulic System Number of Units	Conventional Total hp	Conventional Int	Conventional Number of Units
15	500-kw Rectifier	17.0	--	17.0	1	--		1
15	300-kva Transformer	8.9	--	124.6	14	--		11
15	Electric Cable -- Main Entry	0.008/ft	--	90.0	11,250	--		11,250
15	Entry Rockduster	18.0	15	18.0	1	15		1
15	Mine Track -- 60 lb	0.006/ft	--	67.5	11,250	--		11,250
15	16-in. Pumps	56.5	4,000	226.0	4	4,000		--
15	150-kva Transformer	7.5	--	60.0	8	--		--
15	500-kva Transformer	11.0	--	22.0	2	--		--
15	16-in. Pumps	29.1	1,050	87.3	3	1,050		--
15	14-in. Pumps	20.0	320	80.0	4	320		--
15	14-in. Pumps	31.8	2,800	127.2	4	2,800		--
15	1,000-kva Transformer	25.0	--	275.0	11	--		--
15	10-in. Pumps	19.1	2,400	152.8	8	2,400		--
15	20-in. Pumps	30.5	900	91.5	3	900		--
15	Integrated Hydraulic Control System	258.3	--	258.3	1	--		--
	Plus Engineering at 18% (Control System)	--	--	46.5	--	--		
	Subtotal -- 15-yr Equipment		11,470	1,743.7		11,470		
	10% Contingency			174.4				
	Total -- 15-yr Equipment			1,918.1				
10	Belt Winders	3.5	--	--	--	--		2
10	Single Breakers	5.5	--	82.5	15	--		6
10	Crossbelt Conveyor -- 500 ft	22.6	--	--	--	--		2
10	ME Belt Conveyor -- 42 in. x 4,000 ft	141.8	--	--	--	--		3
10	Double Breakers	8.0	--	112.0	14	--		7
10	Supply Cars	2.8	--	100.8	36	--		36
10	Personnel Carriers	10.0	50	50.0	5	50		5
10	Portal Buses	15.6	120	93.6	6	120		6
10	Electric Cable (Panel) 1/0 3 c	0.006/ft	--	126.0	21,000	--		21,000
10	40-lb Track	0.004/ft	--	84.0	21,000	--		21,000
10	Belt Conveyor -- 36 in. x 3,000 ft (100 hp)	98.6	--	--	--	--		7
10	Trolley Wire	0.00170/ft	--	19.1	11,250	--		11,250
10	4-in. Water Pipe	0.00288/ft	--	32.4	11,250	--		11,250
10	4-in. Drainage Pipe	0.00150/ft	--	48.4	32,250	--		32,250
10	10-in. Slurry Pumps	11.4	800	91.2	8	800		--
10	Pump Feed System	4.0	240	96.0	24	240		--
10	Telephones	0.3	--	2.4	8	--		8
10	Equipment Carriers	2.3	--	4.6	2	--		2
	Subtotal -- 10-yr Equipment		1,210	943.0		1,210		
	10% Contingency			94.3				
	Total -- 10-yr Equipment			1,037.3				
5	14-in. Valves	3.8		106.4	28			--
5	16-in. Valves	4.5		135.0	30			--
5	20-in. Valves	6.1		36.6	6			--
5	14-in. Pipe	0.01513/ft		170.2	11,250			--
5	16-in. Pipe	0.01772/ft		199.4	11,250			--
5	20-in. Pipe	0.02372/ft		266.9	11,250			--
5	2-in. Water Line -- 21,000 ft	0.00108/ft		22.7	21,000			21,000
5	10-in. Pipe	0.012/ft		300.0	25,000			--
5	10-in. Valves	2.4		62.4	29			--
	Subtotal -- 5-yr Equipment			1,299.6				
	10% Contingency			130.0				
	Total -- 5-yr Equipment			1,429.6				
	Grand Totals for 5-, 10-, and 15-yr Life Items Above		12,680	4,385.0		12,680		

Estimated Installed F

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Life in Years	No. of Items	Description	Unit Cost	300-ft Depth Hydraulic hp	300-ft Depth Hydraulic Cost	300-ft Depth Conventio hp	300-ft Depth Conventio Cost	2,000-ft Depth Hydraulic hp	2,000-ft Depth Hydraulic Cost	2,000-ft Depth Conventional hp	2,000-ft Depth Conventional Cost
20	1	Preparation Plant -- Conventional, 300 tons/hr; Hydraulic, 400 tons/hr	985.0-1,120.0	1,100	1,720.0	1,000	1,5	100	1,720.0	1,000	1,585.0
20	1	Clean Coal Storage Silo -- 3,000 tons	200.0	--	200.0	--	2	--	200.0	--	200.0
20	1	Clean Coal Storage Conveyor -- 500 ft	0.2/ft	100	100.0	100	1	100	100.0	100	100.0
20	1	Raw Coal Feed Conveyor -- 500 ft	0.2/ft	100	100.0	100	1	100	100.0	100	100.0
20	1	Raw Coal Storage Silo -- 3,000 tons	200.0	--	200.0	--	2	--	200.0	--	200.0
20	8	6-ft Wide D.S.M. Screens	7.3	--	58.3	--		--	58.3	--	--
20	12	4-ft x 8-ft Dewatering Screens	7.275	10	36.2	--		10	36.2	--	--
20	1	60-ft x 12-ft Thickener and Diaphragm Pump	82.6	5	82.6	--		5	82.6	--	--
20	1	Raw Coal Storage Conveyor -- 500 ft	0.2/ft	100	100.0	--		100	100.0	--	--
20	1	Refuse Bin -- 50-ton Bin and Conveyor	12.0	25	12.0	25		25	12.0	25	12.0
20	1	Slurry Pond	131.6	--	131.6	--	1	--	131.6	--	131.6
20	1	Fresh Water Supply System	300.0-100.0	100	300.0	50	1	100	300.0	50	100.0
20	1	Main Building -- 60 ft x 240 ft	0.017/sq ft	--	250.0	--	2	--	250.0	--	250.0
20	1	Roads and Yard Improvements	75.0	--	75.0	--		--	75.0	--	75.0
20	1	Locomotive (Supply) -- 10-ton Gasoline Powered	25.0	--	25.0	--		--	25.0	--	25.0
20	1	Water and Sewage System	50.0	--	50.0	--		--	50.0	--	50.0
20	1	Main Substation (Underground) -- 69,000 v to 7,200 v; 20,000 kva and 7,500 kva	278.0-220.0	--	278.0	--	2	--	278.0	--	220.0
20	1	Surface Substation -- 69,000 v to 4,160 v to 440 v; 3,500 kva and 2,500 kva	174.0-150.0	--	174.0	--	1	--	174.0	--	150.0
20	1	1-Mile Powerline -- 69 kv	20.0	--	20.0	--		--	20.0	--	20.0
20	1	Main Fan -- 350,000 cfm	120.0-135.0	600	120.0	600	1	00	135.0	900	135.0
20	1	Production Shaft -- 14-ft diam or 16-ft diam	1.06-1.4/ft	--	518.0	--	5	--	2,234.4	--	2,415.0
20	1	Surface Electrical Distribution System	30.0	--	30.0	--		--	30.0	--	30.0
20	2	Ventilation Shafts	0.325-0.4/ft	--	240.0	--	2	--	1,300.0	--	1,300.0
20	1	Service Hoist Including Wire Rope	508.0-543.0	700	543.0	700	5	700	518.0	700	518.0
20	1	Service Hoist Building	40.0-50.0	--	50.0	--		--	40.0	--	40.0
20	1	Service Cage	5.0	--	5.0	--		--	5.0	--	5.0
20	--	Office Furniture	--	--	7.0	--		--	7.0	--	7.0
20	1	Hoist, Emergency	35.0	40	35.0	40		40	35.0	40	35.0
20	1	Bulk Oil System	20.0-30.0	--	20.0	--		--	30.0	--	30.0
20	1	Bulk Rock Dust System	20.0-30.0	--	20.0	--		--	30.0	--	30.0
20	1	Main Circuit Breaker	11.0	--	11.0	--		--	11.0	--	11.0
20	1	Triple Breaker	9.5	--	9.5	--		--	9.5	--	9.5
20	1	Underground Shop	40.0	--	40.0	--		--	40.0	--	40.0
20	--	Mine Track Shaft Bottom -- 7,000 ft	6.00/ft	--	42.0	--		--	42.0	--	42.0
20	--	Trolley Wire Shaft Bottom -- 3,000 ft	1.7/1,000 ft	--	5.1	--		--	5.1	--	5.1
20	2	Locomotives -- Trolley and Battery	40.0	160	80.0	160		60	80.0	160	80.0
20	--	Shaft Pumping Equipment	3.0-13.3	30	3.0	30		50	13.3	250	13.3
20	1	Hydrohoist(1)(2)	2,629.6-3,927.8	1,500	161.3	--		500	3,927.8	--	--
20	1	Shaft Pocket -- Complete	0.1/cu yd	--	175.0	--	1	--	175.0	--	100.0
20	1	Underground Sump for Slurry Water	--	--	5.0	--		--	5.0	--	5.0
20	1	Sump Cleaning and Recovery Equipment	20.0-47.0	25	20.0	--		200	47.0	--	--
20	1	Underground Pump Room	--	--	750.0	--		--	750.0	--	--
20	1	Overland 14-in. Line or Conveyor -- 3,000 ft	60.0-132.0	--	60.0	200	1	--	60.0	200	132.0
20	--	Clean Coal Sampling Facility and Belt Scale	--	10	44.0	10		10	44.0	10	44.0
20	--	Shop Equipment (Surface)	--	50	30.0	50		50	30.0	50	30.0
20	--	Shop Equipment (Underground)	--	50	25.0	50		50	25.0	50	25.0
20	--	Development Costs (Surface)	--	--	75.0	--		--	115.0	--	115.0
20	--	Development Costs (Underground)	--	--	50.0	--		--	50.0	--	50.0
20	--	Hydraulic Pipe (Shaft) -- 14 in.	0.020/ft	--	8.0	--		--	42.0	--	--
20	1	Service Hoist Headframe	175.0	--	175.0	--	1	--	175.0	--	175.0
20	--	Electric Cable (Shaft Bottom) -- 3,000 ft	0.008/ft	--	24.0	--		--	24.0	--	24.0
20	1	500-kva Transformer -- Shaft Bottom	11.0	--	11.0	--		--	--	--	11.0
20	1	300-kva Transformer -- Shaft Bottom	8.9	--	8.9	--		--	8.9	--	--
20	1	1,000-kva Transformer -- Shaft Bottom	25.0	--	--	--		--	25.0	--	--
20	1	Single Breaker	5.5	--	5.5	--		--	5.5	--	5.5
20	5	Telephones -- Shaft Bottom	0.3	--	1.5	--		--	1.5	--	1.5
20	1	Production Shaft Headframe	250.0	--	--	--	2	--	--	--	250.0
20	1	Production Hoist	294.0-394.0	--	--	300	2	--	--	1,000	394.0
20	2	Skips	2.5-3.5	--	--	--		--	--	--	7.0
20	1	Production Hoist Building	--	--	--	--		--	--	--	40.0
20	1	Double Breaker	8.0	--	--	--		--	--	--	8.0
		Subtotals -- 20-yr Equipment		4,705	7,320.5	3,415	6,3	400	13,988.7	4,635	9,381.5
		Subtotals -- Engineered Facilities		--	6,890.0	--	5,8	--	13,493.7	--	8,911.9
		Engineering at 18%		--	1,240.2	--	1,0	--	2,428.8	--	1,604.1
		Subtotal + Engineering		--	8,560.7	--	7,3	--	16,417.5	--	10,985.6
		10% Contingency		--	856.1	--	7	--	1,641.8	--	1,098.6
		Estimated Capital Investment -- 20-yr Facilities		--	9,416.8	--	8,1	--	18,059.3	--	12,084.2
5	1	Refuse Truck		--	80.0	--		--	80.0	--	80.0
5	--	Mobile Equipment		--	223.0	--	2	--	223.0	--	223.0
5	--	Safety Equipment		--	30.3	--		--	30.3	--	30.3
		Subtotals -- 5-yr Equipment			333.3		3		333.3		333.3
		10% Contingency			33.3				33.3		33.3
		Estimated Capital Investment -- 5-yr Facilities			366.6		3		366.6		366.6
3	--	Cars and Trucks		--	20.5	--		--	20.5	--	20.5
		10% Contingency			2.1				2.1		2.1
		Estimated Capital Investment -- 3-yr Facilities			22.6				22.6		22.6
		Total Estimated Capital Investment for 10-, 5-, and 3-yr Life Items Above			9,806.0		8,4		18,448.5		12,473.4

(1) The horsepower shown for each hydrohoist installation is total installed for three units of which one would be a spare. The horsepower would be two-thirds of each figure shown.

(2) Warman Pumps Used in 300- and 500-foot Shafts

Total investment costs for the two-mile main haulage mine are summarized in Table 6-4. In addition to the capital costs shown in Tables 6-1, 2, and 3, miscellaneous investment costs are shown. These include investment charges for development, drilling, land, coal, working capital, and freight.

Development drilling charges have been assumed for test drilling to prove the coal reserves needed to justify engineering and construction. Costs for land include payment for ownership of sufficient surface acreage to accommodate the surface plant and right-of-way for conveyors and power lines. The charge for coal is the amount spent to hold an option on the reserve prior to the start of construction. Working capital has been charged as 10% of the delivered equipment cost. Freight charges for delivery of equipment to the mine site are assumed to be five percent of the Total Facility Cost.

Costs for main haulage will vary with the distance the system must transport the coal. Capital costs shown in Table 6-2 are for a two-mile main haulage system. A more detailed comparison of main haulage costs will be found in Table 6-5. This table lists the equipment required, the haulage system to which it applies, and the cost per foot of advance of the entry for both conventional and hydraulic systems.

Conventional haulage as projected is accomplished with 42-inch rope belt conveyors, each 4,000 feet in length, powered by 200-hp motors operating at 600 feet per minute belt speeds. Costs are for complete conveyors including fire and dust control devices. Each main entry conveyor drive requires a 300-kva transformer and one single-circuit breaker.

TABLE 6-4

Summary of Total Initial Capital Investment
Two Mile Main Haulage
(Thousands of Dollars)

Face Systems	Depth of Cover 300 ft	500 ft	1,000 ft	1,500 ft	2,000 ft
CONVENTIONAL MINE					
Miner, Shuttle Car, Feeder-Breaker, Belt Conveyor					
Face Haulage	3,740.8	3,740.8	3,740.8	3,740.8	3,740.8
Secondary & Main Haulage	2,344.3	2,344.3	2,344.3	2,344.3	2,344.3
Shaft and Surface	8,499.9	9,012.9	10,232.3	11,487.9	12,473.4
Miscellaneous	2,355.8	2,444.9	2,657.9	2,876.6	3,053.2
Total Initial Investment	16,940.8	17,542.9	18,975.3	20,449.6	21,611.7
HYDRAULIC MINE					
Direct Feed to Pump					
Face Haulage	4,413.8	4,413.8	4,413.8	4,413.8	4,413.8
Secondary & Main Haulage	4,385.0	4,385.0	4,385.0	4,385.0	4,385.0
Shaft and Surface	9,806.0	9,848.5	15,615.0	17,105.4	18,448.5
Miscellaneous	3,475.4	3,615.2	3,912.9	4,167.8	4,400.1
Total Initial Investment	22,080.2	22,262.5	28,326.7	30,072.0	31,647.4
Surge Feed to Pump					
Face Haulage	4,903.2	4,903.2	4,903.2	4,903.2	4,903.2
Secondary & Main Haulage	4,385.0	4,385.0	4,385.0	4,385.0	4,385.0
Shaft and Surface	9,806.0	9,848.5	15,615.0	17,105.4	18,448.5
Miscellaneous	3,551.3	3,690.9	3,988.7	4,243.7	4,475.9
Total Initial Investment	22,645.5	22,827.6	28,891.9	30,637.3	32,212.6

TABLE 6-5

Main Entry Extension Costs 4,000-ft Increments

(Thousands of Dollars)

	Conventional	Hydraulic
Belt Conveyor - 42 in. x 4,000 ft	$150.0	$ --
Electric Distribution Cable - 250,000 CM 3 c 8,000 v	28.0	28.0
4-in. Fresh Water Pipe	11.5	11.5
4-in. Drainage Pipe	6.0	6.0
60-lb. Mine Track	24.0	24.0
Trolley Wire	6.8	6.8
16-in. Pipe	--	70.9
14-in Pipe	--	60.5
1200 hp, 16-in. Pump - 1 per 2,800 ft.	--	43.0
350 hp, 14-in. Pump - 1 per 2,800 ft	--	27.6
300 hp, 20-in. Pump - 1 per 4,500 ft	--	27.5
20-in. Pipe	--	94.9
Instrument and Controls	--	31.1
20-in. Valve - 1 per 2,800 ft	--	6.1
16-in. Valve - 4 per 2,800 ft	--	18.0
14-in. Valve - 4 per 2,800 ft	--	15.2
Pump Feed System - 1 per 2,800 ft	--	16.0
300-kva Transformers - 1	8.4	--
Single Circuit Breakers - 1	5.5	--
Double Circuit Breakers - 4	--	32.0
150-kva Transformers - 2	--	15.0
1,000-kva Transformers - 2	--	36.0
1/2 500-kw Rectifier	7.0	7.0
Total	$247.2	$577.1
	÷4,000 ft	÷4,000 ft
Cost per foot of Entry Advance	$ 61.80	$144.28

Also included is equipment common to both haulage systems. This includes a four-inch-diameter fresh water steel pipe and a drainage line of four-inch-diameter plastic pipe. Mine track is 60-lb. rail with a 42-inch gauge laid on a combination of steel and wooden ties. Not shown are two 10-ton, combination trolley-battery locomotives to transport supplies to the working areas in 36 rail-rubber supply cars. Mechanics and mine supervisors are transported throughout the mine on small battery-operated personnel carriers while workmen travel to their work areas in 12- to 16-man battery-powered, portal buses. Costs for these items will be found in Table 6-2.

Hydraulic haulage is based upon the two pipes of 14- and 16-inch-diameter. Appropriate pumps, controls and transfer hoppers are included at 2,800-foot intervals.

As shown in Table 6.5, the capital costs for hydraulic transportation is considerably higher than for conventional haulage. Costs for the pipe alone (excluding pumps and auxiliary equipment) exceed those for a complete belt conveyor, including the drive and motor. Power requirements are roughly 900% higher for hydraulic transport in this area. Capital costs for extension of hydraulic and conventional main haulage systems are shown in Table 6-6. Capital investments for each of the mines are summarized in Table 6-7.

TABLE 6-6

Entry Extension Costs for Conventional and Hydraulic Main Haulage Systems

Distance	Conventional	Hydraulic
1 mile	$ 326,304	$ 761,772
2 miles	652,608	1,523,544
4 miles	1,305,216	3,047,088
8 miles	2,610,432	6,094,176

TABLE 6-7

Summary of Total Capital Investments
(Thousands of Dollars)

Length of Main Haulage	Depth of Cover				
	300	500	1000	1500	2000
Conventional Mine					
1 mile	16,614.5	17,180.6	18,649.0	20,123.3	21,285.4
2 miles	16,940.8	17,542.9	18,975.3	20,449.6	21,611.7
4 miles	17,593.4	18,159.5	19,627.9	21,102.2	22,264.3
8 miles	18,898.6	19,464.7	20,933.1	22,407.4	23,569.5
Hydraulic Mine - Direct Feed System					
1 mile	21,318.4	21,500.7	27,564.9	29,310.2	30,885.6
2 miles	22,080.2	22,262.5	28,326.7	30,072.0	31,647.4
4 miles	23,603.7	23,786.0	29,850.2	31,595.5	33,070.9
8 miles	26,650.8	26,833.1	32,897.3	34,642.6	36,118.0
Hydraulic Mine - Surge Feed System					
1 mile	21,883.7	22,065.8	28,130.1	29,875.5	31,450.8
2 miles	22,645.5	22,827.6	28,891.9	30.637.3	32,212.6
4 miles	24,169.0	24,351.1	30,415.4	32,160.8	33,736.1
8 miles	27,216.1	27,398.2	33,462.5	35,207.9	36,783.2

7.0 OPERATING COSTS

Operating costs consist of labor, and materials and supplies. Of these, labor is the larger and represents the largest single cost in coal production.

7.1 LABOR COSTS

Both the conventional and hydraulic haulage mines will operate three shifts per day, 200 days per year. An additional 43 week days are available, but these are assumed to be lost due to strike and/or maintenance activities.

7.1.1 Labor Costs for Conventional Haulage Mine

Manpower requirements for the conventional haulage mine operating section are shown in Table 7.1-1. Since five sections would operate each shift, the total number of section employees on the payroll would be 140.

Table 7.1-1

Summary of Section Manpower for Conventional Transportation Mine

Work Classification	1st Shift	2nd Shift	3rd Shift	Total
Section Foreman	1	1	1	3
Continuous Miner Operator	1	1	1	3
Continuous Miner Operator Helper	1	1	1	3
Roof Bolter Operator	1	1	1	3
Shuttle Car Operator	2	2	2	6
Clean-up and Supply	1	1	1	3
Ventilation	1	1	1	3
Rockdust	-	-	1	1
Mechanic	1	1	1	3
Total	9	9	10	28

The total mine payroll would include 335 personnel. Of these, 306 would be present each day. A detailed labor analysis is shown in Appendix,

Exhibit 4. This analysis includes the number of personnel in each category and the annual payroll costs.

The labor data shown are for the mines having one and two mile long main haulage systems. As the main haulage system extends beyond two miles to four or eight miles, one additonal belt repairman will be required. An additional two belt patrolmen will be required on each of the three operating shifts for each additional mile of main haulage extension beyond the two miles.

Table 7.1-2 summarizes these costs for work days, idle days, and holidays. Also shown are the total annual man days, labor costs, productivity in tons per man shift, and labor cost.

For the mine developed in this study, annual production is 1,104,150 tons of clean coal. Coal is produced at the rate of 14.17 tons per man shift, and the annual labor cost is $3.59 per ton.

7.1.2 Labor Costs for Hydraulic Haulage Mine

Manpower requirements for the hydraulic haulage mine are somewhat greater than those for the conventional haulage mine. Section manpower is summarized in Table 7.1-3. Much of this additional labor is involved in roof support and piping changes.

Table 7.1-2

Conventional Mining System
Manpower and Productivity

Daily Costs

	Work Day	Idle Day	Holiday
Total Straight Time Day	$14,505.19	$9,093.74	$14,505.19
Necessary O.T., 10% (U)	1,166.93	625.79	1,166.93
Company Men at 7%	198.51	198.51	198.51
Men Working			396.60
Total	$15,870.63	$9,918.04	$16,267.23

Personnel on Payroll

Union	-- 254 x 1.1	=	279
Company	-- 52 x 1.07	=	56
Total Payroll		=	335

Productivity and Annual Labor Costs

			Man Days	$/Year
Production Day	-- 335 men x 200 days	=	67,000	$3,174,126.00
Saturdays	-- 153 men x 9 days	=	1,377	89,302.77
Idle Week days	-- 187 men x 34 days	=	6,358	337,213.36
Idle Week days	-- 67 men x 9 days	=	603	32,006.16
Idle Saturdays	-- 55 men x 43 days	=	2,365	130,446.52
Sundays	-- 4 men x 52 days	=	208	20,623.20
Holidays	-- 4 men x 9 days	=	36	146,405.07
Company Vacation --		=		28,358.90
Total			77,943	$3,958,481.98

$$\frac{\text{Annual Tons}}{\text{Total Man Days}} = \frac{1,104,150}{77,943} = 14.17 \text{ T/M/D Clean Coal}$$

$$\frac{\text{Annual \$}}{\text{Annual Tons}} = \frac{3,958,481.98}{1,104,150} = 3.59 \text{ \$/Ton}$$

Table 7.1-3

Summary of Section Manpower for Hydraulic Transportation Mine

Work Classification	1st Shift	2nd Shift	3rd Shift	Total
Foreman	1	1	1	3
Continuous Miner Operator	1	1	1	3
Continuous Miner Operator Helper	1	1	1	3
Roof Bolter Operator	2	2	2	6
Move Feeders	1	1	1	3
Ventilation	1	1	1	3
Spray Water and Rock Dust	1	1	1	3
Pipeman	1	1	1	3
Clean-up and Supply	1	1	1	3
Mechanic	1	1	1	3
Total	11	11	11	33

As with the conventional haulage mine, five sections would operate each shift resulting in a total of 165 men active in coal production. Total mine payroll is 373 men of which 341 would be present on any work day. This is an 11% increase in total payroll over the conventional haulage system mine.

Appendix, Exhibit 5 provides a detailed analysis of payroll occupations and costs for the hydraulic mines with one and two mile main haulage systems. These data are summarized in Table 7.1-4. From this table, it will be seen that annual labor costs for the hydraulic transportation mine are $4,173,314.50.

Production for the surge feed hydraulic system is 1,672,800 tons of clean coal per year. With this system, the labor productivity is 19.54 tons per man shift, and the labor cost is $2.49 per ton. This is more than a dollar per ton less than the conventional mine cost.

Production for the direct feed hydraulic system is 1,484,100 tons of clean coal per year. The labor productivity is 17.34 tons per

Table 7.1-4

Hydraulic Mining Systems
Manpower and Productivity

Daily Costs

	Work Day	Idle Day	Holiday
Total Straight Time Day	$15,273.99	$ 9,917.74	$15,273.99
Necessary O.T. at 10% (U)	1,226.31	690.69	1,226.31
Company Men at 7%	210.76	210.76	210.76
Men Working			396.60
Total	$16,711.06	$10,819.19	$17,107.66

Personnel on Payroll

Union	-- 284 x 1.10 =	312
Company	-- 57 x 1.07 =	61
Total Payroll		373

Productivity and Annual Labor Costs

			Man Days	$/Year
Production Day	--	373 men x 200 days =	74,600	$3,342,212.00
Saturdays	--	161 men x 9 days =	1,449	92,243.97
Idle Week Days	--	206 men x 30 days =	6,180	367,852.46
Idle Week Days	--	60 men x 9 days =	540	28,290.51
Idle Saturdays	--	60 men x 43 days =	2,580	138,014.52
Sundays	--	4 men x 52 days =	208	20,623.20
Holidays	--	4 men x 8 days =	32	153,968.94
Company Vacation	--			30,108.90
Total			85,589	$4,173,314.50

Surge Feed System

$$\frac{\text{Annuations}}{\text{Total Man Days}} = \frac{1,672,800}{85589} = 19.54 \text{ T/M/D Clean Coal}$$

$$\frac{\text{Annual \$}}{\text{Annual Tons}} = \frac{4,173,314.50}{1,672,800} = 2.49 \text{ \$/T}$$

Direct Feed System

$$\frac{\text{Annual Tons}}{\text{Total Man Days}} = \frac{1,484,100}{85589} = 17.34 \text{ T/M/D Clean Coal}$$

$$\frac{\text{Annual \$}}{\text{Annual Tons}} = \frac{4,173,314.50}{1,484,100} = 2.81 \text{ \$/T}$$

man shift, and the labor cost is $2.81 per ton. This is still well below the cost for the conventional system.

Additional labor will be required as the main haulage system is extended beyond the two-mile length to four or eight miles. One additional pump main would be required on each shift for each additional two miles of main haulage. The pipe extension crew would be increased two men for each additional four miles of main haulage system length beyond the two-mile system shown in the cost estimates. Labor costs for each of the mines are summarized in Table 7.1-5.

Table 7.1-5

Labor Costs Summary
(Shaft Depth 300 to 2000 feet)

Length of Mine Haulage	Cost $/Ton
Conventional	
1 mile	3.59
2 miles	3.59
4 miles	3.72
8 miles	3.98
Hydraulic Direct Feed	
1 mile	2.81
2 miles	2.81
4 miles	2.87
8 miles	2.93
Hydraulic Surge Feed	
1 mile	2.49
2 miles	2.49
4 miles	2.55
8 miles	2.60

7.2 MATERIAL AND SUPPLY COSTS

Material and supply costs for both the conventional and hydraulic haulage mines with two-mile man haulage systems are summarized in Table 7.2-1. The largest single element of this cost is repair parts which is assumed to be 10% of the facilities cost per year. Because of the variation in investment with main haulage system length, supply costs will vary. Supply costs for each of the systems are summarized in Table 7.2-2.

Table 7.2-2

Material and Supply Costs Summary ($/T)

Length of Main Haulage	Depth of Cover (feet)				
	300	500	1000	1500	2000
Conventional Mine					
1 mile	1.95	1.99	2.11	2.22	2.31
2 miles	1.98	2.03	2.14	2.26	2.34
4 miles	2.04	2.08	2.20	2.31	2.40
8 miles	2.15	2.20	2.32	2.43	2.52
Hydraulic Direct Feed					
1 mile	1.78	1.79	2.17	2.27	2.36
2 miles	1.83	1.84	2.23	2.33	2.42
4 miles	1.94	1.94	2.27	2.43	2.51
8 miles	2.14	2.14	2.53	2.63	2.72
Hydraulic Surge Feed					
1 mile	1.68	1.68	2.02	2.11	2.19
2 miles	1.72	1.72	2.07	2.16	2.24
4 miles	1.81	1.82	2.16	2.25	2.33
8 miles	1.98	2.00	2.34	2.43	2.51

TABLE 7.2-1

Supply Costs (per ton of Clean Coal)

(Two Mile Main Haulage)

Cost Item	Cost Estimation	Tonnage Involved Conventional	Tonnage Involved Hydraulic	Cost/ton Conventional	Cost/ton Hydraulic
Machine Repair Parts	10% of Facilities/yr Conventional -- $1,458,500 Hydraulic -- $1,909,420	1,104,150	1,672,800	$1.32	$1.14
Oil and Grease	10-gal hydraulic oil/sec/shift = 150 gpd at $0.95/g = $143. Gear oil and grease: $50/day in sections; $100/day general mine.	5,521	8,364	0.05	0.04
Rock Dust	7 lb of dust used/ton of coal mined. Cost = $14/ton or 286 tons of raw coal/ton of rock dust, or 243 tons of clean coal.	243	243	0.06	0.06
Roof Bolts	5-ft x 4-ft centers = 16 bolts/cut of 81.7 raw tons or 69.5 clean tons. $1.55/bolt x 16 = $24.80.	69.5	69.5	0.35	0.35
Timber	$0.20/lineal ft = $1.10 ea. Estimate 60/day or $66.00.	5,521	8,364	0.01	0.01
Brattice Cloth	1/5 roll/unit/day x 5 units = 1 roll/day. $25.00/roll.	5,521	8,364	Nil	Nil
Stoppings	5.5 ft x 16 ft = 88 sq ft ÷ 1-1/8 sq ft = 80 cement blocks/stopping at $0.30/block = $24.00. 3 sacks cement at $1.50 ea = $4.50 or $28.50 ea stopping. At 20 ft advance/shift we need 1 stopping x 5 units x 3 shifts x $28.50 = $428/day.	5,521	8,364	0.08	0.05
Mine Track	60 lb rail costs $200/ton; 40 lb/ft track = $4.00/ft. All accessories = $2.00/ft Total = $6.00/ft Entries advance 20 ft/shift or 60 ft/day x $6 = $360/day. Entries work 1/3 total time or $120/day.	5,521	8,364	0.02	0.01
Fresh Water Pipe and Drainage Pipe	Entries advances 20 ft/shift or 60 ft/day worked. Work 1/3 of time = 20 ft/day avg. 4 in. pipe at $2.88/ft = $57.60/day, 4 in. pipe at $1.50/ft = $30.00/day.	5,521	8,364	0.02	0.01
Electric Cable	Entries advance as above 20 ft/day x $7.00 = $140.	5,521	8,364	0.03	0.02
Overcasts	Main Entry $\frac{\$1,200 \text{ ea} \times 8}{5,600 \text{ ft}} = 1.72/\text{ft}$ Submain Entry $\frac{\$1,200 \text{ ea} \times 5}{610 \text{ ft}} = \$9.09/\text{ft}$ ($1.72 + $9.09/ft) x 20 ft = $216.20/day.	5,521	8,364	0.04	0.03
	Total Supply Cost			$1.98	$1.72

8.0 COMPARISON OF CONVENTIONAL AND HYDRAULIC HAULAGE MINES

Comparison of conventional and hydraulic transportation mines, as detailed in this study, may be made in a number of areas. These include annual coal production, labor productivity, worker safety and capital and operating costs.

8.1 DAILY AND ANNUAL COAL PRODUCTION

Daily section and annual coal production rates are tabulated in Table 8-1. From this table, it is seen that coal production with continuous hydraulic haulage is 52% higher than with the conventional shuttle car system. Even the direct hydraulic system is 34% higher than for the conventional system.

TABLE 8-1

Coal Production Rates

Haulage System	Tons Raw Coal per Continuous Miner per Shift	Total Tons Clean Coal per Year	Percent Increase
Conventional	433	1,104,150	--
Hydraulic			
Surge Feed System	656	1,672,800	52
Direct Feed System	582	1,484,100	34

8.2 LABOR PRODUCTIVITY

Labor forces for both the hydraulic and conventional haulage mines were developed in Section 7 for mines having a two-mile main haulage system. Labor requirements for the hydraulic transportation mine are greater than those for the conventional haulage mine.

Labor forces and productivity for the two-mile man haulage mines are summarized in Table 8-2. From this table, it will be seen that productivity for the hydraulic haulage mines is greater than that for the conventional mine. In spite of an 11% larger work force, productivity is 38% greater for the surge feed system and 22% greater for the direct hydraulic system.

TABLE 8-2

Productivity for Hydraulic Versus Conventional Haulage Mines with Two-Mile Haulage System

Haulage System	No. Men	Tons/Man Shift Clean Coal	Percent Increase
Conventional	335	14.17	--
Hydraulic			
Surge Feed System	373	19.54	38
Direct Feed System	373	17.34	22

8.3 MINE SAFETY

Although no statistics are available for hydraulic haulage mines, they would be safer than mines equipped with conventional haulage systems. Main haulage clean-up and inspection activities would be confined to pump stations which would be well lighted and safety engineered. Face and secondary haulage activities would also be confined to small areas of the mine and moving equipment would be minimized.

According to Barry[4], 11.6% of all fatal accidents are related to shuttle cars. Of these, nearly one-third are the result of pedestrians being run over or crushed by the shuttle car. The hydraulic haulage mine would eliminate the shuttle car and could be expected to reduce accidents by at least 10% in any mine using it.

Conveyor belt accidents account for about 2% of the serious injuries in underground coal mines. Again, the use of the hydraulic system would eliminate these accidents.

The elimination of conveyor belts would also remove a potential source of fires or gas ignition. Even though deluge sprays are required on belts, they may not completely eliminate the possibility of a burning belt and its potential for a catastrophic ignition of gas or dust.

Another significant safety plus would be the reduction in airborne dust particles. Only in active work sections would this problem exist because coal would be entrained in water everywhere else in the mine. Spills would be infrequent, and when one did occur, the slurry would not become a source of dust for the mine ventilation system.

As stated earlier, no figures are available for safety in hydraulic transport mines. However, it is safe to say that the adaptation of such a system on an industry-wide basis would probably save 10 to 15 lives per year and would eliminate another 50 to 75 serious injuries annually.

8.4 COSTS PER TON OF CLEAN COAL

Table 8-3 shows detailed projected operating costs for the mines developed in this study with a two-mile main haulage system and 300-foot depth. Costs for other depths and distances will be affected by capital, labor, supply, and power cost variations. Additional labor will be necessary for the conventional system at the four and eight mile haulage distances as more men will be needed to patrol the belts. Some increase in power costs will also occur, but this will have a negligible effect on the per-ton cost of coal. The following description should facilitate the use of the table.

Line 1 is the annual production of clean coal.

Line 2 is the total initial capital expenditure from Table 6-4 to place the mine in full design capacity production.

(Projected Operati

Line	Item	Co per ton	ystem ual
1	Tons Sold		4,100
2	Initial Investment		0,200
3	Units Working - Miner		5/5
4	Employees - Nonunion		61
5	Employees - Union		12
6	Tons/Unit Shift - Miner		82
7	Labor - Union	$2.726	4,583
8	Repairs and Supplies	1.980	2,652
9	Power	0.120	2,120
10	Total Underground	4.826	9,355
11	Other Haulage - Overland Conveyor	0.020	2,262
12	Roads	0.020	2,262
13	Coal Preparation - R & S Only	0.150	2,615
14	Supervision and Office	0.860	8,732
15	Total Controllable	5.876	5,226
16	UMW of America Welfare	0.800	7,280
17	Pension and Hospital - $2,400/Co. Man	0.122	6,400
18	Vacation Bonus - $450/Union Man	0.114	7,504
19	Engineering - Land and Drilling	0.040	9,364
20	Taxes - R.E. and P. Prop., etc. - 3%	0.092	2,481
21	Taxes - Sales and Use - 4%	0.080	0,919
22	Division Overhead	0.010	4,841
23	Payroll Taxes - $1,098/Man	0.332	9,554
24	Insurance - General	0.010	4,841
25	Workmen's Compensation - $600/Man	0.182	3,800
26	Reclamation	0.010	4,841
27	Miscellaneous	0.010	4,841
28	Total Noncontrollable	7.678	1,892
29	Royalty	0.250	1,025
30	Depreciation	1.127	0,739
31	Administration and Selling Expense	0.130	2,933
32	Total Mine Cost	$9.185	6,589

Line 3 shows the number of operating units working on each of three shifts.

Line 4 is the number of company supervisory and clerical personnel required on the payroll.

Line 5 is the number of union personnel required.

Line 6 is the projected production of clean coal per unit per shift.

Line 7 is the cost of union labor in dollars on a per-ton basis in the first column and on an annual basis in the second column.

Line 8 covers repair and supply costs which have been developed as illustrated in Table 7.2-1.

Line 9 has been estimated from experience with the power costs of existing conventional mines of this size escalated to 1974 costs. Because connected horsepower for the hydraulic system is about 2.4 times that of conventional haulage, these power costs have been increased by this factor.

Line 10 gives the Total Direct Mine Cost.

Line 11 is the cost for maintaining the overland conveyor or pipelines.

Line 12 covers road maintenance for mine access and haulage roads including supply areas.

Line 13 is the cost of material and supplies for operating and maintaining the preparation plant.

Line 14 shows the direct labor cost for the company employees given in Line 4.

Line 15 is called the "Total Controllable Cost" because it can be controlled by management in the number of personnel, the purchase and operation of equipment and the use of supplies.

Line 16 is the amount paid to the UMWA union to support their Health and Welfare Fund.

Line 17 is the cost to the company for hospitalization and pension plans for their nonunion employees.

Line 18 is the cost of vacation pay for the union employees on the payroll.

Line 19 is a charge for special engineering necessary as the mine progresses plus the cost of bore holes and additional exploratory drilling in advance of mine workings.

Line 20, Real Estate and Personal Property Taxes, vary widely from one locality to another.

However, for this report, these taxes have been calculated as follows: Assumed valuation x 0.03 = annual tax dollars:

$$\frac{\text{Total Investment}}{5} = \text{assumed valuation.}$$

Line 21 also varies with the locality. We have assumed a tax of four percent and have applied it to the operating repairs and supplies cost. (Underground Supply Cost per ton + Preparation Supply Cost per ton) x annual tons x 4% = annual tax dollars)

Annual tax dollars ÷ annual tons = cost/ton

Line 22 covers assigned corporate overhead.

Line 23 (Payroll Taxes) includes FICA and Unemployment Insurance as follows:

FICA = 5.85% x $11,500 = $ 673/yr/man

Unemployment Insurance = 3.7% x $11,500 = $ 425/yr/man

Total Tax = $1,098/man

Line 24 - This item covers the cost of general insurance of mine facilities and equipment.

Line 25 - Workmen's compensation charges vary not only with locality but with safety records. An assumed value of $600/man/year has been used.

Line 26 - This charge has been included to cover the cost for reclaiming surface areas following completion of operations and compensation for damages due to mining. Reclamation includes such things as the slurry and refuse disposal areas, surface plant site rejuvenation, and backfilling the shafts.

Line 27 - This charge is to cover anything not included above and deficiencies in the amounts estimated in some of the above items.

Line 28 - The total of Lines 16 through 27 is designated as total non-controllable costs; that is, they have a fixed rate or the work must be done on a basis which is independent of effective mine operation.

Line 29 - Royalty, of course, depends upon the rates set by the original property owner. A rate of $0.25 per ton of clean coal has been assumed for this study.

Line 30 - Depreciation of capital equipment has been calculated on the basis of its depreciable life. Total mine life has been assumed to be 20 years. Only the capital required to bring the mine to design capacity (initial capital investment) has been depreciated. Capital expenditures for renewals, replacements and subsequent additions to the mine throughout its life have not been considered.

$$\text{Annual Depreciation} = \frac{\text{Equipment Cost}}{\text{Deprec. yrs Life}}$$

Annual Depreciation ÷ Annual Tons = Depreciation Cost/ton.

Line 31 - Administration and selling expense is the cost in a large company for main office operation and sales expenses.

Line 32 - Totals of all above costs are given in this line and indicate the cost for producing each ton of clean coal. These figures do not include amortization of capitalized development charges nor return on investment.

A glance at the total costs of coal produced by each of the mines reveals that both of the hydraulic haulage mines will produce coal at a lower cost than the conventional haulage mine. Mine mouth costs for the surge feed hydraulic system, utilizing the centrifugal pumps for hoisting, are as much as 15% lower than those for the conventional haulage system. A similar comparison for the direct feed system shows a cost reduction of 12%. These savings are reduced about 4% when the Hitachi hydrohoist is used in the deeper shafts, because of the very high cost of this equipment. A comparison of mine mouth coal costs for each of the systems is shown in Table 8-4.

Table 8-4

Summary of Mine Mouth Coal Costs ($/T Clean Coal)

Length of Mine Haulage	Depth of Cover 300	500	1000	1500	2000
Conventional Haulage					
1 mile	9.133	9.211	9.428	9.636	9.804
2 miles	9.185	9.275	9.480	9.698	9.856
4 miles	9.422	9.500	9.717	9.925	10.093
8 miles	9.879	9.967	10.184	10.392	10.560
Hydraulic Haulage - Surge Feed					
1 mile	7.726	7.734	8.355	8.526	8.678
2 miles	7.801	7.809	8.440	8.610	8.763
4 miles	8.021	8.039	8.660	8.830	8.983
8 miles	8.382	8.410	9.031	9.201	9.354
Hydraulic Haulage - Direct Feed					
1 mile	8.079	8.111	8.588	8.792	8.960
2 miles	8.151	8.201	8.682	8.887	9.055
4 miles	8.365	8.442	8.850	9.117	9.270
8 miles	8.713	8.863	9.307	9.516	9.680

9.0 RECOMMENDATIONS FOR FURTHER RESEARCH

Further research work is required to provide hard cost and performance data needed to convince the coal mining industry that the massive investments required by a hydraulic transport system will be cost effective. Areas needing particular attention are determination of actual hydraulic characteristics of coarse coal slurries, including the degree of coal degradation to be expected in hydraulic transport, the design of a coal feeder for the hydraulic system, and the demonstration that the system will perform as well as this study indicates that it will.

9.1 HYDRAULIC DATA

A major area of concern in this study has been the hydraulic data used to predict the flow characteristics for coarse coal slurries. Three theories were used and the data scatter is quite wide.

The economics of the entire hydraulic system may be significantly affected by actual values for such things as slurry concentrations, slurry minimum velocities, and slurry friction losses. For example, an assumed slurry concentration of 33% by weight has been used in this study. In order to move 821.1 tons of coal (three sections), a 16-inch pipe with three-eighths-inch walls was used. Slurry velocity was estimated to be 16.68 feet per second, and friction head losses 34 psi per 1,000 feet. If a 45 weight percent slurry could be pumped at 15 feet per second in a 14-inch pipe, the head loss would only be about 42 psi per 1,000 feet. The power consumption for the smaller pipe would be about 25% higher than for the larger pipe. However, capital investment for the smaller pipe would be approximately

13% lower. Because of the magnitude of the investment relative to the power costs, this would result in a significant cost reduction.

Of even greater impact is the minimum velocity chosen. For example, if the slurry concentration could be increased to 50 weight percent, and the velocity lowered below the Terada prediction, (12.46 fps in a 16-inch pipe) to about nine feet per second, the 821 tons per hour of coal would be pumped through the 16-inch pipe at a considerable power cost savings because the tonnage of coal being moved per installed horsepower would be greater. The data available to compute minimum transport velocities are inadequate, and therefore, substantial cost savings could result if better design data were available.

The effect of hydraulic transportation on coal particle sizes is poorly understood. The two-inch lump coal fed into the system at the face is assumed to be reduced to about 1.5 inches at the hydrohoist. A further reduction would be expected by the time the coal reaches the preparation plant. A rapid increase in coal fines generation might adversely affect the economics of the system because of increased preparation plant costs, while a lesser quantity of fines would increase the pumping power requirements used in this study. In order to determine true system economics, additional research on coal degradation in hydraulic transport is needed.

9.2 HYDRAULIC HOISTING

Another area requiring further research is hydraulic hoisting. The hydrohoist used in this study is extremely expensive, and it appears that a 4% cost reduction could be achieved with centrifugal pumps.

The power requirements needed for pumping coarse slurry up a verticle pipe are not easily defined. The available theories are contradictory and experience at the Research Institute indicates that friction losses may be much higher than indicated by the various theories.

Because the centrifugal pumps have relatively low head capacities, as many as 20 of them might be required in the deeper shafts. The effect on large particle sizes of this many pumps has been assumed to be extreme. However, no actual information is available.

Because the data for developing a precise mathematical model of the vertical hydraulic system are lacking, an extensive research program conducted in a test facility is needed. Such a program should be conducted in several pipe sizes with both coarse and fine solids. Following the data acquisition period, a thorough and competent evaluation should be conducted so that the study will result in adequate modeling and scaling equations for vertical transport systems.

9.3 PUMP EFFICIENCIES

An additional area requiring experimental work is that of pump efficiencies with coarse slurries. Pump curves are run under controlled conditions with water. As yet, little has been done to determine the effect of coarse solids on pump efficiencies. If efficiencies greater than the 45% used in this study may be demonstrated, a considerable cost reduction would be achieved. For example, a 10% increase in pump efficiency would reduce the mine mouth coal cost nearly 10 cents per ton.

9.4 COAL FEEDER

The design for the coal feeder selected in this study needs considerable improvement. Such feeders are in existance and will work. However, no one, with the possible exception of Continental Oil Company[4], appears to have such a feeder working. The design problems are not formidable, and it is recommended that a demonstration feeder of the type discussed in this study be constructed. In addition to the feeder developed here, others may have considerable merit and should not be ignored by the Bureau.

9.5 MINE SIMULATION

The mining simulation efforts attempted as part of this study should be resumed. New simulation programs, available as part of the Bureau research efforts, should allow complete mine systems analysis. Such programs would allow the selection of optimal main haulage system composed of one or more pipes. Simulated operations coupled with available cost analyses should allow the selection of the most cost effective design.

9.6 SUBSYSTEM DEVELOPMENT

A program of subsystem development leading to a full-scale demonstration should be undertaken. The area of coal-water mixing and introduction into the pipe should be carefully examined first and feeders developed. This is a major unknown element in the entire hydraulic system, and the successful development of a feeder will do much to ensure the success of the hydraulic transport concept.

After the feeder concept is proven, the face haulage subsystem should be established in two or more adjacent sections in a coal mine, and the problems of multiple feed points to the main haulage system studied. The operation of two sections simultaneously will require the development of a complete control system capable of operation in a coal mine environment. This study assumes that certain available electronic sensors such as level indicators, flowmeters, and densitometers and controls will perform in the coal mine, but modification to make this equipment permissible and operable will be required.

Next the problems of the main haulage system should be examined. The number and size of pipes, the recycle of water, the staging or separation of pumps, and problems of multiple feed points should be examined. A major effort to define the control problems and to develop a complete automated main haulage system should be undertaken.

9.7 HYDRAULIC HAULAGE DEMONSTRATION

When solutions to the problems outlined above have been demonstrated, a full-scale system demonstration should be conducted. The objective of this demonstration should be to establish that the completely automated, fully hydraulic underground coal transportation system is both economically and technically viable.

The demonstration would probably best be accomplished in conjunction with a coal producer who would operate the system in conjunction with a new mine. In this way, the mine could be designed for the hydraulic system and a realistic evaluation would be possible. A further benefit would be gained by continuing to operate the hydraulic transport system after the research program was completed. This would allow long-term evaluations of maintenance and supply problems to be made.

The principal objective of the full-scale demonstration would be to show that each of the subsystems could be combined into a technically and economically effective coal transport system. From this demonstration, factual data should be available which would demonstrate the cost-effectiveness and safety of the hydraulic transport system.

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APPENDIX

APPENDIX, EXHIBIT 1

PREDICTING ENERGY REQUIREMENTS FOR PIPELINING COAL SLURRIES

SLURRY PIPELINING CONDITIONS IN THE MINE

The design conditions for studying the application of hydraulic transportation to an underground coal mine have been discussed previously in Section 5.

AVAILABLE HEAD LOSS DATA

Little data are available in the literature on the hydraulic transportation of coarse coal in pipelines. Two of the latest references dealing with coal coarser than one inch are discussed, along with U.S.B.M. data.

Terada Data

Terada[27] claims to have substantiated a theory based on the Durand concept of slurry head losses superimposed over those of clear water head losses. His theory is modified for a sliding bedload flow regime in horizontal pipelines. While the regime is not described in detail, it appears to be one in which "all of the solid particles inside the pipeline are sliding". Data from the literature and experience with the Hitachi Hydrohoist have been used to formulate Terada's head loss equation given by:

$$\frac{f(i_m - i)}{C_v \cdot i} = 2.25 \left[V_m^2 / gD(S-1) \right]^{-1.45} \qquad (1)$$

where i_m = slurry head loss, ft of water/ft of pipe

i = clear water head loss, ft of water/ft of pipe

C_v = volumetric concentration (decimal fraction)

Table 1-1 Slurry Pipelining Conditions in the Mine

Subsystem	Coal Sp Gr	Slurry Weight Concentrate %	Pipe Diameter (I. D.) in.	Pipe Roughness, ft x 10^5	Maximum Pipe Length, ft	Coal Rate, ton/hr	Size Distribution in.
	Specifications						
1. Face Haulage	1.42	33	10.020	15	1,000	273.7	2 x 0
2. Secondary Haulage (5 panels operating)			13.250 15.250		2,800	273.7	1-3/4 x 0
3. Main Haulage			13.250 15.250		42,240 42,240	547.4 821.1	1-1/2 x 0
4. Vertical Hoisting			14.3 13.6		2,000	410	1-1/4 x 0
5. Aboveground Haulage (horizontally to wash-plant)			13.25		3,000	410	1 x 0

V_m = mean slurry velocity, fps

g = gravitational acceleration, ft/sec^2

D = inside pipe diameter, ft

S = specific gravity of coal

f = clear water Darcy-Weisbach friction factor

Letting $FR = V_m^2/gD$, Eq. (1) reduces to

$$i_m = f \cdot FR/2 + 1.125\ C_v \cdot FR \left[(S-1)/FR\right]^{1.45} \qquad (2)$$

Equations 1 and 2 are independent of coal size and distribution provided the majority of coal particles possess a settling velocity in the turbulent zone of settling; that is, the drag coefficient is a constant approximating 0.44.

Table 1-2 lists the coarse coal data used by Terada in formulating his head loss correlation. In addition, on the basis of documented data and experience, Terada has modified the Durand equation for the limiting velocity of deposition to one which gives the "practical speed for any type of granular solids", defined by

$$V_o = \sqrt{10.136\ gD(S-1)\ C_v^{\ 0.258}} \qquad (3)$$

with all of the variables previously defined. Terada estimates that the practical speed or operating velocity is approximately 20% to 30% higher than the velocity at which the slurry head losses are a minimum.

Table 1-2 Experiments Made in Various Countries to Determine Sliding-Flow Resistance(1)

Researcher (Country)	Pipe Diameter mm	Coal Diameter mm	Specific Gravity	Volumetric Concentration %
Worster (England)	76	12	1.4	5-20
Worster (England)	305	75-100	1.4	30
Fontein (Holland)	--	--	1.35*	20*
Anon (Japan)	--	--	1.4*	20*

* Estimated.
(1) Taken from Reference 27.

Govier and Aziz Data

Govier and Aziz(2) discuss a theory advocated by Gaessler based on the hydraulic transportation of coarse coal in pipelines. The theory is lengthy and somewhat complex, and was established from various tests. Some of the data from the test involving the coarsest coal and the largest pipe diameter are given in Table 1-3.

Table 1-3 Data on Gaessler's Coarse Coal Test

		Screen Analysis of Coal: Mesh	%
Size Range:	0-10mm		
Mean Diameter:	5.2mm	+3/8 in.	4.5
Specific Gravity:	1.27	3/8 in. x 4M	38.0
Pipe Diameter (I.D.):	160mm	4M x 1/8 in.	32.0
Concentration, C_V:	10-45%	1/8 in. x 16M	21.5
Mean Settling Velocity:	0.168 m/s	16M x pan	4.0
			100.0

The data presented indicate a coarse slurry but not as coarse as required for this study. However, Condolios and Chapus(3) suggest that head losses remain constant for coarse slurries which are defined as those whose mean particle size has a drag coefficient approximating 0.44. Gaessler's theory was found to be extremely complicated, especially since it was formulated from data taken only in a 160 mm diameter pipe. His data were scaled from head loss-velocity plots and recast into a Durand-type equation, and the original coal density of 1.27 (gm/cc or metric tons/m^3) was adjusted to a specific gravity of 1.42. The resulting equation is given by

$$(i_m - i)/C_v \cdot i = 74.5 \left[V_m^2/gD\right]^{-1.30} \quad (4)$$

with all the variables previously defined. The coefficient and exponent include the effect of Gaessler's coarse particle size and distribution. Eq. 4 will yield head losses for pipe wall friction for coarse coal slurries flowing in a heterogeneous suspension flow regime; that is, without a noticeable sliding bed.

The limiting velocity of deposition according to Durand and Condolios[30] is given by

$$V_d = 1.4 \sqrt{2\, gD\, (S-1)} \quad (5)$$

Faddick[31,32] has developed two additional relationships for the limiting velocity of deposition. These are

$$V_d = \frac{1.4 \sqrt{2\, gD\, (S-1)}}{(0.45/C_w)^{1/3}} \quad (6)$$

and

$$V_d = \frac{\sqrt{11\, C_v\, gD\, (S-S_L)}}{\sqrt{C_d}} \quad (7)$$

where C_w = gravimetric concentration (decimal fraction), and

S_L = specific gravity of liquid carrier (usually water)

C_d = drag coefficient of weighted mean particle size.

The other variables have been defined previously. Deposition velocities calculated from Eqs. 5, 6, and 7 were averaged for this design. Unfortunately, the above equations were formulated with slurries not nearly as coarse as the coal slurries proposed in this study.

U. S. Bureau of Mines Data

Pipeline head loss data generated by and for the U. S. Bureau of Mines were made available to the Research Institute. These data were taken with coarse coal slurries whose properties are listed in Table 1-4.

The data were combined in a Durand-type head loss correlation as follows:

$$\frac{(i_m - i)}{C_v i} = 21.2 \left[V_m^2 / gD \right]^{-1.265} \quad (8)$$

with all the variables previously defined.

Because the data taken in the four-inch-diameter pipe were so numerous compared to the data taken in the six-inch-diameter pipe, the former were weighted quite heavily in the correlation.

Table 1-4 Properties of Coal Used in USBM Pipeline Tests

Properties	Test Series 1	Test Series 2
Specific Gravity of Coal:	1.306	1.26
Inside Pipe Diameter, in.:	4.067	6.065
Pipe Type :	Schedule 40 Steel	Schedule 40 Steel
Weighted Mean Size, mm:	4.5	12.5
Coefficient of Variation, %:	94.7	75.0
No. of Pipe Tests:	109	6
Temperature, °F:	80 (est)	74 (avg)
(C_W) Concentration Range, %:	10.5-51.9	11.3-26.3
Velocity Range, fps:	1.1 to 14.2	6.2 to 10.0

Screen Analysis

Test Series 1

Sieve Size	%
1 in. x 3/4 in.	0.2
3/4 in. x 1/2 in.	3.5
1/2 in. x 3/8 in.	9.3
3/8 in. x 1/4 in.	14.7
1/4 in. x 4M	12.3
4 x 8	11.1
8 x 12	16.9
12 x 20	11.7
20 x 50	10.0
50 x 140	4.8
140 x 200	5.5
200 x pan	0.0
	100.0

Test Series 2

Sieve Size	%
1 5/8 in. x 1 1/2 in.	1.0
1 1/2 in. x 1 in.	9.2
1 in. x 3/4 in.	12.5
3/4 in. x 1/2 in.	17.3
1/2 in. x 1/4 in.	35.6
1/4 in. x 5M	1.4
5 x 10	8.8
10 x 16	4.0
16 x 20	2.2
20 x 30	1.3
30 x 50	2.1
50 x 100	1.5
100 x 200	1.1
200 x pan	2.0
	100.0

Effect of "Black Water"

Underground coal mining operations in the United States will, for the most part, require recirculation of water for hydraulic transportation by pipeline. Since total recovery of fine coals are prohibitively expensive and not particularly advantageous, the returned "black water" with a minimum of fine coal can be utilized. Through recirculation, black water can be produced whose viscosity will not be unreasonably high yet will lend supportive power to transporting coarse coal.

Each of the three theories described above (Eqs. 2, 4, and 8) was modified for a heavy medium comprising 10% by weight of fine coal. The specific gravity of the liquid carrier S_L was calculated as follows:

$$S_m = \frac{S(S_L)}{S - C_w(S-S_L)} = \frac{1.42\ (1.00)}{1.42-0.10(1.42-1.00)} = 1.030 \qquad (9)$$

Coal slurries in particular do not effect significant changes in slurry specific gravity for large changes in concentration. Consequently, any increase in specific gravity in the liquid carrier will not reduce pipe wall friction significantly because the deposition of coarse particles is only slightly affected by slurry density. Rather, it is the viscosity of the liquid carrier which is the more significant slurry property. While fluids with increased viscosity require a greater driving force to sustain flow, they will reduce significantly the settling of coarse solids in heterogeneous flow regimes. This latter effect is predominantly responsible for a significantly reduced head loss as verified at least for other mineral slurries tested at the Research Institute.

In this study, no attempt was made to estimate the viscosity of the black water for the following reasons:

1. The three head loss theories discussed above gave too wide a range of head losses indicating a strong necessity for testing the pipeline

transportation of coarse coal in large-diameter pipes. There is little point in scaling viscosity effects when head loss scaling predicts such varied head losses.

2. No data on the viscosity of recirculated black water were available at the time of writing.

In other words, the state-of-the-art of predicting head losses for the pipeline transportation of coarse coal is too rudimentary to foster further speculation on the effect of carrier viscosity on head loss. Suffice it to say that other slurries show a significant reduction in pipe wall friction when a viscous liquid medium exists.

Equations 4 and 8 were modified for black water only through specific gravity of the liquid carrier to give:

$$\frac{(i_m - i)}{C_v \cdot i} = 64.5 \left[V_m{}^2 / gD \right]^{-1.3} \qquad (10)$$

and

$$\frac{(i_m - i)}{C_v \cdot i} = 18.45 \left[V_m{}^2 / gD \right]^{-1.265} \qquad (11)$$

Procedure

Equations 2 through 11 inclusive were coded in Fortran IV language to facilitate the head loss and deposition velocity calculations, Thus, three sets of head loss data were evaluated--one for a sliding bedload flow regime (Eq. 2), and two for a heterogeneous saltation flow regime (Eqs. 4 and 8). One operating velocity was evaluated for the sliding-bedload flow regime (Eq. 3), while three values of deposition velocity were calculated and averaged for the heterogeneous solution flow regime (Eqs. 5, 6, and 7).

None of the three head loss equations are influenced by particle size and distribution, since, according to the theory, there is no increase in head losses for coarse particle slurries as the mean particle size increases for a constant concentration. The minimum value of the mean particle size for coarse coal is about six mesh for coal of specific gravity 1.42.

For comparison, the screen analyses of the coal slurries used in this study were assumed to be as shown in Table 1-5. These were developed from the data shown in Table 5.2-1 by translating size gradation curves on logarithmic paper. A maximum particle size of two inches was assumed at the coal feeder, while 1 1/2 inch was assumed at the mine hoist. It must be emphasized though, that the attrition of coal is not similar for the three distributions shown. For example, the hydraulic transportation of coal in pipelines, and particularly those where centrifugal pumps are used, causes attrition primarily in the larger coal sizes. Therefore, of the three particle size distributions listed in Table 1-5, the 2 x 0 inch distribution would be expected to suffer the greatest amount of attrition. How this would affect the viscosity of the "black water" and its supportive power for coarse solids is not known.

Table 1-5 Screen Analyses of Coal Used in Study

Sieve Size	Coal Samples % Retained 2 x 0 in.	1 1/2 x 0 in.	1 x 0 in.
2 in.	0.0	--	--
1 1/2 in.	17.0	0.0	--
1 in.	18.0	18.0	0.0
3/4 in.	10.0	14.0	20.0
1/2 in.	10.0	13.0	14.0
3/8 in.	7.5	8.0	11.0
1/4 in.	6.0	9.0	10.0
1/8 in.	10.0	12.0	13.0
10 mesh	6.7	8.0	10.5
20 mesh	4.8	6.0	6.7
35 mesh	3.2	3.7	4.8
65 mesh	2.1	2.6	3.2
100 mesh	0.9	1.0	1.1
Pan	3.8	4.7	5.7
	100.0	100.0	100.0

Head Loss Results

Tables 1-6, 1-7, and 1-8 show the predicted horizontal flow properties for a gravimetric concentration of 33% of coarse coal in clear water and black water. Each table represents a different data source. Only a summary data listing is provided from the voluminous computer output. An approximate throughput rate of 274 dry short tons per hour was selected for the smaller diameter pipelines and 548 or 822 dry short tons per hour for the larger diameter pipelines.

The range of head losses for saltation flow varies up to 250%. To complicate matters, Terada's sliding bed regime theory predicts head losses intermediate to those predicted by the saltation flow theories. The deposition velocity (Eq. 3) quoted for Terada's sliding bed regime theory is in fact an operational velocity which is approximately 25% to 30% higher than the depositional velocity.

Table 1-6 Predicted Flow Properties for 33% Coarse Coal-Water Slurries -- Saltation Flow

Data Source: USBM, 4- and 6-in. Diameter

Coarse Coal in Clear Water $S = 1.42$ $S_L = 1.000$ $S_m = 1.108$ $C_w = 0.333$ $C_v = 0.256$

Pipe I.D. in.	Velocity fps	Pipewall Friction psi/1000 ft*	Pipewall Friction kw-hr/ton-1000 ft*	Throughput dry short tons/hr	Deposition Velocity, fps
7.981	16.0	47.9 (38.8)	0.2244 (0.0675)	231.6	5.9
7.981	18.0	57.3 (48.8)	0.2685 (0.0849)	260.5	5.9
10.020	10.0	24.1 (11.9)	0.1127 (0.0206)	228.1	6.1
10.020	12.0	27.8 (16.9)	0.1303 (0.0294)	273.7	6.1
11.938	8.0	20.4 (6.3)	0.0956 (0.0109)	259.0	7.2
13.126	12.0	23.3 (12.2)	0.1093 (0.0212)	469.8	7.5
13.126	14.0	26.6 (16.4)	0.1247 (0.0286)	548.1	7.5
16.000	12.0	20.9 (9.6)	0.0980 (0.0168)	698.0	8.3
16.000	14.0	23.3 (13.0)	0.1091 (0.0226)	814.3	8.3

Coarse Coal in Black Water $S = 1.42$ $S_L = 1.030$ $S_m = 1.134$ $C_w = 0.333$ $C_v = 0.262$

Pipe I.D. in.	Velocity fps	Pipewall Friction psi/1000 ft*	Pipewall Friction kw-hr/ton-1000 ft*	Throughput dry short tons/hr	Deposition Velocity, fps
7.981	16.0	48.3	0.2212	237.0	5.6
7.981	18.0	58.1	0.2659	266.7	5.6
10.020	10.0	23.4	0.1071	233.5	6.3
10.020	12.0	27.4	0.1255	280.2	6.3
11.938	8.0	19.4	0.0888	265.2	6.8
13.126	12.0	22.8	0.1043	480.9	7.1
16.000	12.0	20.3	0.0928	714.5	7.9
16.000	14.0	22.8	0.1045	833.6	7.9

* Numbers in parentheses denote pipewall friction losses for liquid carrier only (clear water) in steel pipe.

Table 1-7 Predicted Flow Properties for 33% Coarse Coal-Water Slurries -- Sliding Bed Flow
Data Source: Terada Data, Hitachi Reference

Coarse Coal in Clear Water $S = 1.42$ $S_L = 1.000$ $S_m = 1.108$ $C_w = 0.333$ $C_v = 0.256$

Pipe I.D. in.	Velocity fps	Pipewall Friction psi/1000 ft	Pipewall Friction kw-hr/ton-1000 ft	Throughput dry short tons/hr	Deposition Velocity, fps
7.981	16.0	51.6	0.2418	231.6	8.3
7.981	18.0	60.3	0.2826	260.5	8.3
10.020	10.0	33.5	0.1572	228.1	9.3
10.020	12.0	35.3	0.1653	273.7	9.3
11.938	8.0	34.9	0.1637	259.0	10.1
13.126	12.0	33.0	0.1545	469.8	10.6**
13.126	14.0	34.5	0.1618	548.1	10.6
16.000	12.0	32.3	0.1516	698.0	11.7**
16.000	14.0	32.7	0.1535	814.3	11.7

Coarse Coal in Black Water $S = 1.42$ $S_L = 1.030$ $S_m = 1.134$ $C_w = 0.333$ $C_v = 0.262$

Pipe I.D. in.	Velocity fps	Pipewall Friction psi/1000 ft	Pipewall Friction kw-hr/ton-1000 ft	Throughput dry short tons/hr	Deposition Velocity, fps
7.981	16.0	50.0	0.2288	237.0	7.9
7.981	18.0	58.8	0.2694	266.7	7.9
10.020	10.0	30.9	0.1414	233.5	8.8
10.020	12.0	33.0	0.1511	280.2	8.8
11.938	8.0	31.4	0.1439	265.2	9.6**
13.126	12.0	30.4	0.1393	480.9	10.1
16.000	12.0	29.6	0.1354	714.5	11.2
16.000	14.0	30.3	0.1389	833.6	11.2

** Unstable flow. This design not recommended.

Table 1-8 Predicted Flow Properties for 33% Coarse Coal-Water Slurries -- Saltation Flow

Data Source: Gaessler Data, Govier/Aziz Text

Coarse Coal in Clear Water $S = 1.42$ $S_L = 1.000$ $S_m = 1.108$ $C_w = 0.333$ $C_v = 0.256$

Pipe I.D. in.	Velocity fps	Pipewall Friction psi/1000 ft*	Pipewall Friction kw-hr/ton-1000 ft*	Throughput dry short tons/hr	Deposition Velocity, fps
7.981	16.0	68.1	0.3192	231.6	5.8
7.981	18.0	76.0	0.3560	260.5	5.8
10.020	10.0	52.8	0.2475	228.1	6.5
10.020	12.0	53.1	0.2489	273.7	6.5
11.938	8.0	54.7	0.2565	259.0	7.2**
13.126	12.0	49.4	0.2317	469.8	7.5
13.126	14.0	50.1	0.2347	548.1	7.5
16.000	12.0	47.6	0.2233	698.0	8.3**
16.000	14.0	47.3	0.2218	814.3	8.3

Coarse Coal in Black Water $S = 1.42$ $S_L = 1.030$ $S_m = 1.134$ $C_w = 0.333$ $C_v = 0.262$

Pipe I.D. in.	Velocity fps	Pipewall Friction psi/1000 ft*	Pipewall Friction kw-hr/ton-1000 ft*	Throughput dry short tons/hr	Deposition Velocity, fps
7.981	16.0	66.7	0.3056	237.0	5.6
7.981	18.0	75.1	0.3438	266.7	5.6
10.020	10.0	49.6	0.2270	233.5	6.2
10.020	12.0	50.5	0.2310	280.2	6.2
11.938	14.0	49.4	0.2261	464.0	6.8
11.938	16.0	52.5	0.2403	530.3	6.8
13.126	12.0	46.6	0.2131	480.9	7.1
16.000	12.0	44.6	0.2043	714.5	7.9
16.000	14.0	44.7	0.2047	833.6	7.9

* Numbers in parentheses denote pipewall friction losses for liquid carrier only (clear water) in steel pipe.

** Unstable flow. This design not recommended.

Comments

The theories just discussed have not been verified with the coarse coal sizes and distributions assumed for this study. Therefore, the predicted head losses, deposition and operating velocities are considered as approximate, possibly with an error as large as $\pm 40\%$. There are weaknesses in the theories, including missing parameters such as the ratio of mean particle size to pipe diameter, and Reynolds number based only on the density of the liquid carrier (in this case, "black water"). The several head loss theories also yield conservatively high head losses at high values of velocity (Heterogeneous suspension flow regime). The design head losses used in this report were taken from Table 1-9, which uses Terada's theory. The difference between the high and low values of head losses predicted by the three theories is in the order of 30.1%.

It is strongly recommended that pipeline head loss data be obtained for coarse coal slurries such as those assumed in this report, in order to substantiate these theories.

TABLE 1-?

Predicted Flow Properties for 33% Coarse Coal-Water Slurries -- Saltation Flow Design Conditions Terada Theory

Coarse Coal in Clear Water $S = 1.42$ $S_L = 1.000$ $S_m = 1.108$ $C_w = 0.333$ $C_v = 0.256$

Pipe I.D. In.	Velocity fps	Pipe Wall Friction psi/1000 ft	Pipe Wall Friction kw-hr/ton-1000 ft	Throughput dry short tons/hr	Deposition Velocity, fps
10.020	8.00	34.225	0.16040	182.5	9.28
	10.00	33.535	0.15716	228.1	
	12.00	35.260	0.16525	273.7	
	14.00	38.749	0.18160	319.4	
	16.00	43.679	0.20471	365.0	
	18.00	49.859	0.23367	410.6	
	20.00	57.179	0.26798	456.2	
11.938	8.00	34.931	0.16371	259.0	10.13
	10.00	33.063	0.15495	323.8	
	12.00	33.570	0.15733	388.6	
	14.00	35.751	0.16755	453.3	
	16.00	39.252	0.18396	518.1	
	18.00	43.874	0.20562	582.9	
	20.00	49.493	0.23196	647.6	
13.126	8.00	35.509	0.16642	313.2	10.62
	10.00	33.053	0.15491	391.5	
	12.00	32.970	0.15452	469.8	
	14.00	34.527	0.16182	548.1	
	16.00	37.357	0.17508	626.3	
	18.00	41.249	0.19332	704.6	
	20.00	46.076	0.21594	782.9	
16.000	8.00	37.120	0.17397	465.3	11.73
	10.00	33.527	0.15713	581.7	
	12.00	32.336	0.15155	698.0	
	14.00	32.748	0.15348	814.3	
	16.00	34.359	0.16103	930.7	
	18.00	36.942	0.17313	1047.0	
	20.00	40.356	0.18913	1163.3	

APPENDIX, EXHIBIT 2

MAJOR EQUIPMENT LIST FOR CONVENTIONAL HAULAGE MINE

FACE AREA EQUIPMENT - PER SECTION

1 ea. - Continuous Miner - Joy Model 12CM1-10B or equivalent

1 ea. - Roof Bolter - Acme Model D-5 or equivalent

1 ea. - Feeder-Breaker - Stammler, BF-14 or equivalent

2 ea. - Shuttle Car - Joy Model 10SC-22 or equivalent

1 ea. - Rock Duster - MSA Section Duster or equivalent

1 ea. - Mobile Shop

1 ea. - 1,000-kva Power Center - Ensign Electric or equivalent

1 ea. - Scooptram - S & S or equivalent

1 ea. - Lube Car - S & S or equivalent

1 ea. - Battery Charger - Gould or equivalent

500 ft - 2 inch Waterline

400 ft - 4 inch Drainline

4 inch Drainage Pump - Flygt Vertical or equivalent

APPENDIX, EXHIBIT 2 (Continued)

MAJOR EQUIPMENT LIST FOR
CONVENTIONAL HAULAGE MINE

SECONDARY HAULAGE EQUIPMENT - PER PANEL

1 ea. - Belt Drive Unit 100 hp - Long Airdox or equivalent

1 ea. - Belt Tail Unit - Long Airdox or equivalent

3,000 ft - 36 in. Rope Belt Conveyor - Long Airdox or equivalent

1 ea. - 300-kva Transformer - Ensign Electric or equivalent

3,000 ft. - 40# Track - 42 in. gauge

3,000 ft. - 2 inch Waterline

3,000 ft. - 4 inch Drainline

1 ea. - Drainage Pump - Flygt Vertical or equivalent

3,000 ft. - Electric Cable (1/0 3C)

APPENDIX, EXHIBIT 2 (Continued)

MAJOR EQUIPMENT LIST FOR CONVENTIONAL HAULAGE MINE

MAIN HAULAGE SYSTEM (2 MILES)

1 ea. - 500-kw Rectifier - Ohio Brass or equivalent

11 ea. - 300-kva Transformer - Ensign Electric or equivalent

11,250 ft. - Electric Cable 250000CM36 8000V

1 ea. - Entry Rock Duster - National Mine Service or equivalent

11,250 ft. - 60# Mine Track 42" gauge

2 ea. - Belt Winders - Continental Conveyor or equivalent

2 ea. - Cross Belt Conveyors (500 ft.) - Long Airdox or equivalent

3 ea. - 42 inch x 4,000 ft Conveyors - Long Airdox or equivalent

36 ea. - Supply Cars - Sanford Day or equivalent

5 ea. - Personnel Carriers (4 man) - Goodman or equivalent

6 ea. - Portal Buses (16 man) - Goodman or equivalent

11,250 ft. - Trolley Wire

11,250 ft. - 4 inch Water Pipe

2 ea. - Equipment Carriers - Sanford Day or equivalent

11,250 ft. - 4 inch Drainage Pipe

2 ea. - 10-ton Electric locomotives - Goodman or equivalent

APPENDIX, EXHIBIT 2 (Continued)

MAJOR EQUIPMENT LIST FOR CONVENTIONAL HAULAGE MINE

MINE SHAFT AND HOISTING FACILITY

1 ea. - 7500-kva Substation 69000V to 7200V - Westinghouse or equivalent

1 ea. - 16 ft diameter x 2 Compartment Production Shaft

1 ea. - Emergency Hoist - Coeur d'Alene or equivalent

1 ea. - Service Hoist including Head Frame - Nordberg or equivalent

1 ea. - Service Hoist Building

1 ea. - Service Hoist Cage

1 ea. - Bulk Oil System

1 ea. - Bulk Rock Dust System - MSA or equivalent

1 ea. - Underground Shop with Equipment

7,000 ft. - 60# 42 in. Gauge Track

3,000 ft. - Trolley Wire

2 ea. - Service Locomotives, Trolley, and Battery - Sanford Day or equivalent

1 ea. - Shaft Drainage Pumps - ASH or equivalent

1 ea. - Shaft Pocket Complete with Automatic Loading Gate

3,000 ft. - Electric Cable

1 ea. - 500-kva Transformer - Ensign or equivalent

1 ea. - Production Hoist, Building, and Head Frame - Nordberg or equivalent

2 ea. - Skips

APPENDIX, EXHIBIT 2 (Continued)

MAJOR EQUIPMENT LIST FOR CONVENTIONAL HAULAGE MINE

SURFACE FACILITIES

1 ea. - Preparation Plant (300 tph)

1 ea. - Clean Coal Storage Silo (3000 tons)

1 ea. - Clean Coal Conveyor (500 feet)

1 ea. - Raw Coal Storage Silo (3000 tons)

2 ea. - Raw Coal Conveyor (500 feet)

1 ea. - Refuse Bin (50 ton) with Conveyor

1 ea. - Tailings Pond

1 ea. - 10-ton Locomotive - Sanford Day or equivalent

1 ea. - 2500-kva Substation 69000V to 4160V to 440V - Westinghouse or equivalent

1 ea. - Main Fan 350,000 cfm - Jeffrey or equivalent

2 ea. - Ventilation Shafts 12 ft. Diameter

3000 ft. - 36 inch Conveyor with Drive - Continental Conveyor or equivalent

1 ea. - Main Shop

APPENDIX, EXHIBIT 3

MAJOR EQUIPMENT LIST FOR HYDRAULIC HAULAGE MINE

FACE AREA EQUIPMENT - DIRECT FEED SYSTEM - PER SECTION

1 ea. - Continuous Miner - Joy Model 12CM1-10B, or equivalent, with breaker

2 ea. - Roof Bolter - Acme Model D-5 or equivalent

1 ea. - Rock Duster - MSA Section Duster or equivalent

1 ea. - Mobile Shop

1 ea. - 1,000-kva Power Center - Ensign Electric or equivalent

1 ea. - Scooptram - S & S or equivalent

1 ea. - Lube Car - S & S or equivalent

1 ea. - Battery Charger - Gould or equivalent

500 ft - 2 inch Waterline

400 ft - 4 inch Drainline

1 ea. - 4 inch Drainage Pump - Flygt Vertical or equivalent

4 ea - Pump Feed System

4 ea. - 10 inch Pumps - Wilfley Model 10K or Equivalent

24 ea. - Skids for 10 inch Hose

1 ea. - Electric Winch

800 ft - 10 inch Hose - Gates Rubber or equivalent

13 ea - 10 inch Plate Valves - Chronister Valve Co. or equivalent

2200 ft - 10 inch Pipe

APPENDIX, EXHIBIT 3 (Continued)

MAJOR EQUIPMENT LIST FOR HYDRAULIC HAULAGE MINE

FACE AREA EQUIPMENT - SURGE FEED SYSTEM - PER SECTION

1 ea. - Continuous Miner - Joy Model 12CM1-10B or Equivalent

2 ea. - Roof Bolter - Acme Model D-5 or Equivalent

1 ea. - Feeder-Breaker - Stammler, BF-14 or equivalent

1 ea. - Shuttle Car - Joy Model 10SC-22 or equivalent

1 ea. - Rock Duster - MSA Section Duster or equivalent

1 ea. - Mobile Shop

1 ea. - 1,000-kva Power Center - Ensign Electric or equivalent

1 ea. - Scooptram - S & S or equivalent

1 ea. - Lube Car - S & S or equivalent

1 ea. - Battery Charger - Gould or equivalent

500 ft - 2 inch Waterline

400 ft - 4 inch Drainline

1 ea. - 4 inch Drainage Pump - Flygt Vertical or equivalent

4 ea. - Pump Feed System

4 ea. - 10 inch Pump - Wilfley Model 10K or Equivalent

24 ea. - Skids for 10 inch Hose

1 ea. - Electric Winch

800 ft - 10 inch Hose - Gates Rubber or equivalent

13 ea. - 10 inch Plate Valves - Chronister Valve Co. or equivalent

2200 ft - 10 inch Pipe

APPENDIX, EXHIBIT 3 (Continued)

MAJOR EQUIPMENT LIST FOR
HYDRAULIC HAULAGE MINE

SECONDARY HAULAGE EQUIPMENT - PER PANEL

1 ea - 150-kva Transformer - Ensign Electric or equivalent

1 ea. - 10 inch Centrifigal Pump with Slurry Feeder - Wilfley Model 10K or equivalent

3000 ft - Electric Cable (1/0 3C)

3000 ft - 40# Track - 42 inch Gauge

3000 ft - 2 inch Waterline

3000 ft - 4 inch Drainline

6000 ft - 10 inch Pipe (3/8 Wall)

4 ea. - 10 inch Plate Valves - Chronister Valve or equivalent

APPENDIX, EXHIBIT 3 (Continued)

MAJOR EQUIPMENT LIST FOR HYDRAULIC HAULAGE MINE

MAIN HAULAGE SYSTEM (2 MILES)

1 ea. - 500-kw Rectifier - Ohio Brass or equivalent

14 ea. - 300-kva Transformer - Ensign Electric or equivalent

11250 ft. - Electric Cable 250,000 CM36 8000V

1 ea. - Entry Rock Duster - National Mine Service or equivalent

11250 ft - 60# Mine Track 42 inch Gauge

8 ea. - 16 inch Centrifugal Pumps - Georgia Iron Works or equivalent

2 ea. - 500-kva Transformers - Ensign Electric or equivalent

8 ea. - 14 inch Centrifugal Pumps - Georgia Iron Works or equivalent

11 ea. - 1000-kva Power Center - Ensign Electric or equivalent

3 ea. - 20 inch Centrifugal Pumps - Georgia Iron Works or equivalent

1 ea. - Integrated Hydraulic Control System - Johnson Service or equivalent

36 ea. - Supply Cars - Sanford Day or equivalent

5 ea. - Personnel Carriers (4 man) - Goodman or equivalent

6 ea. - Portal Buses (16 man) - Goodman or equivalent

11250 ft. - Trolley Wire

11250 ft. - 4 inch Water Pipe

11250 ft. - 4 inch Drain Pipe

2 ea. - Equipment Carriers - Sanford Day or equivalent

28 ea. - 14 inch Plate Valves - Chronister Valve or equivalent

30 ea. - 16 inch Plate Valves - Chronister Valve or equivalent

6 ea. - 20 inch Plate Valves - Chronister Valve or equivalent

APPENDIX, EXHIBIT 3 (Continued)

MAJOR EQUIPMENT LIST FOR HYDRAULIC HAULAGE MINE

MAIN HAULAGE SYSTEM (2 MILES)

11250 ft. - 14 inch Pipe (3/8 Wall)

11250 ft. - 16 inch Pipe (3/8 Wall)

11250 ft. - 20 inch Pipe (S40)

2 ea. - 10-ton Electric Locomotives - Goodman or equivalent

APPENDIX, EXHIBIT 3 (Continued)

MAJOR EQUIPMENT LIST FOR HYDRAULIC HAULAGE MINE

MINE SHAFT AND HOISTING FACILITY

1 ea. - 20,000-kva Substation 69000V to 7200V - Westinghouse or equivalent

1 ea. - 14 ft. diameter Production Shaft

1 ea. - Emergency Hoist - Coeur d'Alene or equivalent

1 ea. - Bulk Oil System

1 ea. - Bulk Rock Dust System - MSA or equivalent

1 ea. - Underground Shop with Tools and Equipment

7000 ft. - 60# 42 inch Gauge Track

3000 ft. - Trolley Wire

2 ea. - Service Locomotives - Trolley and Battery - Sanford Day or equivalent

1 ea. - Hitachi Hydrohoist or Warman High Pressure Pumps

1 ea. - Dewatering Sump

300-2000 ft. - 14 inch Pipe (Variable Wall Thickness)

APPENDIX, EXHIBIT 3 (Continued)

MAJOR EQUIPMENT LIST FOR HYDRAULIC HAULAGE MINE

SURFACE FACILITIES

1 ea. - Preparation Plant (400 tph)

1 ea. - Clean Coal Storage Silo (3000 ton)

1 ea. - Clean Coal Conveyor (500 feet)

1 ea. - Raw Coal Storage Silo (3000 ton)

2 ea. - Raw Coal Conveyor (500 feet)

8 ea. - 6 ft. DSM Screens

12 ea. - 4 ft x 8 ft Vibrating Screens

1 ea. - 60 ft x 12 ft Hydroclarifier

1 ea. - Refuse Bin (50 ton) with Conveyor

1 ea. - Tailings Pond

1 ea. - 10 ton Locomotive - Sanford Day or equivalent

1 ea. - 3500-kva Substation 69000V to 4160V to 440V - Westinghouse or equivalent

1 ea. - Main Fan 350,000 cfm - Jeffrey or equivalent

2 ea. - Ventilation Shafts 12 ft diameter

3000 ft - 14 inch Pipe

1 ea. - Main Shop

APPENDIX, EXHIBIT 4

LABOR COSTS FOR CONVENTIONAL MINING SYSTEM

Table 4-1 Summary of Labor Costs Per Year

5/5/5 Mining Units
433 T/U/S

January 1974 Labor Costs
LABOR

1,299,000 T/yr Raw
1,104,150 T/yr Clean
200 Days

CODE	DESCRIPTION	Table	Cost Per Day	No Days	Cost Per Year
1	Normal Work Day	4-2	$15,870.63	200	$3,174,126.00
2	Sat. Work Day				
3	Holiday Work Day				
5	Idle Day – Men Wkg	4-3	9,918.04	34	337,213.36
6	Idle Day – All Other	4-4	3,556.24	9	32,006.16
7	Idle Day – Sat. Men Wkg.	4-5	9,922.53	9	89,302.77
8	Idle Day – Sat. after 4 or less	4-6	3,033.64	43	130,446.52
9	Sunday – After 6 Work Days				
10	Sunday – After 5 or less wk. day	4-7	396.60	52	20,623.20
11	Holiday – Idle	4-8	16,267.23	9	146,405.07
	Company Vacation		2,835.89	10	28,358.90
	TOTAL LABOR				$3,958,481.98

$$\frac{2,835.89 \times 313 \times 1.07}{1,109,250\ T} = \text{Company}$$

Total = $3.569/ T Clean
Company = -0.856
Union = $2.713/ T Clean

5/5/5 Mining Units
433 T/U/S

TABLE 4-2 -- Conventional Mining System
Straight Time Workday

1,299,000 T/yr Raw
1,104,150 T/yr Clean
200 Days/yr

DAILY LABOR COST ESTIMATE
January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator	5	5	5	15	750.00
Loader Helper	5	5	5	15	708.75
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators	10	10	10	30	1,297.50
Roof Bolters	5	5	5	15	708.75
Timbermen					
Ventilation	5	5	5	15	671.25
Rock Dusters					
Belt Patrol					
Cleanup	5	5	5	15	671.25
Water	3			3	134.25
TOTAL PRODUCTION	43	40	40	123	$5,804.40
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners	2	2	3	7	350.00
Cagers	1	1	1	3	128.25
Trackmen			2	2	85.50
Pumpmen	1	1	1	3	128.25
Timbermen	2		2	4	171.00
Ventilation	4	4	4	12	537.00
Rock Dusting			2	2	85.50
Recovery			4	4	173.00
Rock Loading					
Motor & Supply	3	3	3	9	389.25
Hopper Operator					
Main Belt Patrol	4	4	4	12	513.00
Belt Extension			6	6	256.50
Dust Control	1			1	35.00
TOTAL NON-PRODUCTION	24	18	35	77	$3,562.35

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen	5	5	5	15	750.00
Unit Greasers			5	5	213.75
Shop Repairmen	3	3	3	9	405.00
Misc. Repairmen	3	2	2	7	350.00
Wiremen			1	1	42.75
Elec. Insp.	1			1	50.00
Welders	1	1	1	3	150.00
Belt Repairs			2	2	85.50
Instrument Maint.	1			1	35.00
TOTAL MAINTENANCE	17	13	21	51	$ 2,530.57
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	130.50
Watchmen					
Janitor					
Lamp House Attend.	1	1	1	3	123.75
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver	1	1		2	82.50
Yard Men	1	1	1	3	123.75
Bulldozer Operator	1			1	43.50
Crane Operator					
Top Repairmen					
Nippers & Droppers Sh. Col.	1	1	1	3	123.75
TOTAL TOP	7	6	5	18	$ 800.28
WASHER					
Foremen	1	1	1	3	172.53
Operators	2	2	2	6	270.00
Repair & Maintenance	3	3	3	9	405.00
Nippers & Droppers	1	1	1	3	123.75
Refuse Disposal	2	2	2	6	261.00
TOTAL WASHER	9	9	9	27	$ 1,232.28
Supervision	8	1	1	10	399.51
TOTAL MINE					$14,329.39
Shift Differential		58.60	117.20		175.80
TOTAL MINE					$14,505.19

Date:		Non-Union Men	52
		Union Men	254
Made By:		TOTAL MEN	306

COLORADO SCHOOL OF MINES RESEARCH INSTITUTE

TABLE 4-3 -- Conventional Mining System
Idle Weekday

34 Days

DAILY LABOR COST ESTIMATE

January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters	5	5	5	15	708.75
Timbermen					
Ventilation	5	5	5	15	671.25
Rock Dusters					
Belt Patrol					
Airdox					
Water	3			3	134.25
TOTAL PRODUCTION	18	15	15	48	$2,376.90
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners	2	2	3	7	350.00
Cagers	1	1	1	3	128.25
Trackmen			2	2	85.50
Pumpmen	1	1	1	3	128.25
Timbermen	2			2	85.50
Ventilation	3	3	3	9	402.75
Rock Dusting					
Recovery			4	4	173.00
Rock Loading					
Motor & Supply	3	3	3	9	389.25
Hopper Operator					
Main Belt Patrol					
Belt Extension			6	6	256.50
Dust Control	1			1	35.00
TOTAL NON-PRODUCTION	19	13	26	58	$2,744.10

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen	5	5	5	15	750.00
Unit Greasers					
Shop Repairmen	3	3	3	9	405.00
Misc. Repairmen	3	2	2	7	350.00
Wiremen					
Elec. Insp.	1			1	50.00
Welders	1	1	1	3	150.00
Belt Repairs					
Instrument Maint.	1			1	35.00
TOTAL MAINTENANCE	17	13	13	43	$2,188.57
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	130.50
Watchmen					
Janitor					
Lamp House Attend.	1	1	1	3	123.75
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver	1			1	41.25
Yard Men	1	1	1	3	123.75
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers	1	1	1	3	123.75
TOTAL TOP	6	5	5	16	$ 715.53
WASHER					
Foremen	1	1	1	3	172.53
Operators					
Repair & Maintenance	3	3	3	9	405.00
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER	4	4	4	12	$ 577.53
Supervision	8	1	1	10	399.51
TOTAL MINE					$9,002.14
Shift Differential		30.40	61.20		91.60
TOTAL MINE					$9,093.74

Date:	Non-Union Men	52
	Union Men	135
Made By:	TOTAL MEN	187

TABLE 4-4 -- Conventional Mining System
Idle Weekday -- No Work

9 Days

DAILY LABOR COST ESTIMATE

January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters					
Timbermen					
Ventilation					
Rock Dusters					
Belt Patrol					
Airdox					
Clodmen					
TOTAL PRODUCTION	5	5	5	15	$ 862.65
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners			3	3	150.00
Cagers					
Trackmen					
Pumpmen	1	1	1	3	128.25
Timbermen					
Ventilation					
Rock Dusting					
Recovery					
Rock Loading					
Motor & Supply					
Hopper Operator					
Main Belt Patrol					
Belt Extension					
Dust Control	1			1	35.00
TOTAL NON-PRODUCTION	8	4	7	19	$1,023.35

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen					
Unit Greasers					
Shop Repairmen					
Misc. Repairmen	2	2	2	6	300.00
Wiremen					
Machinists					
Welders					
Belt Repairs					
Instrument Maint.	1			1	35.00
TOTAL MAINTENANCE	6	4	4	14	$ 783.57
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	130.50
Watchmen					
Janitor					
Lamp House Attend.					
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver					
Yard Men					
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers					
TOTAL TOP	2	2	2	6	$ 303.03
WASHER					
Foremen	1	1	1	3	172.53
Operators					
Repair & Maintenance					
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER	1	1	1	3	$ 172.53
Supervision	8	1	1	10	399.51
TOTAL MINE					$3,544.64
Shift Differential		3.20	8.40		11.60
TOTAL MINE					$3,556.24

Date:	Non-Union Men	52
	Union Men	15
Made By:	TOTAL MEN	67

5/5/5 Mining Units | TABLE 4-5 -- Conventional Mining System | 9 Days

Saturday -- Full Maintenance

DAILY LABOR COST ESTIMATE

January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters					
Timbermen					
Ventilation					
Rock Dusters					
Belt Patrol					
Airdox					
Clodmen					
TOTAL PRODUCTION	5	5	5	15	862.65
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners	2	2	3	7	525.00
Cagers	1	1	1	3	192.38
Trackmen					
Pumpmen	1	1	1	3	192.38
Timbermen	2		2	4	256.50
Ventilation	3	3	3	9	604.13
Rock Dusting					
Recovery			4	4	259.50
Rock Loading					
Motor & Supply	3	3	3	9	583.88
Hopper Operator					
Main Belt Patrol					
Belt Extension			6	6	384.75
Dust Control	1			1	35.00
TOTAL NON-PRODUCTION	19	13	26	58	$3,743.62

Date:	Non-Union Men	52
	Union Men	101
Made By:	TOTAL MEN	153

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen	5	5	5	15	1,125.00
Unit Greasers					
Shop Repairmen	3	3	3	9	607.50
Misc. Repairmen	2	2	2	6	525.00
Wiremen					
~~Machinist~~ Elec. Insp.	1			1	75.00
Welders	1	1	1	3	225.00
Belt Repairs					
Instrument Maint.	1			1	35.00
TOTAL MAINTENANCE	16	13	13	42	$3,041.07
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	195.75
Watchmen					
Janitor					
Lamp House Attend.	1	1	1	3	185.63
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver	1			1	61.88
Yard Men	1	1	1	3	185.63
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers Sh. Col.	1	1	1	3	185.63
TOTAL TOP	6	5	5	16	$ 987.05
WASHER					
Foremen	1	1	1	3	172.53
Operators					
Repair & Maintenance	3	3	3	9	607.50
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER	4	4	4	12	$ 780.03
Supervision	8	1	1	10	399.51
TOTAL MINE					$9,813.93
Shift Differential		34.80	73.80		108.60
TOTAL MINE					$9,922.53

TABLE 4-7 -- Conventional Mining System
Sundays

DAILY LABOR COST ESTIMATE
January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen					
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters					
Timbermen					
Ventilation					
Rock Dusters					
Belt Patrol					
Airdox					
Clodmen					
TOTAL PRODUCTION					
NON PRODUCTION					
Foremen					
Surveyors					
Examiners			3	3	$300.00
Cagers					
Trackmen					
Pumpmen					
Timbermen					
Ventilation					
Rock Dusting					
Recovery					
Rock Loading					
Motor & Supply					
Hopper Operator					
Main Belt Patrol					
Belt Extension					
TOTAL NON-PRODUCTION			3	3	$300.00

Date:	Non-Union Men	--
	Union Men	4
Made By:	TOTAL MEN	4

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician					
Shop Foremen					
Unit Repairmen					
Unit Greasers					
Shop Repairmen					
Misc. Repairmen					
Wiremen					
Machinists					
Welders					
Belt Repairs					
TOTAL MAINTENANCE					
TOP					
Foremen					
Hoist Engineer			1	1	$ 87.00
Watchmen					
Janitor					
Lamp House Attend.					
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver					
Yard Men					
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers					
TOTAL TOP					$87.00
WASHER					
Foremen					
Operators					
Repair & Maintenance					
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER					
Supervision					
TOTAL MINE				4	$387.00
Shift Differential			9.60		9.60
TOTAL MINE					$396.60

APPENDIX, EXHIBIT 5

LABOR COSTS FOR HYDRAULIC TRANSPORTATION SYSTEM

Table 5-1 Summary of Labor Costs Per Year

5/5/5 Mining Units

January 1974 Labor Costs
LABOR

200 Days

CODE	DESCRIPTION	Table	Cost Per Day	No Days	Cost Per Year
1	Normal Work Day	5-2	$16,711.06	200	$3,342,212.00
2	Sat. Work Day				
3	Holiday Work Day				
5	Idle Day – Men Wkg	5-3	10,819.19	34	367,852.46
6	Idle Day – All Other	5-4	3,143.39	9	28,290.51
7	Idle Day – Sat. Men Wkg.	5-5	10,249.33	9	92,243.97
8	Idle Day – Sat. after 4 or less		3,209.64	43	138,014.52
9	Sunday – After 6 Work Days				
10	Sunday – After 5 or less wk. day	5-6	396.60	52	20,623.20
11	Holiday – Idle		17,107.66	9	153,968.94
	Company Vacation		3,010.89	10	30,108.90
	TOTAL LABOR				$4,173,314.50

5/5/5 Mining Units

TABLE 5-2 -- Hydraulic Mining System
Straight Time Workday

200 Days

DAILY LABOR COST ESTIMATE

January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator	5	5	5	15	750.00
Loader Helper	5	5	5	15	708.75
Cutter Operator					
Drillers					
Shooters					
More Feeders	5	5	5	15	648.75
Roof Bolters	10	10	10	30	708.75
Timbermen					
Ventilation	5	5	5	15	671.25
Rock Dusters					
Spray Water	5	5	5	15	671.25
Cleanup	5	5	5	15	671.25
Pipeline	5	5	5	15	671.25
TOTAL PRODUCTION	50	50	50	150	$ 6,363.90
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners	2	2	3	7	350.00
Cagers	1	1	1	3	128.25
Trackmen			2	2	85.50
Pumpmen	1	1	1	3	128.25
Timbermen	2		2	4	171.00
Ventilation	3	3	3	9	402.75
Rock Dusting			2	2	85.50
Recovery			4	4	173.00
Hyd. Pumps	3	3	3	9	384.75
Motor & Supply	3	3	3	9	389.25
Hopper Operator					
Main Belt Patrol					
Extension Pipe			12	12	513.00
Dust Control	1			1	35.00
TOTAL NON-PRODUCTION	22	16	39	77	$3,556.35

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen	5	5	5	15	750.00
Unit Greasers			5	5	213.75
Shop Repairmen	3	3	3	9	405.00
Misc. Repairmen	3	3	3	9	450.00
Wiremen			1	1	42.75
Elec. Insp.	1			1	50.00
Welders	1	1	1	3	150.00
Belt Repairs					
Instrument Maint.	2	2	2	6	210.00
TOTAL MAINTENANCE	18	16	22	56	$ 2,720.07
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	130.50
Watchmen					
Janitor					
Lamp House Attend.	1	1	1	3	123.75
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver	1	1		2	82.50
Yard Men	1	1	1	3	123.75
Bulldozer Operator	1			1	43.50
Crane Operator					
Top Repairmen					
Nippers & Droppers Sh. Col.	1	1	1	3	123.75
TOTAL TOP	7	6	5	18	$ 800.28
WASHER					
Foremen	1	1	1	3	172.53
Operators	3	3	3	9	270.00
Repair & Maintenance	3	3	3	9	405.00
Nippers & Droppers	1	1	1	3	123.75
Refuse Disposal	2	2	2	6	261.00
TOTAL WASHER	10	10	10	30	$ 1,232.28
Supervision	8	1	1	10	399.51
TOTAL MINE					$15,072.39
Shift Differential		67.20	134.40		201.60
TOTAL MINE					$15,273.99

Date:		Non-Union Men	57
		Union Men	284
Made By:		TOTAL MEN	341

5/5/5

TABLE 5-3 -- Hydraulic Mining System Idle Weekday

34 Days

DAILY LABOR COST ESTIMATE

January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters	5	5	5	15	708.75
Timbermen					
Ventilation	5	5	5	15	671.25
Rock Dusters					
Belt Patrol					
Airdox					
~~Chxxxx~~ Pipeline	5	5	5	15	671.25
TOTAL PRODUCTION	20	20	20	60	$2,913.90
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners	2	2	3	7	350.00
Cagers	1	1	1	3	128.25
Trackmen			2	2	85.50
Pumpmen	1	1	1	3	128.25
Timbermen	2			2	85.50
Ventilation	3	3	3	9	402.75
Rock Dusting					
Recovery			4	4	173.00
Rock Loading					
Motor & Supply	3	3	3	9	389.25
Hopper Operator					
Main Belt Patrol					
~~Sxx~~ Extension Pipe			6	6	256.50
Dust	1			1	35.00
TOTAL NON-PRODUCTION	19	13	26	58	$2,744.10

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen	5	5	5	15	750.00
Unit Greasers					
Shop Repairmen	3	3	3	9	405.00
Misc. Repairmen	3	3	3	9	450.00
Wiremen					
~~Mxxxxxx~~ Elec. Insp.	1			1	50.00
Welders	1	1	1	3	150.00
Belt Repairs					
Instr. Maint.	2	2	2	6	210.00
TOTAL MAINTENANCE	18	16	16	50	$2,463.57
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	130.50
Watchmen					
Janitor					
Lamp House Attend.	1	1	1	3	123.75
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver	1			1	41.25
Yard Men	1	1	1	3	123.75
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers	1	1	1	3	123.75
TOTAL TOP	6	5	5	16	715.53
WASHER					
Foremen	1	1	1	3	172.53
Operators					
Repair & Maintenance	3	3	3	9	405.00
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER	4	4	4	12	$ 577.53
Supervision	8	1	1	10	399.51
TOTAL MINE					$9,814.14
Shift Differential		35.20	68.40		103.60
TOTAL MINE					$9,917.74

Date:	Non-Union Men	57
	Union Men	149
Made By:	TOTAL MEN	206

TABLE 5-4 -- Hydraulic Mining System
Idle Weekday, No Work

9 Days

DAILY LABOR COST ESTIMATE

January 1974 Rates

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$862.65
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters					
Timbermen					
Ventilation					
Rock Dusters					
Belt Patrol					
Airdox					
Clodmen					
TOTAL PRODUCTION	5	5	5	15	$862.65
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners					
Cagers					
Trackmen					
Pumpmen					
Timbermen					
Ventilation					
Rock Dusting					
Recovery					
Rock Loading					
Motor & Supply					
Hopper Operator					
Main Belt Patrol					
Belt Extension					
Dust	1			1	35.00
TOTAL NON-PRODUCTION	7	3	3	13	$745.10

Date:	Non-Union Men	57
	Union Men	3
Made By:	TOTAL MEN	60

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen					
Unit Greasers					
Shop Repairmen					
Misc. Repairmen					
Wiremen					
Machinists					
Welders					
Belt Repairs					
Instr.	2	2	2	6	210.00
TOTAL MAINTENANCE	5	4	4	13	$ 658.57
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	130.50
Watchmen					
Janitor					
Lamp House Attend.					
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver					
Yard Men					
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers					
TOTAL TOP	2	2	2	6	$ 303.03
WASHER					
Foremen	1	1	1	3	172.53
Operators					
Repair & Maintenance					
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER	1	1	1	3	$ 172.53
Supervision	8	1	1	10	399.51
TOTAL MINE					$3,141.39
Shift Differential		.80	1.20		2.00
TOTAL MINE					$3,143.39

5/5/5 Mining Units

TABLE 5-5 -- Hydraulic Mining System
Saturday -- Full Maintenance

9 Days

DAILY LABOR COST ESTIMATE

PRODUCTION	1	2	3	Tot.	Cost
Foremen	5	5	5	15	$ 862.65
Loader Operator					
Loader Helper					
Cutter Operator					
Drillers					
Shooters					
Shuttle Car Operators					
Roof Bolters					
Timbermen					
Ventilation					
Rock Dusters					
Belt Patrol					
Airdox					
Clodmen					
TOTAL PRODUCTION	5	5	5	15	$ 862.65
NON PRODUCTION					
Foremen	3	3	3	9	575.10
Surveyors	3			3	135.00
Examiners	2	2	3	7	525.00
Cagers	1	1	1	3	192.38
Trackmen					
Pumpmen	1	1	1	3	192.38
Timbermen	2		2	4	256.50
Ventilation	3	3	3	9	604.13
Rock Dusting					
Recovery			4	4	259.50
Rock Loading					
Motor & Supply	3	3	3	9	583.88
Hopper Operator					
Main Belt Patrol					
Extension Pipe			6	6	384.75
Dust Control	1			1	35.00
TOTAL NON-PRODUCTION	19	13	26	58	$3,743.62

Date:	Non-Union Men	57
	Union Men	104
Made By:	TOTAL MEN	161

MAINTENANCE	1	2	3	Tot.	Cost
Chief Electrician	2	1	1	4	$ 276.04
Shop Foremen	1	1	1	3	172.53
Unit Repairmen	5	5	5	15	1,125.00
Unit Greasers					
Shop Repairmen	3	3	3	9	607.50
Misc. Repairmen	3	3	3	9	675.00
Wiremen					
Elec. Insp.	1			1	75.00
Welders	1	1	1	3	225.00
Belt Repairs					
Instr. Maint.	2	2	2	6	210.00
TOTAL MAINTENANCE	18	16	16	50	$ 3,366.07
TOP					
Foremen	1	1	1	3	172.53
Hoist Engineer	1	1	1	3	195.75
Watchmen					
Janitor					
Lamp House Attend.	1	1	1	3	185.63
Blacksmith					
Carpenter					
Airdox Maintenance					
Truck Driver	1			1	61.88
Yard Men	1	1	1	3	185.63
Bulldozer Operator					
Crane Operator					
Top Repairmen					
Nippers & Droppers Sh. Col.	1	1	1	3	185.63
TOTAL TOP	6	5	5	16	$ 987.05
WASHER					
Foremen	1	1	1	3	172.53
Operators					
Repair & Maintenance	3	3	3	9	607.50
Nippers & Droppers					
Refuse Disposal					
TOTAL WASHER	4	4	4	12	$ 780.03
Supervision	8	1	1	10	399.51
TOTAL MINE					$10,138.93
Shift Differential		34.80	75.60		110.40
TOTAL MINE					$10,249.33