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UNDERGROUND HIGH VOLUME HYDRAULIC COAL MINING

Volume III
The Pittsburg & Midway Coal Mining Co.
Kemmerer Coal Mine

Contract JO233911
B. C. Coal, Ltd.
R. G. Heers, Consultant

BUREAU OF MINES
UNITED STATES DEPARTMENT OF THE INTERIOR

NOTICE

This report was completed under a contract to the U.S. Department of Energy (D.O.E.). Responsibility for the program to which the report relates has recently been transferred to the Bureau of Mines, U. S. Department of the Interior, and the report is made public under a Bureau cover. Inquiries concerning the report should be directed to the Bureau.

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16. Abstract (Limit: 200 words) This volume, one of five in this study, examines the technical and economic feasibility of hydraulically mining the three thickest seams at Pittsburg & Midway's property at Kemmerer, Wyoming. The great thickness of the seams, 30-90 ft., the 20° pitch, the absence of gas, the self-supporting top coal, and the low ash and sulfur of the coal, all contribute to making this property feasible for hydraulic mining.

Access to the coal seams is by 11,000 ft. twin rock tunnels, declining on a 21% grade to a central underground dewatering station where the plus 14M coal is screened off and conveyed out. The minus 14M coal and spent water is pumped to the surface for cleaning and clarification. About 4,000 hp of hydraulic energy is provided to each of the three monitors, one in each seam. This energy, together with an induced caving system, is expected to produce over 1400 tons of raw coal per monitor shift. Only 3% of the coal mined comes from development which is accomplished by roadheader type machines.

Capital cost is estimated at $88 million including interest during construction, for a production of 3 million tons per year. IRR ranges from 18%-27%, based on sales prices from $21-$26 per ton FOR. Overall productivity is estimated at 42 tons per man shift and recovery at 60%. Extensive measures are provided for the prevention, detection, and control of spontaneous combustion. Safety is further enhanced by reduction of dust and by remote controlled extraction of top coal and pillars. Recovery and productivity are improved over any other underground mining method for such thick seams.

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FOREWORD

This report was prepared by R. G. Heers, Mining Consultant, as subcontractor for B. C. Coal, Ltd., of Vancouver, British Columbia, under USBM Contract Number J0233911. (B. C. Coal changed its name to Westar Mining Ltd. in June 1983.) The contract was initiated under the Division of Solid Fuels, Mining and Preparation of the Department of Energy. It was administered under the technical direction of the Twin Cities Research Center with Mr. George Savanick acting as Technical Project Officer. Darlene Wilson was Contract Specialist for the Bureau of Mines. This report expands the work recently completed as a part of this contract during the period February 1, 1982 to February 29, 1984, as submitted by the authors in February 1984. Feasibility studies, more detailed in scope, have been completed on the more favorable sites for hydraulic mining, both in the bituminous and anthracite coalfields.

In the United States there are potentially some 60 billion tons of coal in situ on gradients of more than 5 degrees, in coal seams more than 6 ft. thick, and under less than 3,000 ft. of overburden. These reserves, comprising approximately 20% of the U.S.A.'s underground coal resources to this depth, could in many cases be mined more efficiently, safely, and with better recovery using hydraulic mining methods. This applies especially where the coal seams are thick, steep and/or faulted.

The Federal Government, through its agencies, the Department of Energy and the U. S. Bureau of Mines, has recognized this for a number of years and has prepared or sponsored many investigations on the subject.

The findings from these investigations, together with the successful application of full scale hydraulic mining in other countries, e.g., Japan and U.S.S.R., influenced Kaiser Steel in

1969 and 1970 to open a commercial scale hydraulic mine in the Balmer South area of the Crows Nest Pass coal field near Sparwood, British Columbia, Canada. (B. C. Coal now operates this mine and is the prime contractor for this study.)

The mine, now working at the rate of 1,000,000 tons per year, was an outstanding success from the start. It improved safety, productivity and seam recovery to higher standards and on a more sustained basis than had ever been reached before in North America in similar geological conditions.

To introduce this technology into the United States, and to further develop and broaden the use of this advanced mining technology, the U.S.B.M. planned a phased study program. B. C. Coal (formerly Kaiser Resources U.S. Ltd., successor to Kaiser Steel) was granted a contract in 1980 to review as many potential sites as possible for hydraulic mining and to select the most favorable locations for further study. In that report, Kemmerer Coal Company's underground coal property in Wyoming was considered one of three preferred sites. In this more detailed study, the plans for hydraulically mining this site have been expanded, revised, updated and improved.

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INTRODUCTION

The Kemmerer coal property is located in southwestern Wyoming, near the towns of Frontier and Kemmerer (see Figure 1). The coal bearing formation with the thickest seams is the Adaville (Upper Cretaceous Age), which contains up to 12 seams currently being mined by surface methods near the outcrop. The formation runs some 40 miles north and 50 miles south of Kemmerer's property.

Pittsburg & Midway Coal Company, a subsidiary of Gulf Oil Corporation, purchased the Kemmerer Coal Company in 1980 and controls approximately 9,980 acres, containing more than 2.5 billion tons of reserves in place, of which 170 million tons are recoverable by present open pit methods. To extend the open pit reserves in the location of the 1-U-D Pit, where the thickest seams are located, would require stripping ratios in the range of 7 to 8 bank cubic yards to 1 ton of coal, as well as expensive haulage costs to remove the rock from a deep pit against long adverse grades.

The present surface operation, the Elkol-Sorenson Mine, is located near the town of Kemmerer on a branch of the Union Pacific Railroad. It produces some 4.0 million tons per year, of which 2.5 million tons is purchased by Utah Power & Light for their mine mouth Naughton Generating Station. The balance is shipped to various steam coal markets, such as Texas Gulf Sulphur, Allied Chemical, Oregon Portland Cement Co., etc.

Pittsburg & Midway's vast underground reserves, containing many thick coal seams pitching about 20 degrees, present an attractive future reserve for application of underground hydraulic mining methods.

This plan, some parts of which are conceptual, explores only one alternative mining scenario. The property is so large and the seams so numerous that further study may provide other attractive alternates.

The production and economics are estimated for the three thickest seams down dip from the projected final highwall of the 1-U-D Pit.

Because the seams are so lenticular over relatively short distances, additional exploratory drilling should be done before carrying out a more in-depth and detailed feasibility study. Also, more accurate production and recovery rates could be estimated if the large scale mine could be preceded by a smaller scale "test mine" using a full-scale monitor and development machine. Such a mine would logically be a short term proposition, about one-third the tonnage, and would develop a much smaller area than that in the following plan. Since the operations would be smaller in scale, the economics would not be as attractive as for the full-scale mine.

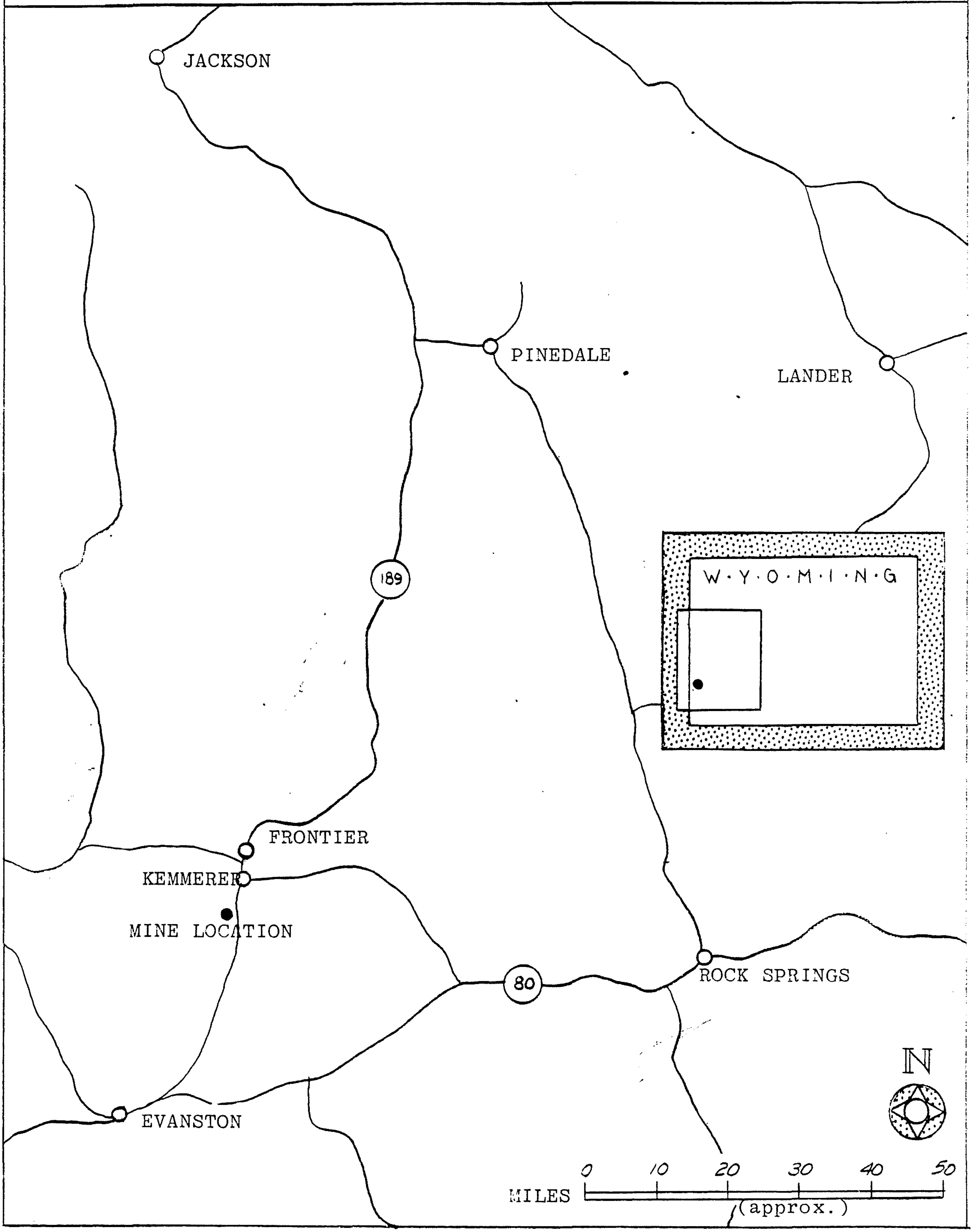
However, to indicate the ultimate potential of large volume underground mining at Kemmerer, the larger mine, taking three seams, was used as the basis of this study. Comparisons can thus be made with alternate plans for deepening the strip mine with its attendant higher strip ratios and longer and more adverse rock and coal hauls.

A preparation plant is provided primarily to remove the rock dilution inherent in the full seam extraction and higher recovery planned.

Special measures are incorporated in the plan to prevent and control spontaneous combustion and, at the same time, avoid large losses of reserves in barrier pillars.

Unless otherwise specified, all tonnage figures are in short tons (2,000 lbs.).

FIGURE 1
MINE LOCATION



SUMMARY

The three thickest seams, Nos. 1, 2 and 3, are planned for mining in this study. Their thicknesses are 90, 30 and 50 ft. respectively, with an average of 57 ft.

The mine will be developed by two rock tunnels, each approximately 10,600 ft. in length, in the Lazeart sandstone. They will start just south of the UP Oregon Short Line Railroad, about a mile northeast of the northern crest of the 1-U-D Pit, decline southwesterly 21%, pass by the northwest corner of the open pit, and terminate at an underground screening and pumping station below the lowest seam, No. 1. Extension of the two tunnels upgrade at 12% will intersect Seams No. 2 and 3.

All coal will flow by gravity to the underground screening station where the plus 1mm coal will be dewatered and then conveyed to the surface. The minus 1mm raw coal and water will be pumped as a dilute slurry to the preparation plant for cleaning.

Special measures to prevent and control spontaneous combustion (sponcom) fires include:

- Constant maintenance of an inert (oxygen deficient) atmosphere in the gob (worked out mine areas).
- Pressure balanced ventilation system (no pressure gradient between the retreat faces and the surface).
- Ready availability of impermeable sand backfill in the form of 1mm x 0 cleaning plant reject, for quickly sealing old workings and/or gob fires with impermeable squeezeproof stoppings.
- Separation of main intakes and returns by sand backfilled seals.
- Continuous retreat of the pillar working faces, 3 shifts per day 7 days per week, minimizing the time any area is exposed to conditions conducive to sponcom.

- Minimizing the number of working places and, in turn, air requirements, and thus, ventilation pressures. In each seam only one production face will be operated at a time. Thus, only one split of air will be used for each seam. This split of air will alternately serve the development and retreat faces. New pairs of rooms will not be developed until the previous pair has been fully retreated and sealed with backfill.
- Separating as much as possible the fresh air in which men are working at the retreat face from the oxygen deficient atmosphere in the gob.

The same production machine will be used for both development and retreat. It will be a crawler mounted continuous miner using a boom type cutter head during development, interchanged for a monitor during retreat. Inasmuch as hydraulic haulage in flumes is used for transportation out from the face, the machine will not have the usual gathering head and conveyor. Instead, a lump breaker will be provided to reduce all coal to a maximum 6-inch top size.

When a pair of rooms is fully developed up the pitch, the cutter boom will be removed and replaced with a monitor for retreat extraction of top and pillar coal. Two such machines in each of the three seams (one producing and one moving back) working 3 shifts per day, 358 days per year, will provide an estimated 3 million tons of clean coal per year.

The dewatering and slurry pumping station is located to provide a 35 year mine life. The life of the mine can, of course, be extended, or production increased, by adding new stations, either to the north or south or further down dip, wherever further exploration proves up the most favorable seams. The life of the mine could also be increased by including reserves in areas where the upper seams thicken.

The highlights of the hydraulic mine planned in this report for the Kemmerer underground reserves are summarized in Table 1 below.

TABLE 1

SUMMARY OF HYDRAULIC MINE

Annual production (Table 8)	
Year 4	1,304,000 clean tpy
Years 5 through 23	3,000,000 clean tpy
Year 24	1,687,000 clean tpy
Years 25 through 35	1,205,000 clean tpy
Reserves, recoverable clean coal (Table 2)	73.8 million tons
Recovery	
Mining	60%
Preparation	82%
Mine life (Tables 7, 7-A and 8)	
Construction period	3 years
Full production reached	Beginning of 5th year
Total	35 years
Production units with hydraulic haulage, used alternately on development & retreat	
Deployed, 2 per seam	6 total
Producing at any one time	3
Productivity of raw coal	
Development with boom cutter (2 machines alternate cutting)	
Raw tons per crew per shift	300
Raw tons per face man shift	43
Retreat with monitor (2 machines alternately produce coal and are moved back) Raw tons/monitor shift	1482
Raw tons per man shift	212
Overall average raw tons/unit shift:	1289
Retreat:development ratio	34:1 average
Water:coal ratio on retreat	1.98:1

	Hourly Workers /Unit Shift	Unit Shifts /Day/Seam
Crew size (Table 6-A)		
Advance	7	As needed
Retreat		
Monitor operation	2	6
Monitor move-back	5	15
Total Retreat	7	21
Productivity, overall	42 clean tons per man	

TABLE 1 (continued)

Working schedule:			
Days per year	358		
Mining shifts per day	3		
Payroll:			
Total employees	318		
Salaried	83		
Hourly	235		
Financial:			
Capital cost, not including interest (Table 10)	$71,194,000		
Capital cost, including interest during construction	$88,281,000		
Selling price/ton:	$21	$23.50	$26
Operating cost/ton clean coal f.o.r. (Table 9)	$14.53	$14.87	$15.22
Rate of Return (Table 11)	18%	23%	27%

GEOLOGY, TOPOGRAPHY AND CLIMATE

The Kemmerer reserves of Pittsburg & Midway Coal Company (P&M) lie in Wyoming's Hams Fork Coal Region (see Figure 2). The coal seams of interest in this study are contained in the Upper Cretaceous Adaville Formation, the most important coal-bearing formation in the Hams Fork Coal Region (Glass, 1977). The Adaville Formation ranges from 2900 to 4500 ft. thick. The basal 1200 ft. of interbedded shales, claystones, and sandstones of this formation contain up to 32 sub-bituminous coals, many of extraordinary thickness.

In the area of this study, the coal seams outcrop along the eastern limb of the Lazeart Syncline and dip westerly between 18 and 20 degrees (see Figures 3 and 5). The syncline is disrupted approximately four miles to the west by the Absaroka Thrust fault. There is some minor faulting in the block selected, but none of the faults are known to be severe enough to impede hydraulic mining.

The topography at Kemmerer is dominated by rolling hills with rock outcropping. The upper soil is poor, supporting a sparse cover of weeds, grasses and shrubs. The climate is semi-arid with annual precipitation of less than 15 inches. Winter temperatures often fall below zero Fahrenheit, and summer temperatures occasionally exceed 100 degrees. There are no perennial streams across the site area, although there are a few springs located in the hills to the west. The mine area drains to the Hams Fork River, which flows through Kemmerer, and the East Fork of Twin Creek, located approximately one mile north of Pit 1-U-D. The elevation ranges from 7,000 to 7500 ft. above sea level.

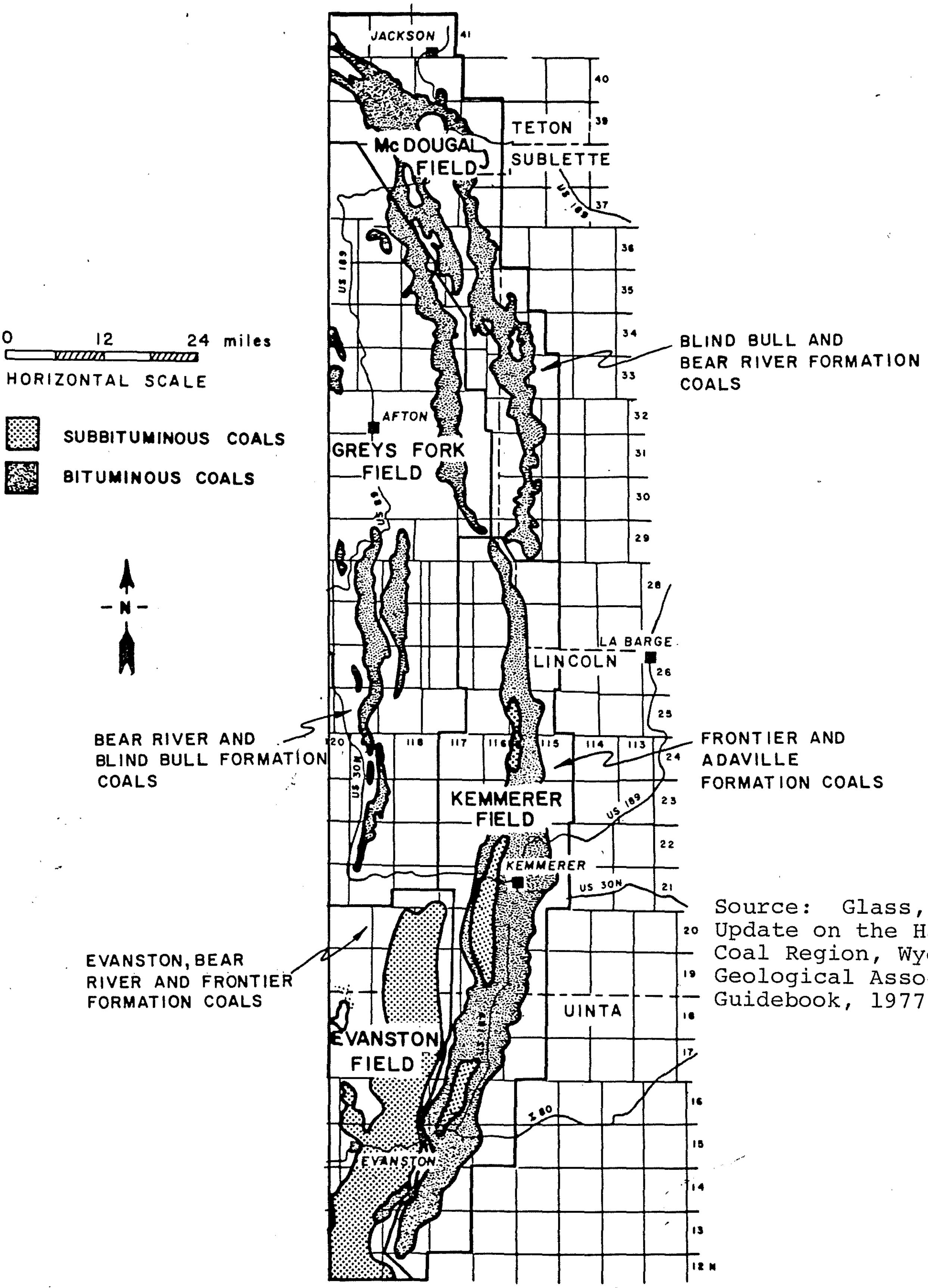
P&M's holdings at the Kemmerer site are divided into two blocks, the South block and the North block. The hydraulic mine planned in this study lies in the South block in the area immediately west and down dip from the projected ultimate highwall of the surface Pit 1-U-D. (See Figure 4, which also shows the location relative to the towns, rail lines, highways, power plants and other existing facilities.)

With regard to the thick seams planned for hydraulic mining, the lowest seam, Adaville No. 1, is the thickest, most consistent and well-known. The No. 1 Seam generally sits almost on top of the underlying Lazeart Sandstone. From the Adaville No. 1 seam, the sub-bituminous coals are numbered upward. These beds vary from a few inches to as much as 50 ft. in thickness. (See Figure 5 for a typical cross-section in the area with the thickest known seams.) At times in the 1-U-D Pit, up to 7 of these coal seams are more than 10 ft. thick, while others range between 4 and 8 ft.

All of these coals pinch and swell in thickness and split and coalesce over short distances (Glass, 1977), so that correlation of the beds is extremely difficult without close drilling, continuous high wall exposure, and/or underground development. However, in the area of the 1-U-D Pit and down dip therefrom, the three lowest seams are the most consistent. Five drill holes down dip from the 1-U-D Pit indicate reasonably consistent thicknesses in Nos. 1, 2 and 3 seams for a strike length of about 6,000 ft.

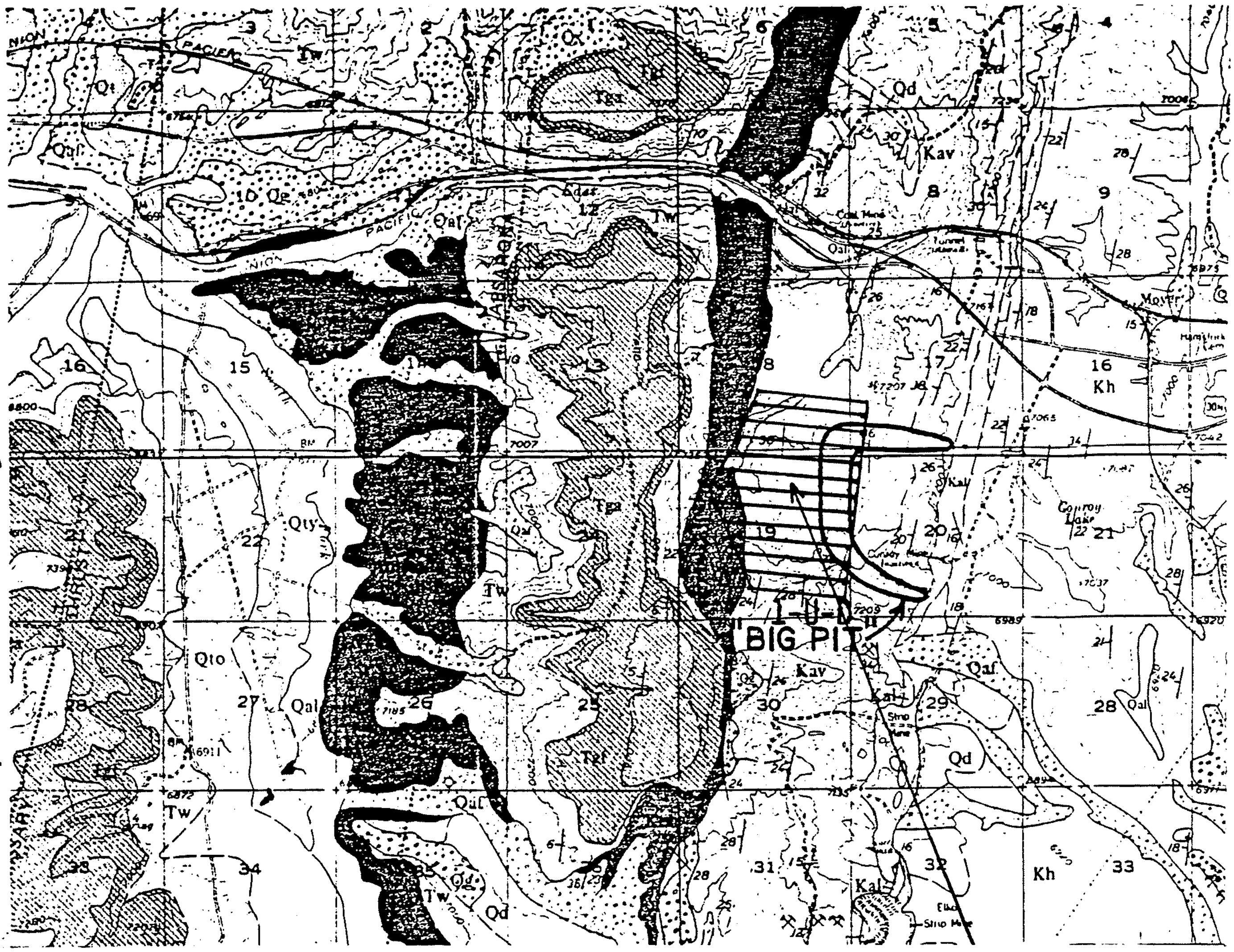
FIGURE 2

HAMS FORK COAL REGION

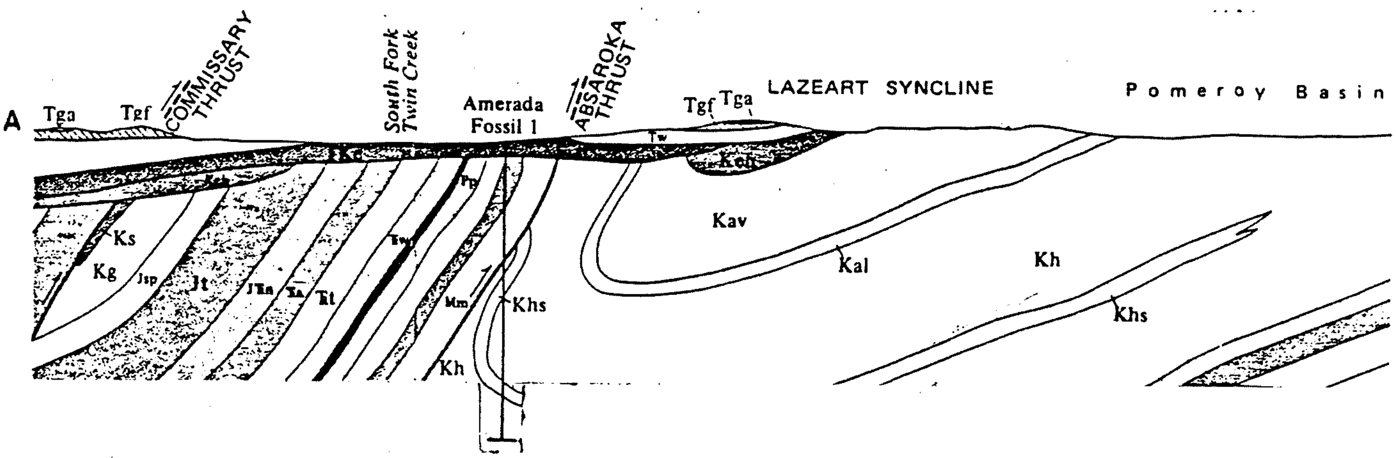


Source: Glass, Gary B., Update on the Hams Fork Coal Region, Wyoming Geological Association Guidebook, 1977.

FIGURE 3
GEOLOGIC MAP
AND
CROSS SECTION A-A'



LOCATION OF PROPOSED HYDRAULIC MINE



SCALE 1"= 1 mile

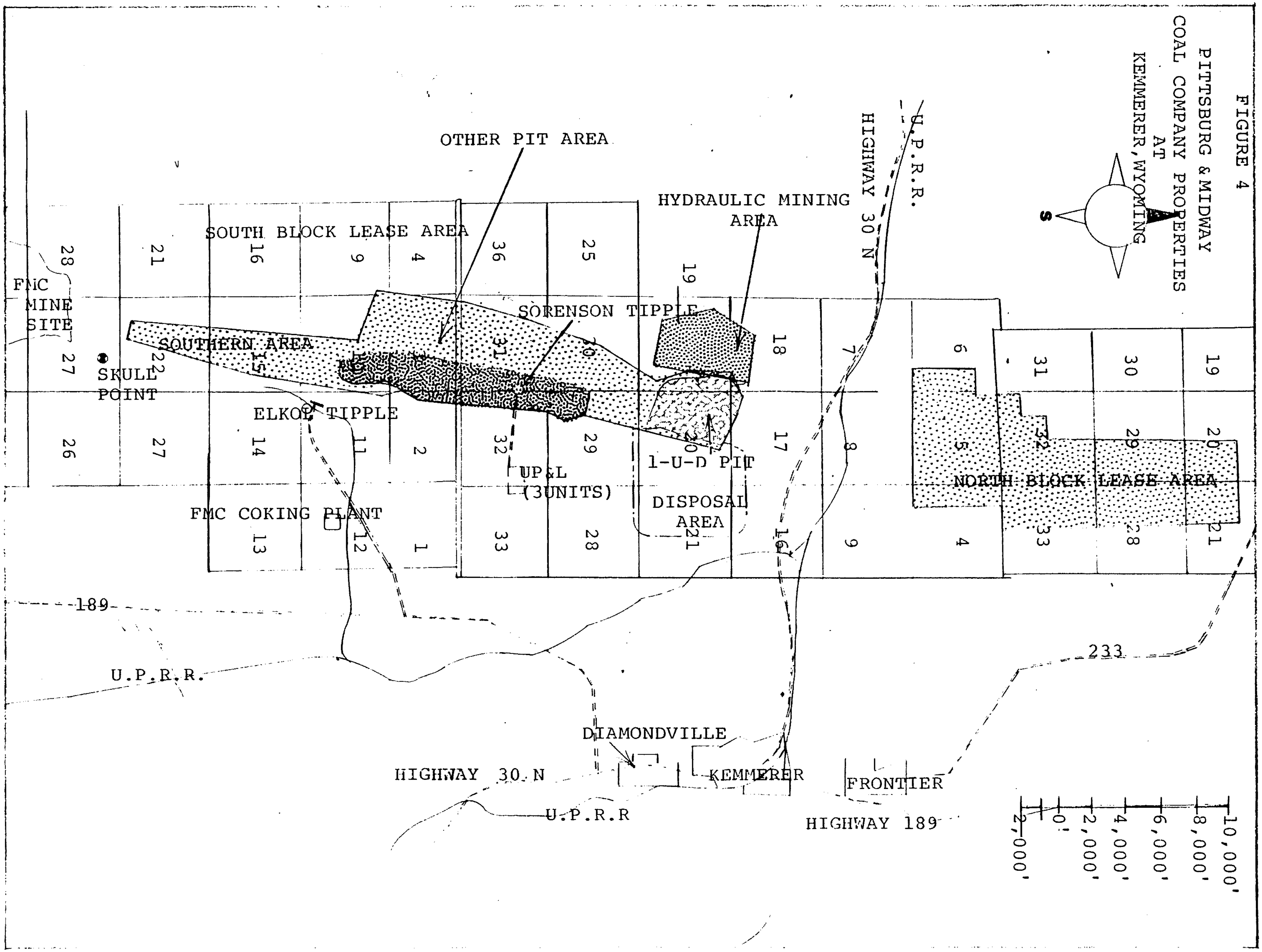
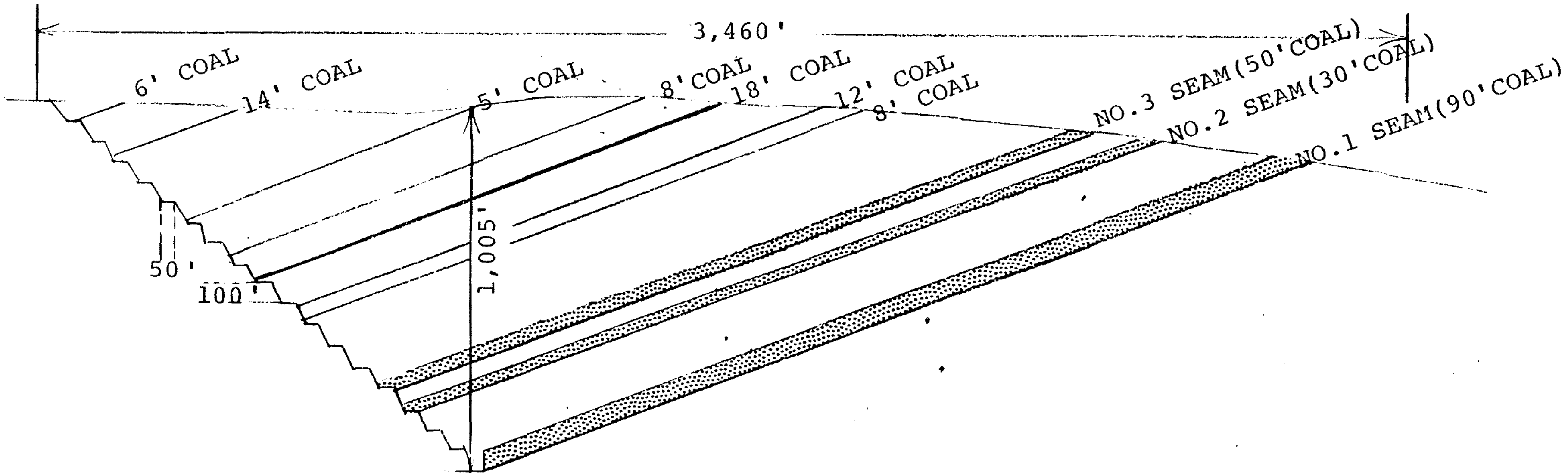


FIGURE 4
PITTSBURG & MIDWAY COAL COMPANY PROPERTIES AT KEMMERER, WYOMING

FIGURE 5

TYPICAL CROSS SECTION IN 1-U-D - PIT,
ADJACENT TO PROPOSED HYDRAULIC MINING AREA
SCALE 1" = 400'



MINING CONDITIONS

The following conditions and their effect on hydraulic mining apply to the three thick seams in the South Block.

Number and Multiplicity of Seams

The hydraulic mine plan for Kemmerer envisions mining the three thickest seams down dip from the 1-U-D Pit starting at the bottom: Nos. 1, 2 and 3. Interburden between seams is variable as indicated by drill hole data and exposure along the highwall. For the purpose of this report the general values of 150 ft. between Seam Nos. 1 and 2, and 50 ft. between Seam Nos. 2 and 3 are used. To minimize pressure interaction between seams, the upper seam retreat line will be kept well ahead of each successively lower seam, and, to the extent possible, pillars will not be left. To avoid pressure concentrations from upper seam pillars, the extraction should be as high as possible, even at the expense of extra dilution. A pillar should only be left if it is thin enough to eventually crush and yield. Main slope aircourse pillars will be columnized from one seam to the next. Further discussion of the pressures created in mining multiple seams and the remedies will be discussed under retreat mining.

Seam Thickness

Seam Nos. 3, 2 and 1 all vary in thickness along the highwall adjacent to the area proposed for mining; however, for simplicity, the average seam thicknesses are assumed as follows:

	Ft.
No. 3 Seam:	50
No. 2 Seam:	30
No. 1 Seam:	90
Total all seams	170

Average thickness used for purpose of this study: 57 ft.

The coal beds pinch and swell in thickness, split, and coalesce over short distances and there is no drilling in the area under consideration. Therefore, seam thicknesses are assumed to average the same as the known information from Pit 1-U-D.

The extraordinary seam thickness favors hydraulic mining. It is the only underground mining method proven to be successful in such thick seams in North America. Coal over 12 ft. is considered too thick for economic single slice longwall systems; and, even though multi-slice systems are being attempted in other countries, they are as yet untried and unproven as a commercial technique in the U.S. Generally speaking, hydraulic mining productivity increases with coal seam thickness because the amount of development and moveback work is decreased. Also, the network of pipe, flumes, power cable, and ventilation works is more a function of area mined than tonnage produced.

<u>Seam Pitch</u>

The predominant pitch is west 18 to 20 degrees, and the strike is approximately N 10° E. Rooms will be driven straight up the pitch, which provides advantages over rooms (or sublevels) driven across the pitch. They are:

- Coal can be extracted on both sides of the room, thus reducing development work by as much as one-half for equal cutting distance.
- Intersections are not required at sharp acute angles.
- Retreat mining can be started much sooner after completion of the screening and pumping station and the ventilation air courses. Development time is reduced.
- Mine layout is simplified.
- Distances for haulage and man travel are reduced.
- Ventilation distances are reduced.

To handle these steep grades in the rooms, monorail haulage will be used, and the mining machines will be equipped with hydraulic jacks to grip the ribs and to provide forward thrust as well as anti-rollback protection. Winches will be provided on machines, and the crawler treads will be equipped with tungsten carbide spikes.

Roof and Floor

The roof and floor of Seams 1, 2, and 3 are interbedded claystones, siltstones, sandstones and carbonaceous shales which, like the coal seams, are lenticular over short distances. The sandstone ranges from moderate strength to fine grained, poorly cemented, friable and weak material. The siltstones are generally hard, well cemented, and of moderate strength. The claystone beds are of more variable texture and strength. They range from moderate strength to carbonaceous clay shales of very low strength which tend to be as fractured as are the coal members. Dames & Moore report the Adaville rock hardness usually ranges from 6,000 to 10,000 psi.

Illustrative of the relative weakness of the rock at Kemmerer is the powder factor in the open pit mine. It is only 0.5 lbs per bank cubic yard (BCY), which compares with Balmer's open pit where the factor is about double.

Based on this rock strength, it can be safely assumed that the flumes in the rooms up the pitch can be cut into the rock with a boom type cutter.

Generally, the rock over the coal is not expected to provide a strong competent roof. Roof spans on pillar extraction, therefore, will need to be somewhat narrower than at other locations with stronger overlying rock.

Because of the relatively lower rock strengths, abutment pressures are not expected to be severe, and the overburden will cave readily to the surface.

The old Elkol underground mines in the No. 1 Seam were worked between 1906 and 1950. The mine openings were developed under top coal which is reported for the most part to have been self-supporting. Only occasional timber supports were said to have been required. Development in this plan will be in the bottom horizon of the seam under top coal.

However, to cope with deeper cover than was experienced in the old mines and the abutment pressures from upper seam workings, the places will be driven narrow and roof bolted. Supplementing this support will be a row of hydraulic props on 10 ft. centers along one side of the flume. They will also support a handrail to provide security for anyone walking or working in the 20 degree room. In addition, single hydraulic props will be installed at the appropriate density at the caving lip and outbye as far as needed. They will be used, in whatever spacing is required, to resist any abutment loads imposed from upper seam mining.

Faults and Intrusions

The Absaroka Fault lies about 4 miles west of the site and, therefore, would not directly affect the hydraulic mine. Well defined major joint sets have been encountered in the surface pits (1-U-D) (Dames & Moore, 1975), and some minor faults could be encountered underground. In the coal block proposed for hydraulic mining, displacements are not expected to be enough to disrupt the hydraulic system planned. There are no known intrusions at Kemmerer.

Overburden

In the area proposed for hydraulic mining, the overburden ranges from zero at the highwall outcrops to 2,200 ft. Mining conditions generally deteriorate as the overburden depth increases. However, the hydraulic mining system is less sensitive to increases in overburden pressure than other mining methods since the width of entries can be decreased with hydraulic haulage, and since the pillars are recovered with a water jet without men or equipment having to work in the immediate area being extracted. Increasing abutment pressures will, in fact, aid productivity from the hydraulic monitor or decrease the pressure and quantity of water required to cut a given amount of coal. However, with comparatively weak rock over the coal, abutment pressure will not be as pronounced as at other locations with strong rock.

The general weakness of the rock, and the ease with which it caves, is beneficial in that it provides natural sealing between seams against movement of air or gob gases. Also, the overburden will tend to seal the worked out seams from the surface. There should be little, if any, communication of ventilation air between seams.

Gas

The old underground Elkol Mine, over its 44 year history, encountered no methane. The well-jointed coal and exposure along the outcrop has apparently allowed gas to drain from the seams. It is possible that, under deeper conditions down dip, gas may be encountered. Further exploration drilling before mining could determine the gas content, if any, at depth. Since methane is not expected, ventilation volumes are keyed to meet the requirements for diesel equipment.

Coal Hardness and Toughness

Experience and practice have shown the coal at Kemmerer to be strong and tough. Top coal in the old mines stood unsupported. The powder factor of 0.45 lbs. per cubic yard of coal also indicates the toughness of the coal. Moreover, the mine operators report a powerful front-end loader cannot dig the fresh coal face without blasting.

Conventional coal hardness tests also confirm the toughness of Kemmerer coal. The Hardgrove test yields an index of 55. The Protodyakonov test, used by the Russians to determine coal hardness in relation to hydraulic mining, shows an index of 4.2.

While most available data shows Kemmerer coal to be strong, unconfined compressive strength tests performed by Dames & Moore, 1975, indicate comparatively low strengths: 1,104 to 1,530 psi. To relate the available data to the ability of the water jet to cut or break the coal at Kemmerer, the following summary shows a comparison between Balmer and Kemmerer coal characteristics:

	Balmer	Kemmerer
Support required for top coal	Steel arches, fully lagged	Little or none
Powder factor for coal in open pit	-0-	0.45
Capability of loading unshot coal with heavy front-end loader	High	Very little
Unconfined compressive strength, psi range:	540-1000	1104-1530*
average:		1317
Protodyakonov "f" average tested:	1.12	4.2**

*Information in report by Dames & Moore, 1975.
**It is noteworthy that, while Kemmerer coal made very few .5 mm fines in the Protodyakonov tests, a high percentage of cubes of approximately 1/8 inch in size was produced.

The first three characteristics listed on the above table are considered most important and override the rather low compressive strength characteristics as reported by Dames & Moore. For the purposes of this report, Kemmerer coal will be viewed as a harder coal requiring a higher amount of energy to mine than used at

Balmer. The required pressure and quantity of water needed to mine the coal are discussed in the section on retreat mining.

Spontaneous Combustion

The Kemmerer coal is highly susceptible to spontaneous combustion (sponcom).

The old underground mine incurred many sponcom heatings and fires. The former underground mine superintendent advises that fires would occur in old rooms and in crosscuts where air leaked around the board and brattice stoppings. Usually a fire would give about 2 weeks notice. To control the problem, pillars were left between every room, and the room necks were quickly sealed. Thus, mining recovery was very low, only 13 to 20%.

Sponcom fires also occur on the surface in loose piles of coal. Therefore, surface stockpiles must be compacted to prevent sponcom. Fires have been reported in the exposed seam outcrops in the open pit operations.

Spontaneous combustion will generally occur in pillared areas and in any mine opening exposed to partial ventilation. Therefore, the mining, ventilation and sealing system planned herein provides for many preventive measures, safeguards and methods of rapidly controlling any sponcom that does occur. These are described in the mining plan.

COAL RESERVES

Total underground reserves at the Kemmerer site are broadly estimated by the company at 2.65 billion tons in place. The reserves planned for hydraulic mining in this study are contained in the three thickest seams immediately down dip from the planned ultimate highwall of the 1-U-D pit, along approximately 6,000 ft. of outcrop. The dip dimension of the reserves is controlled by the location of the dewatering and pumping station which, for this 3 million ton per year plan, is believed to be optimum at about 3,750 ft. down the pitch. This location is under about 2200 ft. of overburden.

As shown in Figure 6, the reserves are bounded on the east by the planned final highwall of 1-U-D Pit, and on the west by the 7 degree main levels running N 30° E and S 10° E from the dewatering station. The main levels average 3200 ft. slope distance down dip from the final highwall. The reserves between the main levels and the highwall in place are 134 million tons. Allowing a coal seam mining recovery of 60%, a dilution of 12% and a washing yield of 82%, total clean saleable coal reserve from this station is 73.8 million tons (see Table 2). Including initial construction of 3 years, with full production reached the beginning of the fifth year, the total mine life is approximately 35 years.

FIGURE 6
UNDERGROUND HYDRAULIC RESERVES
SCALE 1" = 2000'

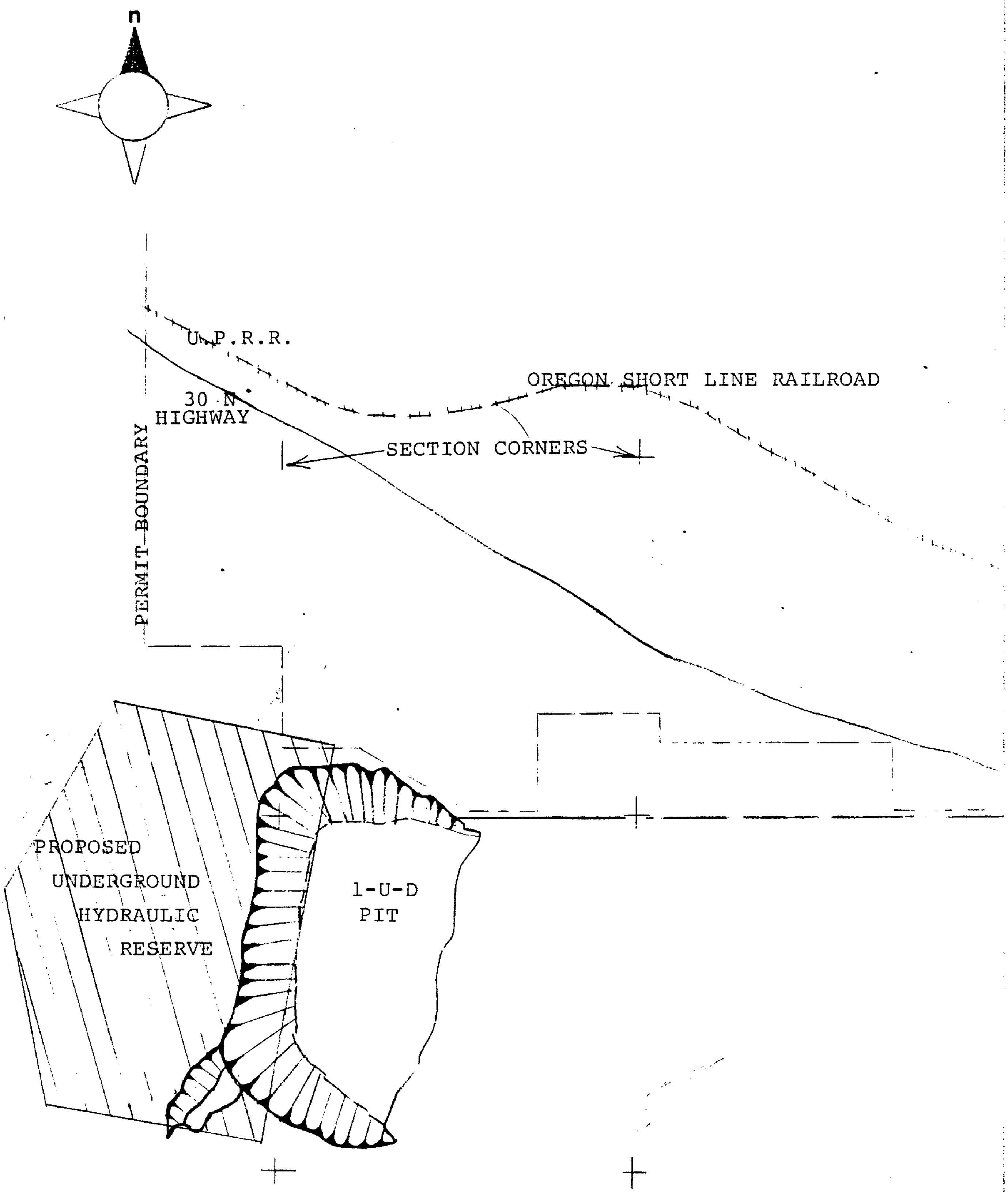


TABLE 2

Coal Reserves Accessible to Dewatering Station

Coal Seam:	No. 1	No. 2	No. 3	Total
Average thickness (ft.)	90	30	50	170
Acres	440	460	470	
Acre feet	39,600	13,800	23,500	
Tons in-place (000)	69,000	24,050	40,950	134,000
Tons of in-place coal recovered, 60% (000)	41,400	14,430	24,570	80,400
Tons of diluted coal recovered, add 12% (000)	46,368	16,162	27,518	90,048
Tons of clean coal at 82% recovery (000)	38,021	13,253	22,565	73,839
Production rate per year clean coal (000) (from Table 8)	1,205	658	1,137	3,000*
Life in years	31.6	20.1	19.8	

Total clean coal recoverable: 73,839,000

Reserve life:

19 years at 3,000,000 tons per year, plus
13 years at 1,205,000 tons per year or more

* When all three seams are in production (years 5 through 23).

Coal Quality

Preparation Plant Recovery

The present raw coal quality as received from open pit mining is shown in Table 3. The average ash shipped without washing from the open pit mine, mostly from lower seams, is 4%. The upper seams are believed to have about the same ash content raw. A disadvantage of hydraulic mining is that the method is less selective than strip mining, and dilution of the coal from roof rock requires a preparation plant. The ash content in the washed product is assumed at 2.5%.

The following rationale is used to estimate the preparation plant recovery.

	Air dry basis
Raw coal ash (diluted with 12% out-of-seam rock added to in-seam coal)	13%
Clean coal ash after washing	2.5%
Ash in reject	60%
Recovery	82%

Coal Characteristics

The seams consist of high-quality, low ash and sulfur sub-bituminous "B" coal.

Table 3 is a typical analysis, as received, from published material printed by Kemmerer Coal Company. This analysis is for run-of-mine coal from surface operations since there is no coal washing facility at the present operations.

TABLE 3

COAL QUALITY

Proximate Analysis (As Received)	Present Raw Product from Surface Mine Average	Present Raw Product from Surface Mine Range	Estimated Washed Product
% Moisture	20.0*	19.0-23.0	21.5**
% Ash	4.0	2.5- 7.0	2.5
% Volatile Matter	35.0	32.0-38.0	35.0
% Fixed Carbon	40.4	36.0-43.0	40.4
% Sulfur	0.6	0.3- 0.7	0.3-0.7
Total	100.0		
Btu	10,000	9800-10,200	10,000
Sulfur Forms			
% Pyritic	.05	.01-.28	
% Sulfate	.01	.00-.01	
% Organic (diff.)	.44	.33-.68	

Mineral Analysis of Ash	% Weight--ignited basis Average	% Weight--ignited basis Range
Silica, SiO_2	36.0	30.0-50.0
Alumina, Al_2O_3	18.0	9.5-24.0
Titania, TiO_2	0.5	0.4- 1.0
Ferric Oxide, Fe_2O_3	5.6	2.9-6.8
Lime, CaO	15.6	3.5-18.0
Magnesia, MgO	4.7	2.5- 7.0
Potassium Oxide, K_2O	0.2	0.1- 1.0
Sodium Oxide, Na_2O	0.7	0.1- 1.5
Sulfur Trioxide, SO_3	16.5	3.2-18.5
Phos. Pentoxide, P_2O_5	0.1	0.09-0.43
Undetermined	2.1	
Total	100.0	

*Surface moisture is 2%

**Surface moisture is assumed to be 3.5%

TABLE 3 (continued)

Ultimate Analysis	Average	Range
% Moisture	20.0	19.0-23.0
% Carbon	58.2	51.0-60.0
% Hydrogen	4.0	3.7- 4.2
% Nitrogen	1.0	0.9- 1.2
% Chlorine	0.02	.01- .07
% Sulfur	0.6	0.3- 0.7
% Ash	4.0	2.5- 7.0
% Oxygen (diff.)	12.18	
Total	100.0	

Fushion Temperature of Ash (^{o}F)	Reducing	Oxidizing
Initial deformation	2010-2345	2155-2350
Softening (H = W)	2055-2500	2200-2450
Softening (H - 1/2W)	2205-2545	2210-2610
Fluid	2360-2700	2495-2800

Hardgrove Grindability	55.0
% Equilibrium Moisture	17.0
Alkalies as Na_2O (dry)	0.04
Silica Value	70.0
Free Swelling Index	Nil
Base: Acid Ratio	0.5
T250 Temp.	2500^{o} F.

Source of the product analysis from the present mine: Kemmerer Coal Company, 1981.

MINE PLAN

Introduction

This preliminary mine plan for certain of the Kemmerer underground reserves is for a large scale mine, recovering the three thickest seams, Nos. 1, 2 and 3, averaging 3,300 ft. down dip from the final highwall of the 1-U-D Pit. The upper seams in this area occasionally show thick lenses but are more prone to pinch, split and coalesce over short distances.

The area proposed for mining should be closely drilled. Thickness isopachs and structural contours for each seam would then permit a more definitive mining plan, which might include the thicker coal lenses above Seam No. 3. Lacking such data at this writing, this plan assumes that Seams 1, 2 and 3 average the same thickness and are separated to the same degree as in the present pit. The generalized thicknesses for the three seams in this area are 90, 30 and 50 ft. respectively, for a total of 170 ft. The average seam thickness is then 57 ft.

The mine layout (see Figures 7 and 8) is very simple. From a screening and dewatering station located about 3,750 ft. down dip from the ultimate highwall of the 1-U-D Pit, main levels are driven in each seam on a 12% grade N. 30° E. and S. 10° E. about 3,200 ft. to the limits of the mining block. From the main levels, rooms are driven straight up the 18 to 20° pitch to the highwall outcrops.

Transportation of workers and material will be typically for a distance of 3 miles. Transportation of coal will be by fluming.

In the interest of conservatism, before proceeding with this large scale, long life mine, a test mine of commercial scale, driven a few thousand feet into the uppermost thick mineable seam, costing in the range of $5 to $8 million, could be

FIGURE 7

MINE LAYOUT PLAN VIEW

SCALE 1" = 2000'

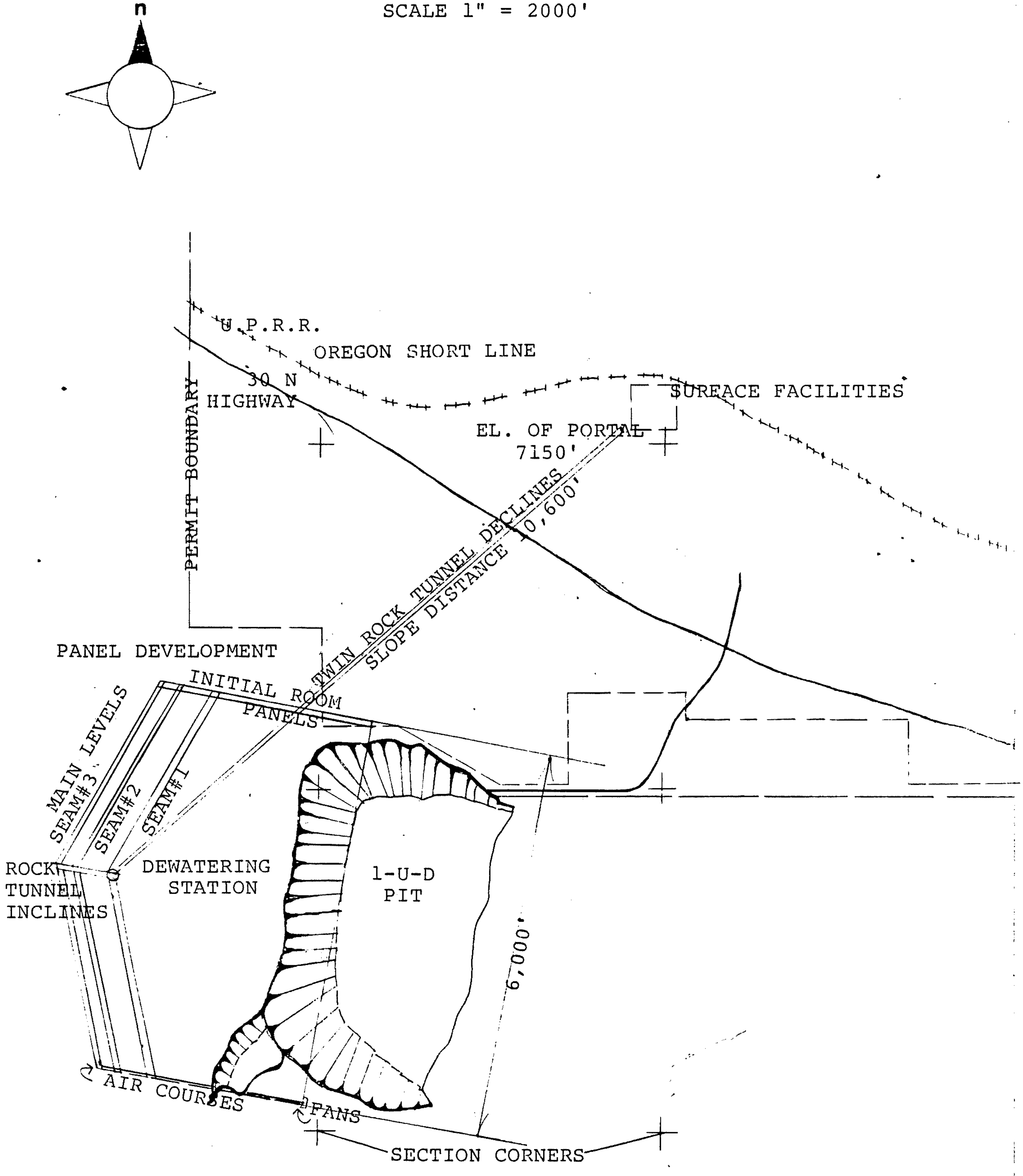
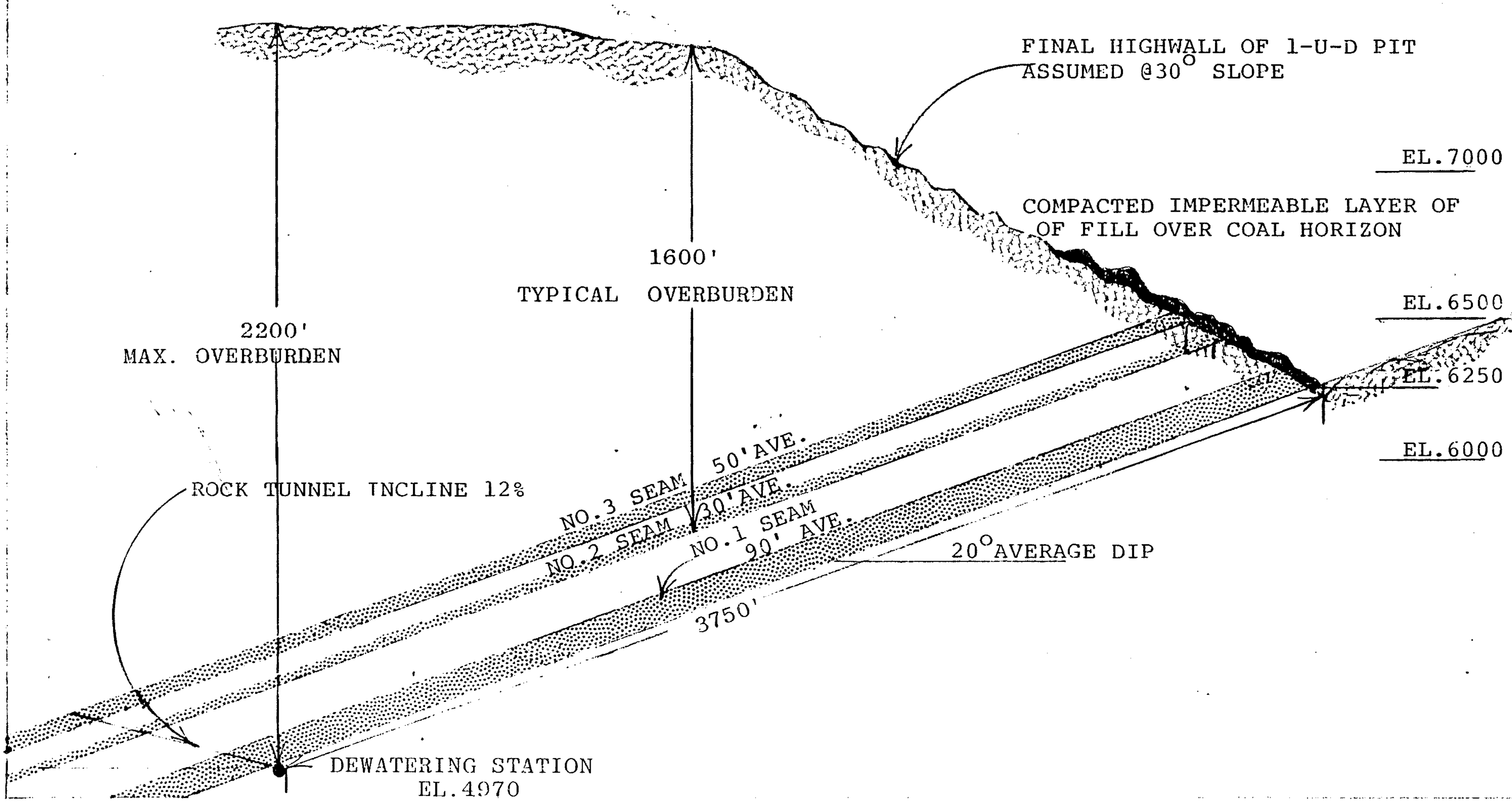


FIGURE 8

MINE LAYOUT PROFILE

GENERALIZED CROSS SECTION OF SEAMS CONSIDERED FOR HYDRAULIC MINING DOWN DIP FROM 1-U-D PIT

SCALE 1"= 500'



developed and mined to determine many of the parameters needed for more accurately estimating production rates, roof control requirements, costs, etc.

Preventing and Controlling Sponcom

To a large degree the mine plan is designed to prevent and control spontaneous combustion (sponcom). Sponcom occurs in susceptible coals when the oxygen supply is sufficient to support oxidation of the coal at a rate which generates heat faster than can be dissipated by the ventilation currents. For each susceptible coal and mining condition there is an incubation period. In the old Elkol Mine the incubation period in worked-out rooms was about 2 weeks.

Following are the measures included in this plan for identifying, preventing and controlling sponcom:

1. Each seam will have its own independent ventilation system. In other words the mine will be compartmentalized into three working areas, each isolated from the other.
2. The ventilation pressure gradient across the gob will be kept as near to zero as possible by using both forced air fans on the intake portals and exhaust fans on the return portals.
3. Obviously, no bleeders will be used in order to further prevent partial ventilation of the gob.
4. An inert oxygen deficient atmosphere will be maintained in each gob. The opening from the room to the area being pillared at the retreat line will be closed by brattice or flaps to the extent necessary to minimize commingling of fresh air with the inert gob atmosphere.
5. The retreat line, where some fresh air commingles with oxygen deficient air from the gob, will be kept moving by working 3 shifts per day, and as close to 365 days per year as possible (358 days assumed). The retreat rate in the

thickest seam, No. 1, will be approximately 4 ft. per shift for a pair of rooms on 80 ft. centers. In two weeks the face would retreat 168 ft. (14 days x 3 shifts x 4 ft. per day).

With the pressure balanced across the gob, that is, no pressure to cause air to flow into or out of the gob, this rate of retreat is believed to be adequate to allow the inert atmosphere from the gob to blanket the coal lost in retreat mining before sponcom can occur. Experience at other mines has shown that if oxygen can be kept below 11%, sponcom will not spread.

Thus, no area in the mine should be exposed to partial ventilation for a long enough time to allow sponcom to get out of control. In the thinner seams the retreat rate will be faster; therefore, there is less danger.

6. Main level and gob seals and stoppings will be constructed with impervious, hydraulically transported and placed sand backfill, 14M x 0 in size, from the preparation plant reject. At least 100,000 tons of this material will be available per year, enough to build five hundred 50 ft. long sand backfilled seals in 7 x 10 ft. crosscuts. Only 100 crosscuts will be driven each year, leaving 80% of the backfill for either backfilling old gob areas, sealing unforeseen fires, or disposal on the surface. In addition to the minus 14M washer reject, the mine mouth power plants make about 80,000 tons per year of fly ash which could be used as an additive to the reject to decrease permeability and give it the quality of cemented backfill. This availability of material provides a ready means of effectively sealing any gob area.

7. The volume of air required will be kept to a minimum by maintaining only two working places in each seam at any one time. The mining machines, two in each seam, will serve to extend the main levels and the rooms. The same machines will then retreat the top coal and pillars by monitoring and

crushing the coal before it enters the flume. While development is in progress, there will be no retreat work; and while retreat is underway, there will be no development headings to ventilate, inspect or maintain. Thus, each seam will require only one split of air. During development only one machine will operate at a time; similarly on retreat.

8. Rooms will be driven in pairs for their full length before connecting with the gob by retreat mining. During retreat, the remainder of the pillar dividing the room pairs (approximately 50%) will be extracted.

9. An early warning system will be provided for continuous monitoring of the working places, return air courses and gob areas for oxygen and carbon monoxide, and any other hydrocarbon gases which indicate imminent sponcom.

10. For the purpose of this study, the final pit highwall where the coal seams outcrop is assumed to be on a 30 degree slope, so that it can be covered with a compacted layer of impervious fill. Should it be necessary, the surface over the deeper mining areas could also be covered with compacted impervious fill. However, because of the soft nature of the rock and the presence of some soft clay layers, this is not believed to be necessary.

11. A supply of inert gas, mostly nitrogen and carbon dioxide, is available from the stack of the mine mouth Naughton plant. Its analysis after scrubbing is:

H_2O	5-6%
N_2	70-80%
CO_2	12.5%
O_2	3.5-4%
Argon	1%
CO	<10 ppm
SO_2	50-60 ppm

This oxygen deficient gas can be further scrubbed to reduce the SO_2, if necessary, and piped into sealed gobs to further insure that the gob is filled with inert atmosphere.

Mine Development

Rock Tunnels

The surface mining operations which are now conducted along the outcrop, and which will continue for many years in the future, would interfere with direct access to the underground reserves by way of the seam exposures on the present highwall position. Therefore, twin rock tunnels, 10,600 ft. long, passing by the northwest corner of the 1-U-D Pit and any northern extension thereof on a bearing of S. 50° W. are planned to access the underground reserves, so that the hydraulic mine will not sterilize reserves now planned for surface mining. These tunnels, for most of their distance, will be in the Lazeart sandstone, the rock member assumed to be the most competent for tunnel support. With this tunnel location, future surface operations can be maintained independent of underground operations. Moreover, the long rock tunnels will bring the coal within 800 ft. of the UP Oregon Short Line railroad northeast of the 1-U-D Pit. The tunnel portal area chosen is at an elevation of 7,150 ft. (See Figure 7.)

A tunnel boring machine (TBM) will be used to drive the tunnels 14 ft. in diameter on 50 ft. centers, downgrade 21%. The approximate 24,000 ft. of tunneling required is more than ample to justify the use of a TBM by reducing development time and costs. The rock would be removed by using a 1,000 ft. extendable belt conveyor which, in turn, would deliver onto the main slope conveyor. After driving the rock tunnels, the TBM could either be retained for future development, such as additional air courses or accessing other seams, or sold. The resale value is estimated at approximately one-third the original cost.

The rock tunnels will intersect the bottom of the screening and pumping station, approximately 60 ft. below the bottom of the No. 1 Seam. At a point about 270 ft. upslope from the bottom, one of the declines will be leveled off to meet the level of the top of the screening station. From this point, it turns up, on a 12% grade bearing N. 80° W., for 800 ft. to intersect Seams 2 and 3. At the bottom of the screening and pumping station, the other tunnel will be extended to accommodate a slurry sump and pump room.

The Lazeart sandstone is reported to be a cliff-forming rock, resistant to erosion. Roof bolts are, therefore, assumed to be adequate for roof support for most of the distance. Zones of weak rock or coal will be supported by the addition of a shotcrete lining, and/or steel arches, if required.

Due to the relatively low compressive strength of the Lazeart sandstone (reported at 6,000 to 10,000 psi), high penetration rates are expected for TBM drivage. The estimated time to drive these tunnels is based on a TBM penetration rate of 130 ft. per day for each heading. The rationale for this rate follows:

Boring time: 50%, or 12 hours per day

Boring rate: sandstone, siltstone, shale and claystone, 6,000 to 10,000 psi: 18 ft. per hour.

Penetration: 12 hrs/day x 18 ft. = 216 ft. per day

Contingency for poor ground conditions, driving crosscuts and water pumping: 40%

Feet per day: 60% x 216 = 129.6 ft. per day

Use: 130

Rock tunnel work is estimated to require 15 months, as follows:

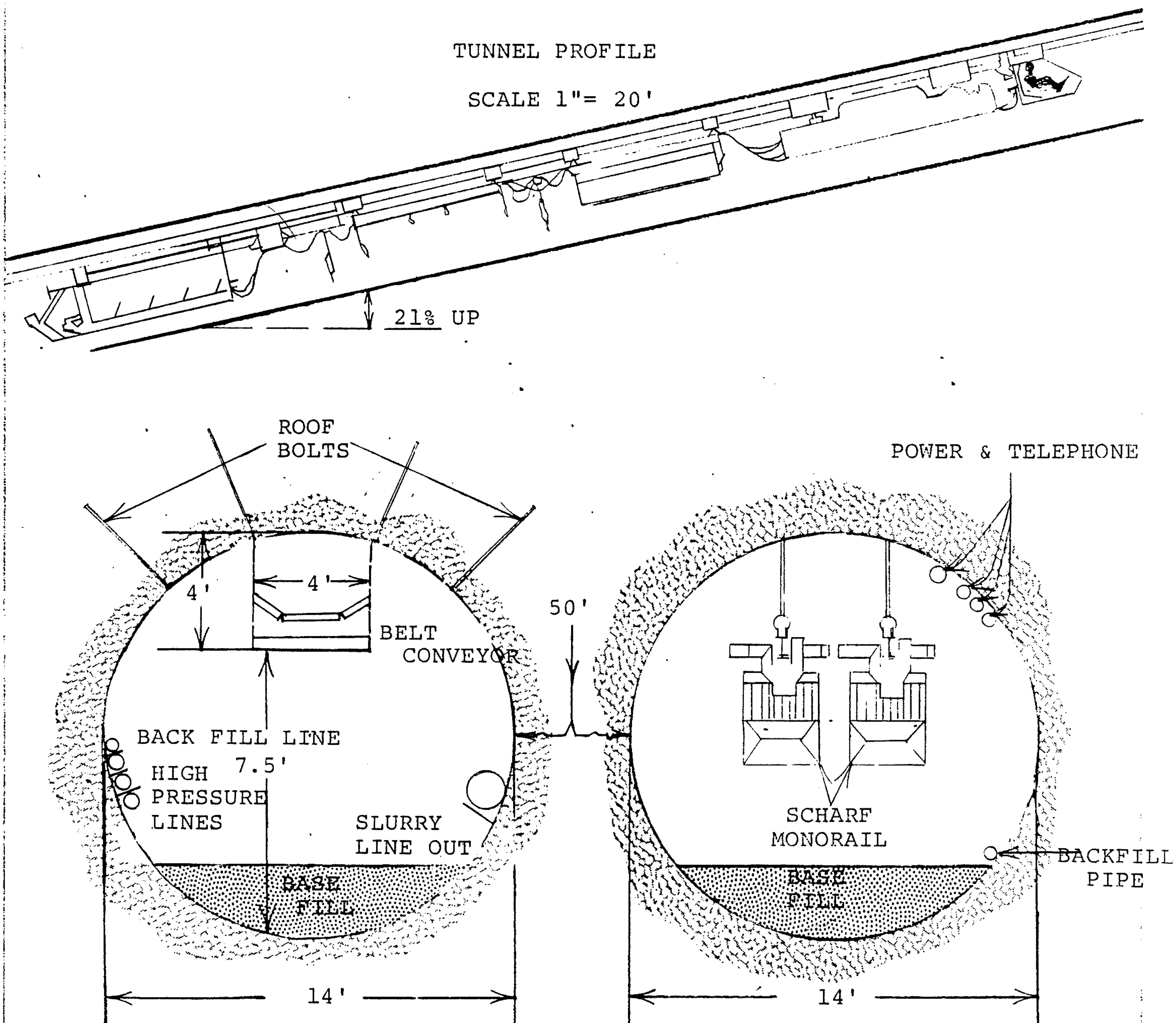
	Working Days
Mobilization:	60
Declines: 10,600 ft. 2 entries x 10,600 ft. ÷ 130 ft./day =	163
Level section to top of dewatering station 250 ft. (curved)	15
Declines for conveyor and ventilation to bottom of station sump and pump room: 850 ÷ 130 =	7
Inclines: 2 x 800 ft. ÷ 130 ft./day =	12
Contingency for turning TBM and moving 3 times from tunnel to tunnel	90
Additional contingency for bad ground	100
	447

Work days per month: 30

Months required: 447 days ÷ 30 days/month = 14.9 months

Use: 15 months

FIGURE 9
ROCK TUNNELS



SCALE 1"= 5'

Dewatering Station

See Figures 10 and 11. After reaching the site of the dewatering station, the connecting entries for the dewatering raise, sump, pump room, ventilation, etc. will be driven. A raise 60 ft. high by an average diameter of 40 ft. to house dewatering screens will be excavated, as well as a second raise with the same approximate volume to serve as an emergency decanting bunker.

The tunnel serving as the sump below the station is sized to provide storage for approximately 500,000 gallons of slurry (water and 14M x 0 coal). Below the sump in a parallel entry at the termination of the lower rock slope, the entry will be enlarged to 14 ft. x 18 ft. for electrical transformers and a pumping station for plunger type slurry pumps. The total distance to be enlarged for the sump and pump room is 750 ft.

Construction and start-up of the dewatering station will require about 11 months.

	Months
• Mobilization	0.5
• Reaming rock tunnel to 14 x 18 ft. for 750 ft.; concurrent with raises and ground support	
• Raises: 120 ft. @ 5 ft./day	1.0
• Ground support of raises	2.0
• Construction concrete and steel	2.0
• Installation of screens, pumps, electrical	2.0
• Contingency	1.0
Sub-Total	8.5
• Extra ground support for incline tunnels through coal seams	2.0
Total	10.5

Use 11 months

FIGURE 10

DEWATERING STATION PLAN
SCALE 1" = 100'

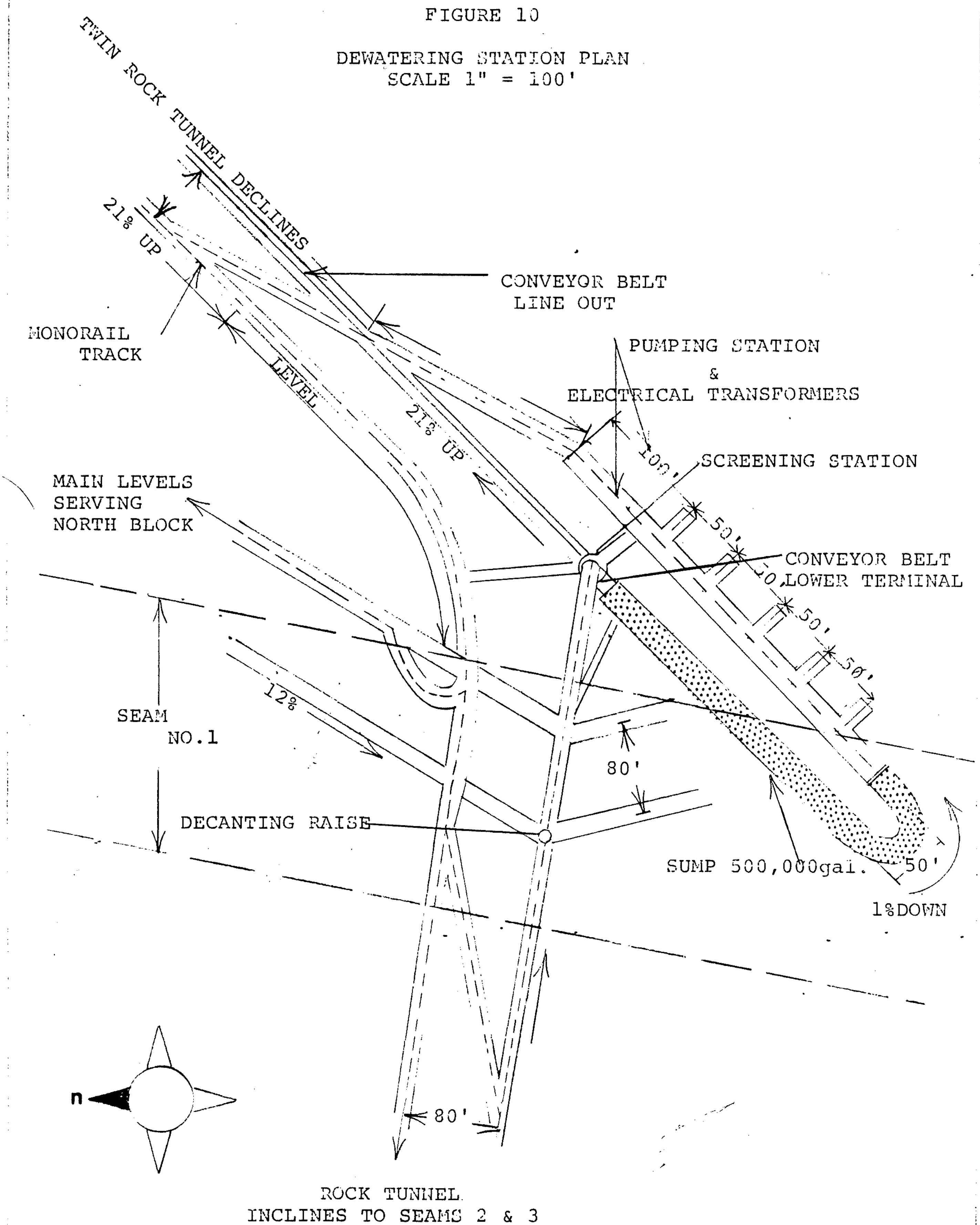
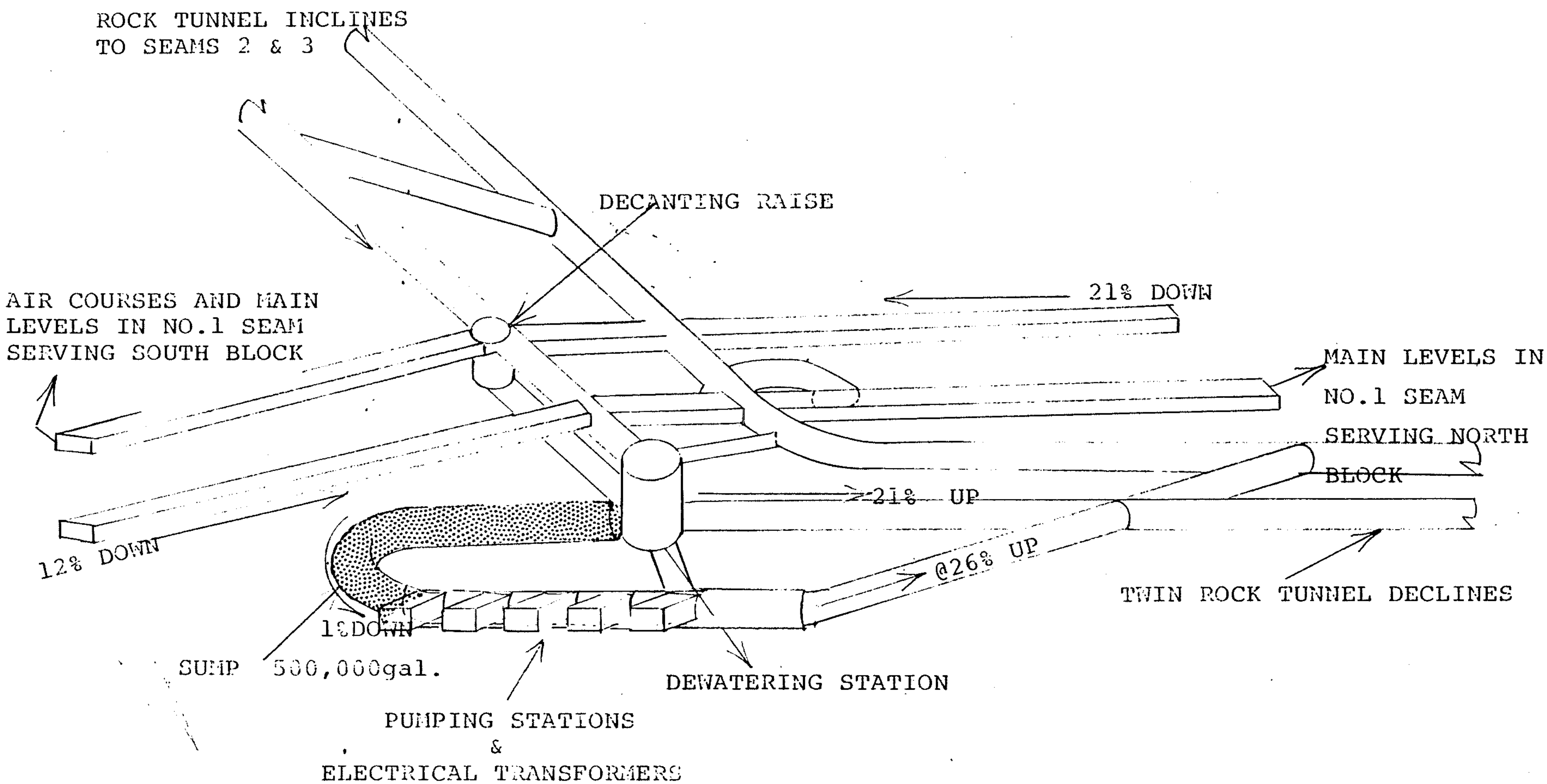


FIGURE 11

DEWATERING STATION PROFILE



Main Level Development

See Figure 12 for the general mine layout for one of the seams.

From the intersection of the rock tunnels and the seam, main levels will be driven about 3,200 ft. on a 12% upgrade to the limits of the mining block. The main levels will be on a bearing about 20 degrees off of strike to provide the necessary grade for reliable fluming. For the purpose of this report, mining is assumed to start on the north end of the block.

Following completion of the rock tunnels and dewatering station, main level development using roadheaders and hydraulic transportation can begin in all of the seams. Priority will be given to the upper seam, No. 3. Development of panels in all seams will be identical, and, with the exception of the main levels, the development headings will be superimposed as much as possible to alleviate interaction problems between seams from pillars.

Mining in the upper seam will precede the middle seam. One full panel will be retreated in the upper seam before retreat mining begins in the next lower seam. If such thick seams were not mined in descending order, undermining would subside, collapse and seriously fracture the upper seams. Therefore, throughout the life of the reserves, mining will be scheduled to maintain recovery of upper seams in advance of the lower seams.

Development in each seam will include two main levels, driven on or near the floor of the seam on 80 ft. centers, at a 12% (6.8°) gradient. Each main level entry will be driven 12 ft. wide by 8 ft. high and supported by roof bolts which are mechanically anchored for their full length, and are easily extracted. See Figure 13 for an example of the type of bolt planned. All main level development will be accomplished by utilizing a boom-type roadheader (continuous miner) as shown in Figures 14-A, B and C. Crosscuts will be driven on 300 ft. centers. Each crosscut will be sealed with sand backfill after it is no longer needed for ventilation.

As the main levels advance, a 10-inch Schedule 80 pipe, an abrasion resistant high molecular weight polyethylene (HMWPE) flume, a monorail, a 3-inch sand backfill line and a 7200 volt power line will be installed. Crosscuts will be on 160 ft. centers, to line up with the inside room of each future room pair.

The coal to be mined per ft. of the two entry main level system is: $\frac{8 \times 12 \; [(2 \times 160) + 68]}{25 \times 160} = 9.3$. Production is estimated at 300 tons per crew shift, equivalent to 32 ft. of advance for both faces per shift. The time required to advance the main levels is thus estimated to be 100 shifts, or 1.1 months.

FIGURE 12
TYPICAL SEAM LAYOUT

SEAM No. 3
SCALE 1"=1,000'

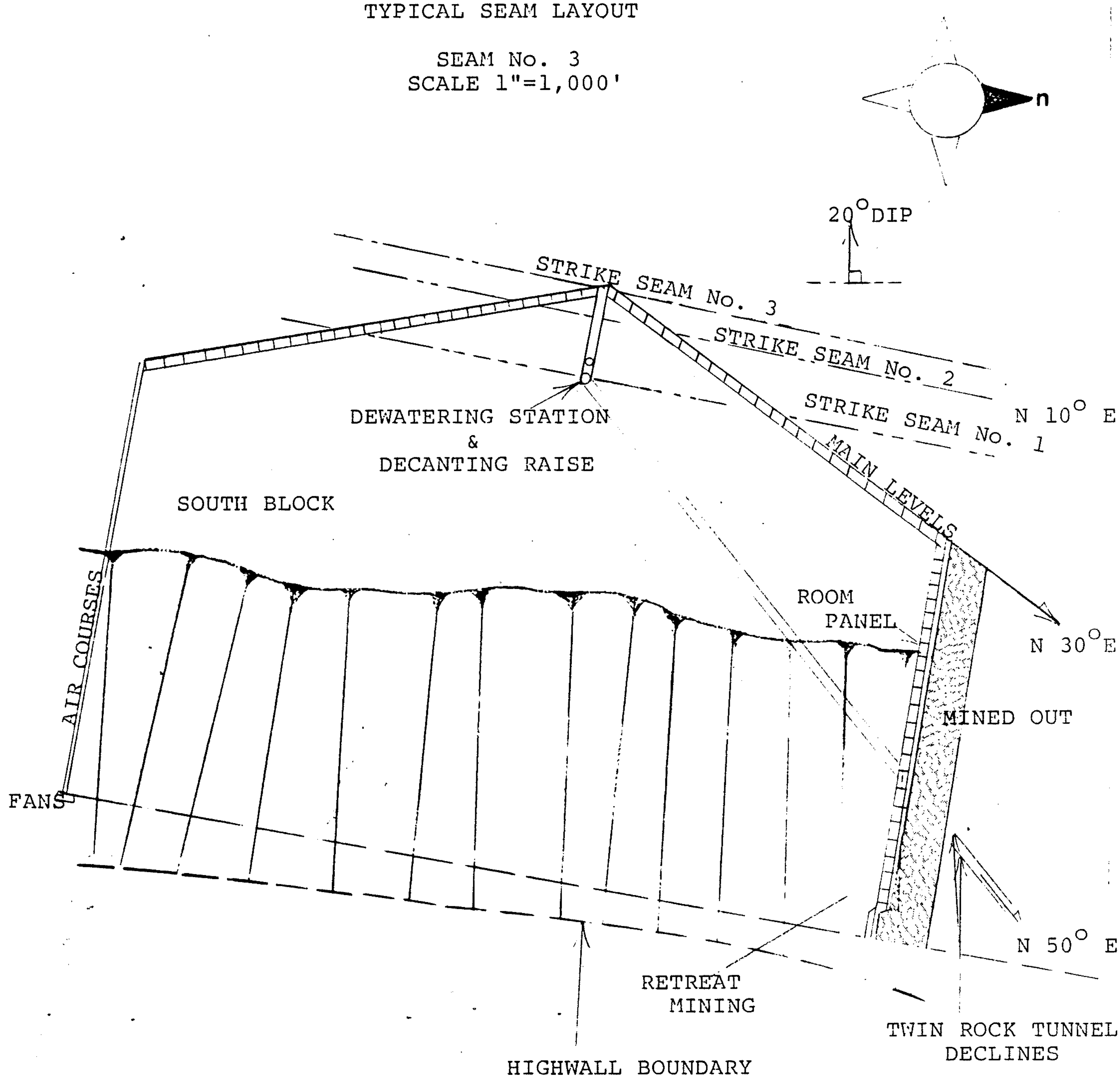
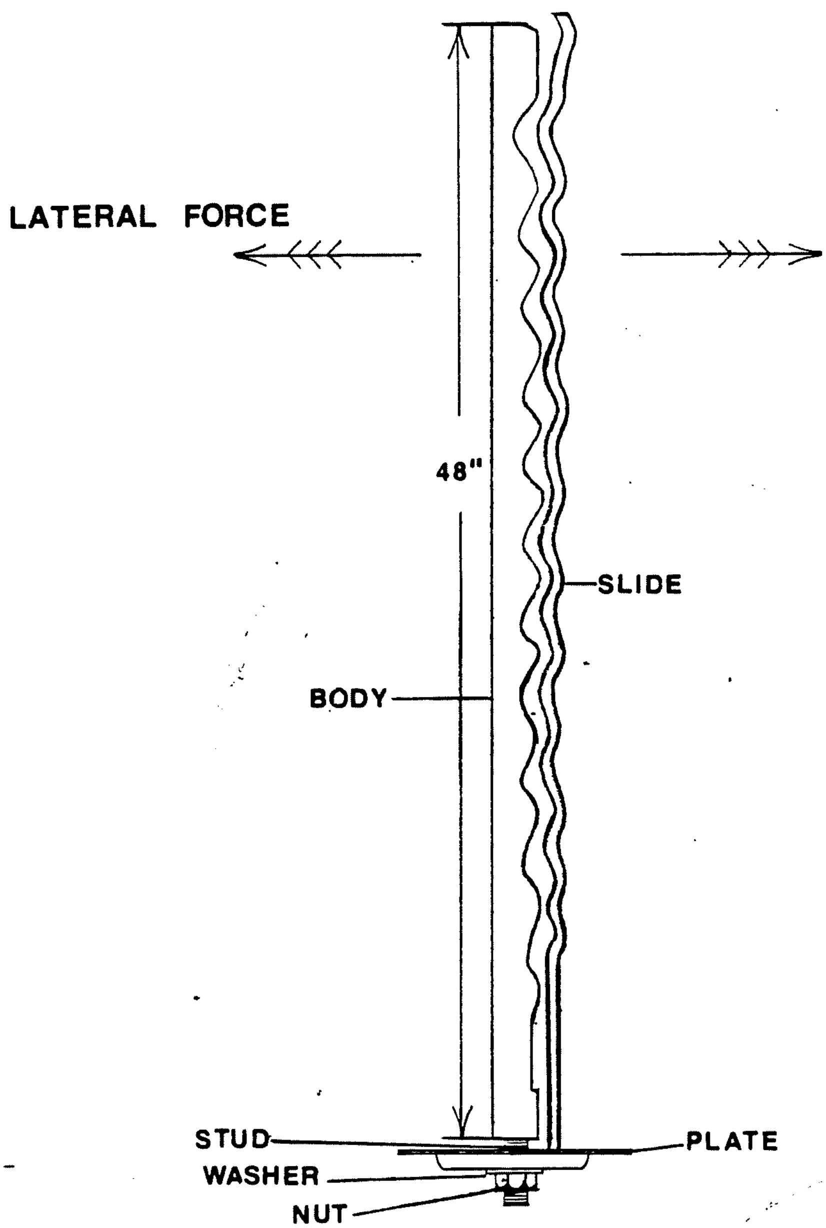


FIGURE 13

ROOF BOLT

(R-X SYSTEM)



REVISIONS

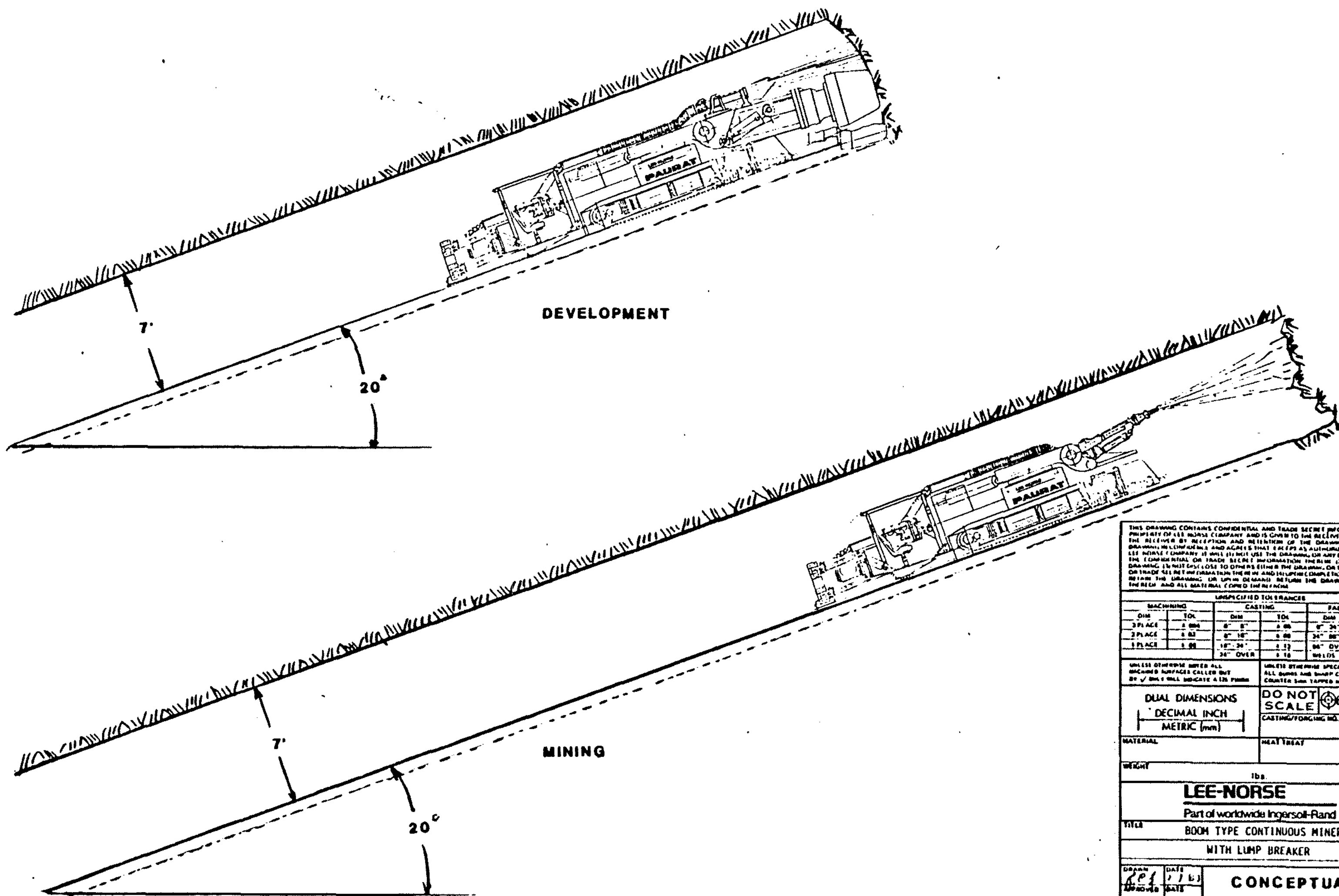
STANDARD TOLERANCE
± .010" FRACTIONAL DIM.
± .005" DECIMAL DIM.

PATT. NO.

MATERIAL

FIGURE 14-A

CONCEPTUAL CRAWLER MONUNTED ROADHEADER MACHINE



UNSPECIFIED TOLERANCES

DUAL DIMENSIONS
DECIMAL INCH
METRIC (mm)

DO NOT SCALE

THIRD ANGLE PROJECTION

CASTING/FORGING NO.

MATERIAL

HEAT TREAT

WEIGHT
lbs.
kgs.

LEE-NORSE
Part of worldwide Ingersoll-Rand

TITLE
BOOM TYPE CONTINUOUS MINER
WITH LUMP BREAKER

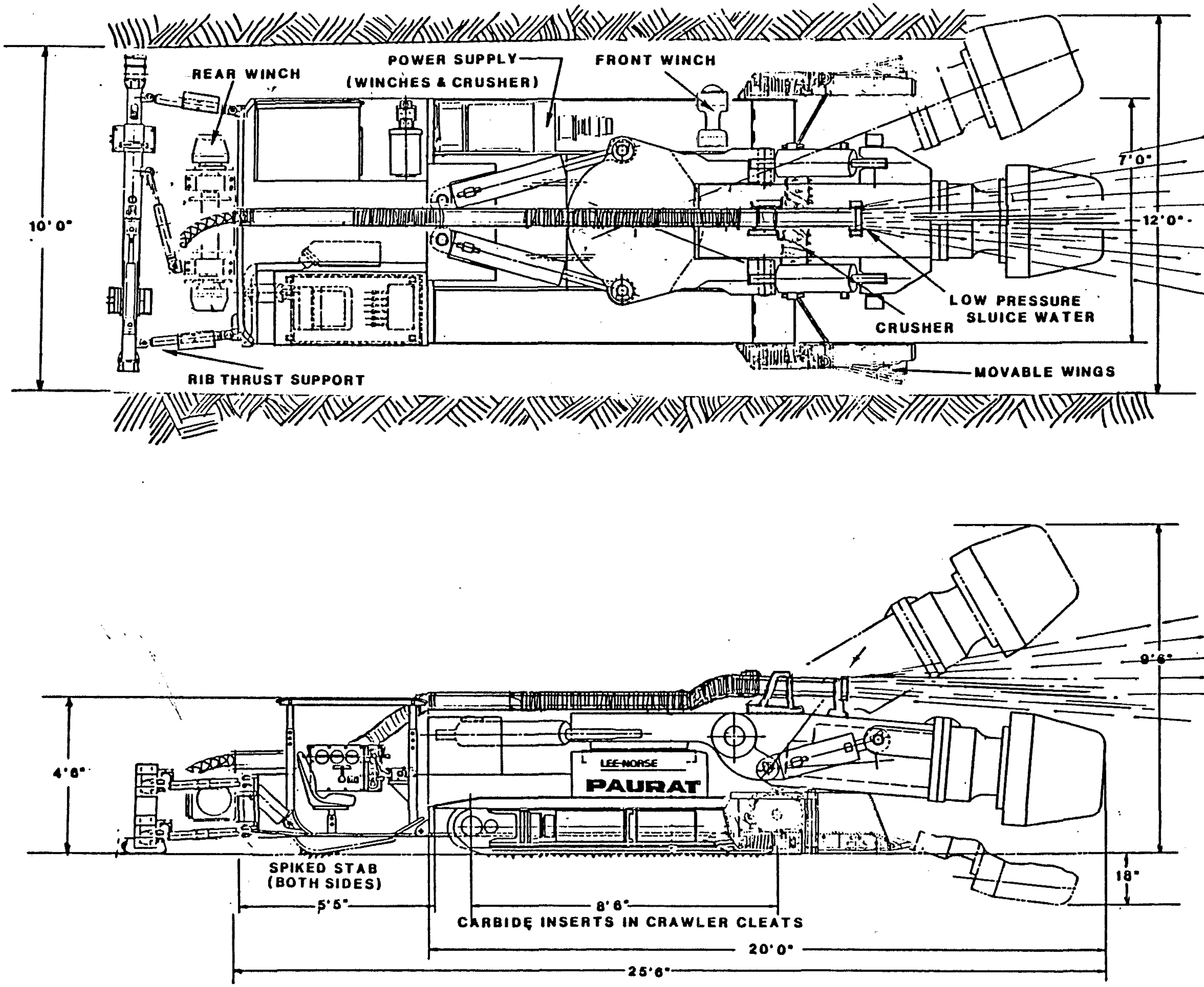
DRAWN
DATE
APPROVED
DATE

CONCEPTUAL

APPROVED
DATE
SIZE C
SHEET
CONT'D ON

FIGURE 14-B

ROADHEADER DEVELOPMENT SEQUENCE



NOTES

1. CUTTERHEAD AND PEDESTAL REMOVEABLE TO ALLOW FOR MOUNTING WATER MONITOR & CONTROLS
2. CRUSHER POWERED BY 75 HP MOTOR (7 INCH SHAFT, 20 INCH PITCH, 25,000 POUNDS OF BIT FORCE)
3. WATER MONITOR TO PROVIDE 17 GPM AT 1500 PSI
4. STABILIZER JACKS BOTH SIDES IN REAR OF MINER EQUIPPED WITH NONSKID CLEATS
5. CRAWLER CHAIN EQUIPPED WITH CARBIDE INSERTS FOR POSITIVE TRACTION
6. 40 INCH THRUST ASSISTS SUPPLIED WITH RIB SUPPORT ARRANGEMENT
7. APPROXIMATE SHIPPING WEIGHT 80,000 POUNDS
8. TWO-SPEED CUTTER MOTOR: 107 OR 215 HP

REVISIONS					
REV	ZONE	DESCRIPTION	DATE	NUMBER	MF

DUAL DIMENSIONS
DECIMAL INCH
METRIC (mm)

DO NOT SCALE

LEE-NORSE
Part of worldwide Ingersoll-Rand

TITLE: BOOM TYPE CONTINUOUS MINER
WITH LUMP BREAKER

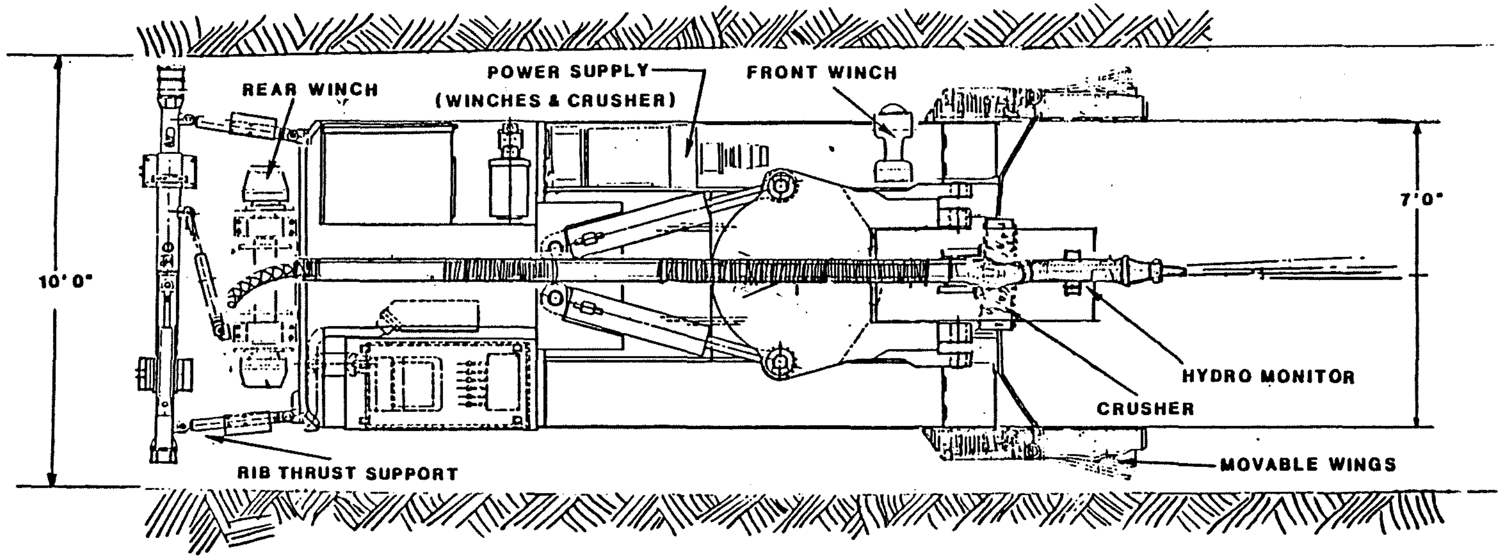
CONCEPTUAL

SIZE C SHEET CONT'D ON

REV	ZONE	DESCRIPTION	DATE	NUMBER	MF

FIGURE 14-C

ROADHEADER WITH MONITOR MOUNT
&
RETREAT MINING SEQUENCE



NOTES

1. CUTTERHEAD AND PEDESTAL REMOVEABLE TO ALLOW FOR MOUNTING WATER MONITOR & CONTROLS
2. CRUSHER POWERED BY 75 HP MOTOR (7 INCH SHAFT, 20 INCH PITCH, 25,000 POUNDS OF BIT FORCE)
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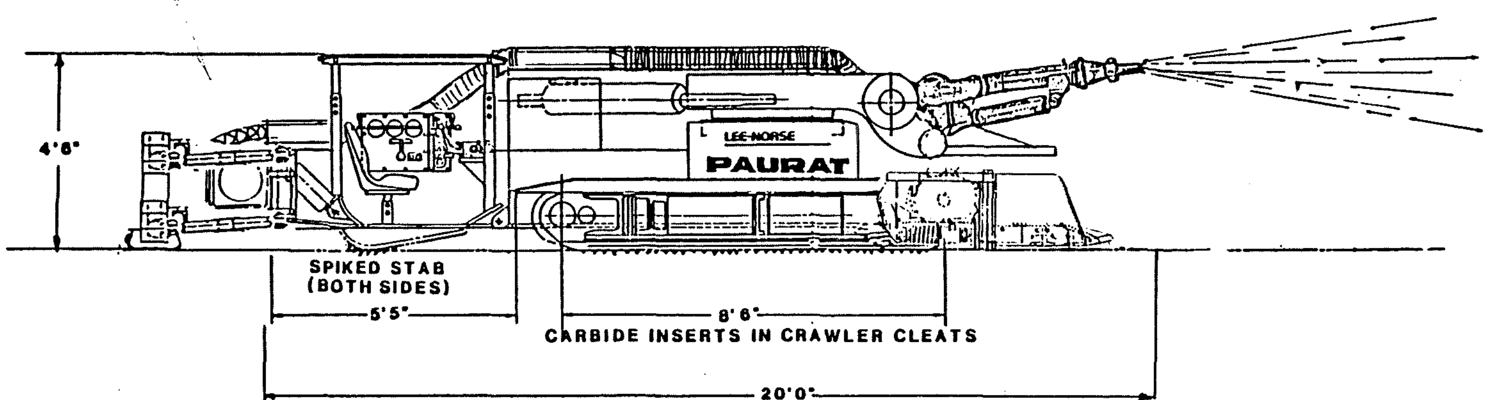
4. STABILIZER JACKS BOTH SIDES IN REAR OF MINER EQUIPPED WITH NONSKID CLEATS
5. CRAWLER CHAIN EQUIPPED WITH CARBIDE INSERTS FOR POSITIVE TRACTION
6. 40 INCH THRUST ASSISTS SUPPLIED WITH RIB SUPPORT ARRANGEMENT
7. APPROXIMATE SHIPPING WEIGHT 80,000 POUNDS
8. TWO-SPEED CUTTER MOTOR: 107 OR 215 HP



DUAL DIMENSIONS
DECIMAL INCH
METRIC (mm)

DO NOT SCALE

LEE-NORSE
Part of worldwide Ingersoll-Rand

TITLE BOOM TYPE CONTINUOUS MINER WITH LUMP BREAKER

CONCEPTUAL

56

Room Development

Pairs of rooms with a cross-section 7 ft. high and 10 ft. wide will be driven straight up the 20 degree pitch (see Figure 15). Crosscuts will be on 300 ft. centers and center-to-center distance between rooms will be 80 ft. Each pair of rooms will develop a panel of coal 160 ft. wide and on average about 3300 ft. long.

Based on an average seam thickness of 57 ft. and assuming a mining recovery of 60%, rock dilution of 12%, and washer recovery of 82%, the recoverable clean coal developed by each pair of rooms will be approximately 660,000 tons.

Transportation of men and material will be by monorail.

In each pair of rooms, connecting crosscuts will provide ventilation, but there will be no connections from any pair of rooms to the gob area, except at the retreat line. The main levels will be sealed from the gob by sand backfill to create an impervious barrier to reduce the risk of sponcom.

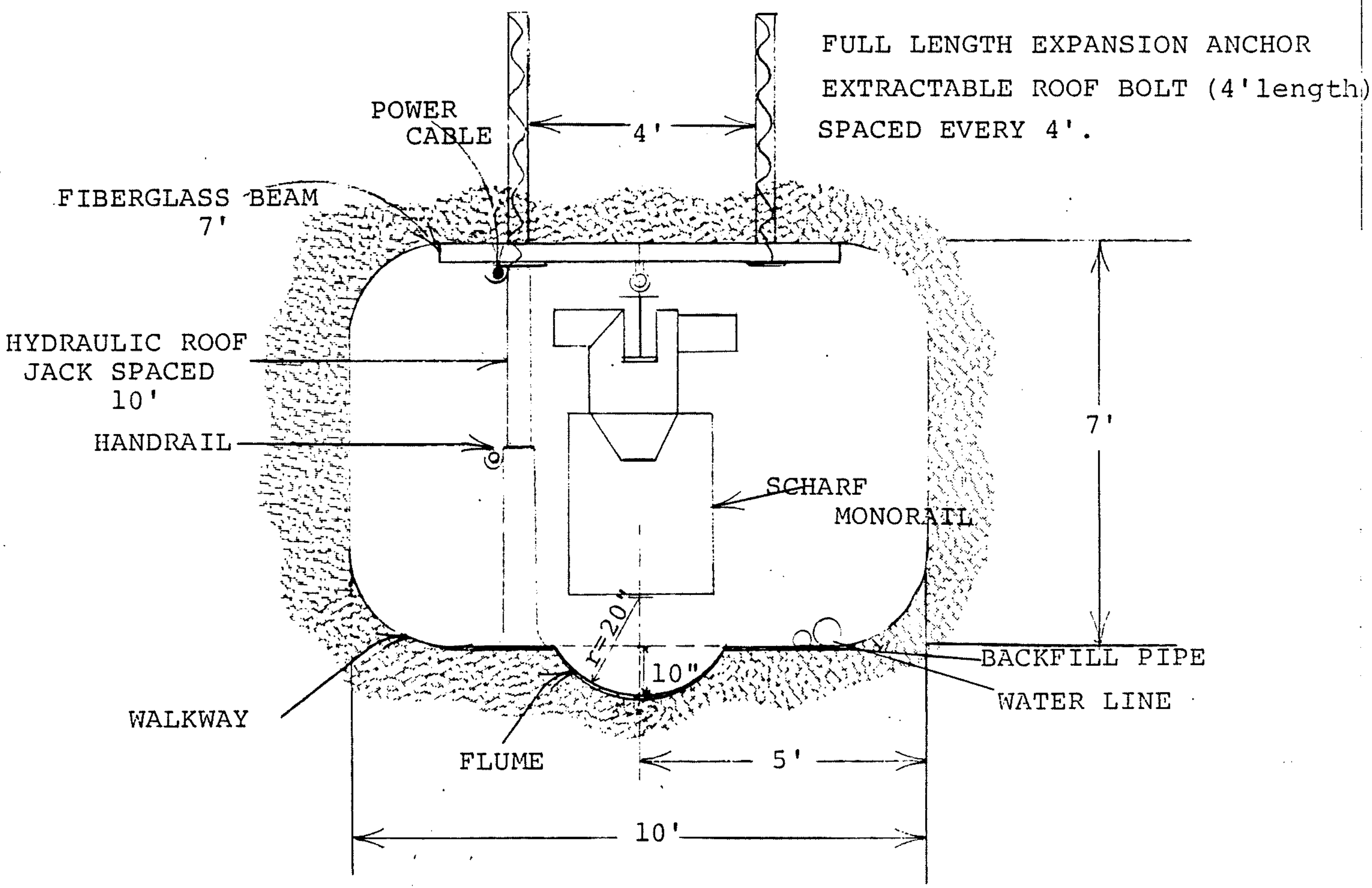
The rooms, as well as the main levels, will be driven by a crawler mounted roadheader machine of the type shown in Figures 14-A, B and C. The distinctive features of this machine are:

1. The cutting boom can be quickly removed and replaced with a monitor when the rooms reach the outcrop.
2. A lump breaker, as in a feeder-breaker, will be provided to insure that oversize pieces of coal or rock (plus 6" to 8") do not enter the flume.
3. There will be no gathering head, loading device, or conveyor through the machine. Hydraulic transportation makes these components unnecessary.
4. Stelling jacks will be provided to grip the ribs and provide security against roll-back. Forward thrust jacks will push the machine forward while sumping in the cutter head.

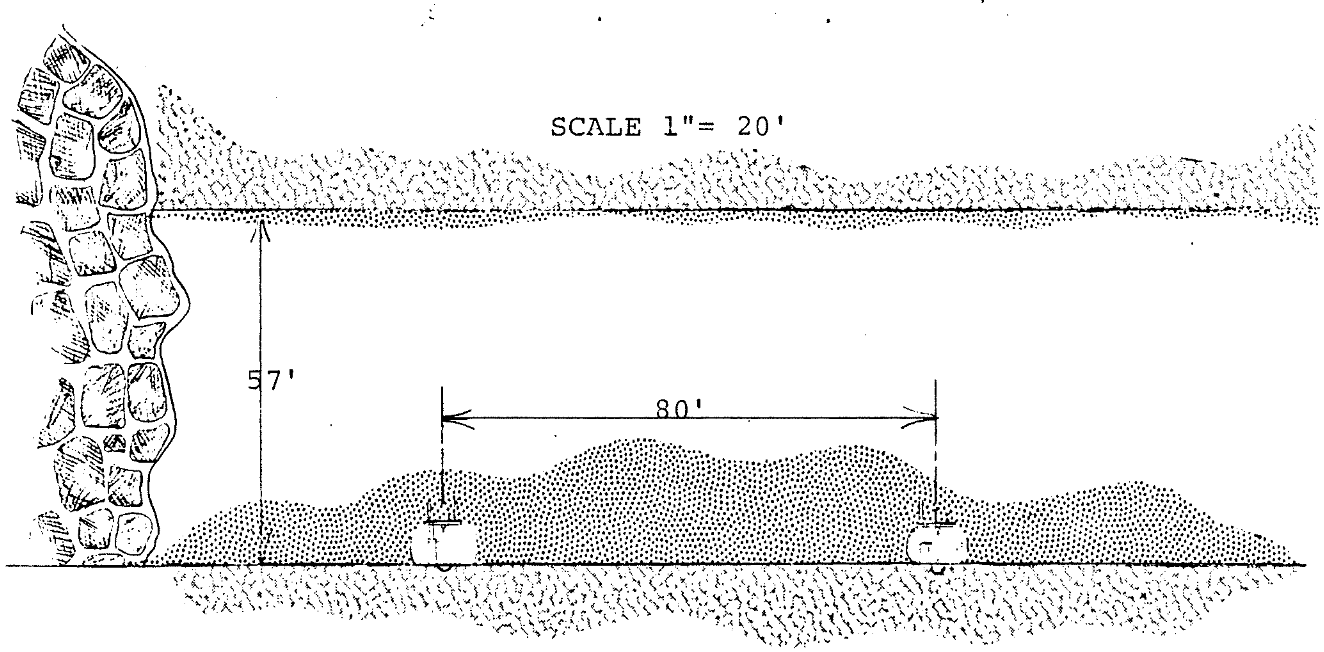
FIGURE 15

ROOM CROSS SECTION

SCALE 1" = 3'



SCALE 1"= 20'



5. The crawler treads will be equipped with tungsten carbide spikes to provide a secure footing on the steep 20^{o} grades.
6. A winch will be provided to pull pipes, roof bolts, hydraulic jacks, ventilation tubing and other supplies into place.
7. Cutting ability will be approximately 5 tons per minute using a 170 kw cutter motor and a 40" diameter cutting head.

Each of the two headings in a pair of rooms will be equipped with a machine, but only one of the machines will be operated at a time. While coal is being cut in one place, the other machine will be advancing the pipe, monorail, ventilation tubing, power cable and roof bolts. Except for driving crosscuts, the mining machines will not move from place to place.

The boom cutter has the capability of cutting a flume in the floor rock. For the purpose of this report, the rock is assumed to be strong enough to resist excessive erosion for the life of the room (equating to a tonnage of 660,000 tons in the case of Seam No. 1). Therefore, steel or plastic flumes would only be required in the main levels.

The roof will be supported by reuseable 7 ft. fiberglass beams bolted to the roof. The bolts will be anchored mechanically over their full length and, therefore, will be easily extractable (see Figure 13). Bolts will be installed two per row (per beam) with rows on 4 ft. centers. Supplementing the roof bolts and beams will be a row of 22 ton aluminum hydraulic props on 10 ft. centers on one side of the flume. The props will also support a hand rail to provide security for workers moving on the 20^{o} gradient and to guard against their falling into the flume. Anyone required to work on the flume side of the guard rail while fluming water is flowing would be required to wear a safety rope to prevent being swept down the flume in case of an accidental fall or misstep.

Development Productivity

Development tonnage mined per foot of panel (a pair of rooms) is estimated as follows:

Tons of raw development coal mined per ft. of heading:
(7 ft. x 10 ft. ÷ 25) x 1.12 for dilution = 3.14 tons/ft.

Tons mined on development per ft. of panel, including crosscuts:

[(2 x 300 ft.) + 80 ft.] 3.14 = 2135 tons
2135 ÷ 300 ft. = 7.12 tons/ft. of panel. Use 7

Production per shift on development:

Cutting rate after sumping in: 5 tons per minute

Average, including sumping: 3 tons per minute

Face time: 8 hours = 480 minutes

Time spent sumping and cutting: 100 minutes = 21% of total time

Coal cut per shift: 100 minutes @ 3 tpm = 300 tons

Distance of heading advance/shift: 300 ÷ 3.14 tons/ft. = 96 ft.

Distance of panel advance per shift: 300 ÷ 7 = 43 ft.

No. of 20 ft. lengths of pipe required to be added per machine shift: 96 ÷ 20 = 4.8; use 5

No. of monorail extensions: 96 ÷ 10 ft. = 9.6; use 10

No. of roof bolts required:

per 4 ft. advance: 2

per shift: (96 ÷ 4) x 2 = 48

No. of hydraulic props per machine shift: 96 ÷ 10 = 9.6; use 10

Crew size estimated: 7 face workers

Only 2 men are required to operate the monitor on retreat. For three monitors there will be three crews of 2 men.

For development, a support crew of 5 men will be used to assist the monitor crew in supporting the roof and extending the pipe, monorail, cable and ventilation tubing. Time needed for development work is only a small fraction of the total production time.

Two men are required for the cutting operation for 100 minutes per shift. The balance of their time would be spent assisting the other 5 crew members. The average coal cut per shift of 300 raw tons includes time lost in advancing the power center, changing the boom cutter for a monitor at the top of the room and changing the monitor for a boom cutter at the bottom of the room.

Productivity of face workers: 300 tons ÷ 7 men = 43 tons raw coal per man shift

Time to develop one typical panel, in shifts:

3300 ft. ÷ 43 ft/shift = 77 shifts

(The first room panel, being about 500 ft. shorter than the average, will require 65 shifts, or 0.72 months for development.)

Retreat Mining

As soon as a pair of rooms is completed, the boom cutter will be replaced with a monitor, converting the mining machine to a crawler mounted feeder-breaker monitor unit (see Figure 14-C). Retreat mining will then commence. (See Figure 16)

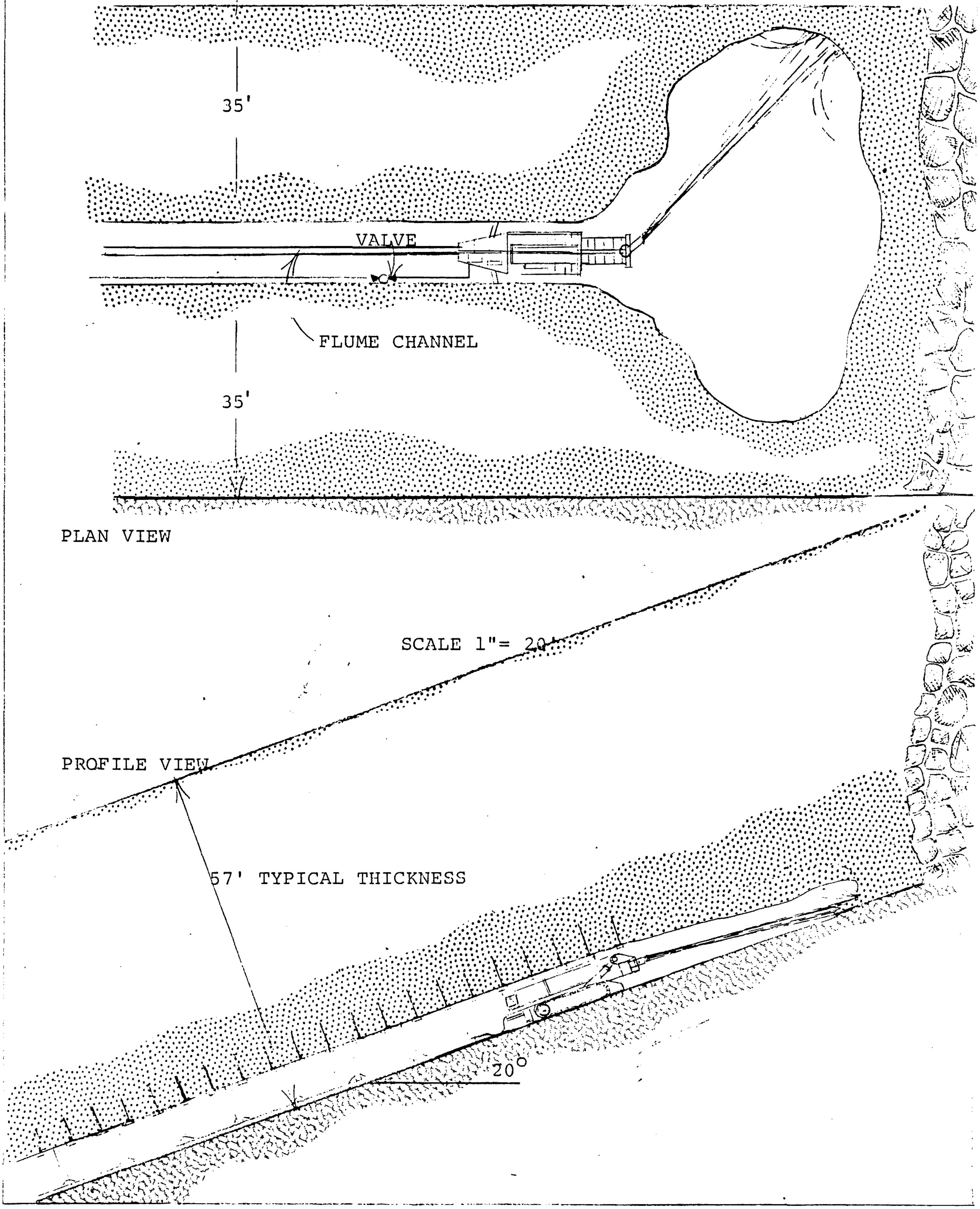
The monitor in each room will take approximately 40 ft. of coal on each side (about one-half of the distance being extracted at the Balmer Mine in Canada), thus accommodating the tougher coal at Kemmerer. Water pressure at the nozzle will be 12-1500 psi, similar to that at Balmer. In each seam, two monitors will be deployed, one in each of the panel rooms. While one place is in production, the other will be moving back.

Other factors being equal, the production rate of the monitor is a function of the energy supplied to the high pressure system. In effect, each monitor is on a closed water system, energized by two pumps, one 1800 hp slurry pump and one 2200 hp surface monitor pump.

The cutting sequence will be to first undercut the pillars on either side, taking out about 2 to 4 ft. of coal from the bottom horizon of the seam. This phase will be slow cutting, estimated at 1 ton per minute. The remaining coal will thereby be induced to cave and will be broken by gravity and abutment pressure. The broken coal from this caved mass will be readily flushed to the feeder-breaker on the steep 20 degree grade by the 2,000-3,000 gpm of water, at a rate estimated to average 6 tons per minute.

Near the outcrop where overburden pressure is lighter, roadheaders may be used to aid in undercutting. The degree to which this may be necessary will be determined by experience.

FIGURE 16
RETREAT MINING



To prevent excessive amounts of coal from flushing out of the area being pillared, the cross-section of the room is deliberately designed small, i.e., 7 ft. x 10 ft. At the caving lip, extra hydraulic props will be installed, forming an impenetrable grill to hold back excessive flushing from the gob. To control the roof in the room under abutment pressure, additional hydraulic props will be used on the density that experience dictates as necessary. Because of the soft characteristics of the roof rock, dilution is assumed to be 12%, which compares with Balmer's 7%. In this mine design more emphasis is being placed on recovery.

Given the relative weakness of the overburden rocks, abutment pressures and seam interaction, pressures are expected to be less severe than in most mines under similar depths of cover. Nevertheless, the mine should be prepared to place extra roof support over and above roof bolts to cope with such pressures should they occur. Such measures could include arches or additional hydraulic single props. The latter is much preferred and is lower in cost. Because the monorail haulage requires only narrow passageways, hydraulic props can be spaced on whatever density is required. Aluminum or fiberglass beams can effectively span the 5 ft. width needed for the monorail. Workmen have been provided in the manning estimates to take care of this contingency.

The monitor will be moved back when either the coal is "mined out" or dilution becomes excessive. A convenient interval for the moveback is assumed to be 40 ft. (two pipe lengths). The tasks involved in this phase of the operation include:

- Uncoupling the high pressure pipe.
- Tramming the feeder-breaker-monitor 40 ft. back to the new location.
- Looping or shortening the power cable.
- Shortening the ventilation tube.

- Extracting the roof bolts and moving back the hydraulic props to allow the top coal to cave in, as if retreating a single prop longwall face.
- Recoupling the pipe and flume to the feeder-breaker-monitor.

Monitor production is estimated as follows:

Average seam thickness: 57 ft.

Coal in place per average move-back of 40 ft.
(40 x 57 x 80 ÷ 25) - (40 x 3.5) = 7156 tons

In-place coal recovered at 60% = 4294 tons

Diluted coal recovered: x 1.12 = 4809 tons

Solid coal cut at estimated rate of 1 ton per minute to induce caving: assume an average undercut of 3 ft. near the bottom of the seam; equivalent to: 3 ÷ 57 x 4809 = 253 tons

Caved coal cut at an estimated rate of 6 tons per minute; 4809 - 253 = 4556 tons

Weighted average is:

		Minutes Required
253 tons of coal by undercutting at 1 tpm	=	253
4556 tons of "caved" coal at 6 tpm	=	759
4809	=	1,012

Aside from delay time, average cutting rate: 4.75 tons/min.

The crew size for monitoring is 2 workers per shift. Moving back is estimated to take 5 men 3 shifts. On the average, about 1 move-back will be required per day for the three seams.

Linear distance of panel extracted per day:

Average production of raw coal/day: 10,231 tons (Table 4)

Average production raw less development:

10,231 x 34 ÷ 35 = 9,939

Panel length recovered per day: 9939 ÷ 238 = 42 ft. This is equivalent to about one 40 ft. move-back per day.

Number of bolts and crossbars installed:

42 ft. of panel requires $\frac{(2 \times 300) + 80 \times 42}{300}$ = 95 ft. of development per day.

No. of bolts required per day = 95 ÷ 4 x 2 = 48

No. of crossbars required per day = 95 ÷ 4 = 24

Retreat-Development Ratio

The retreat-development ratio is the coal mined by the monitor on retreat divided by the coal mined on development. The cost of development with its slower production rate and cost of installing pipe, cable, flumes, roof support, etc. is a significant factor in the economic feasibility. The thick seams and the absence of gas make for a very favorable ratio, calculated as follows for the thickest, thinnest and average seam thickness.

	Thickest	Thinnest	Average
Thickness (ft.)	90	30	57
Tons/ft. of panel:			
in place:	576	192	365
recovered @ 60%	346	115	219
recovered with dilution	387	129	245
mined on development	7	7	7
mined on retreat	380	122	238
Retreat:development	54:1	17:1	34:1

Water-Coal Ratio

The water to coal ratio is estimated as follows:

(2250 gpm x 8.3453) ÷ 2,000 = 9.39 tpm

9.39 ÷ 4.75 = 1.98

Ratio = 1.98:1

Monitor Retreat Productivity

When the two mining machines in each panel are not developing, they will be equipped with monitors. With three seams under production, a total of six units will be deployed. While one of the monitors in each seam is producing, the other will be moved back and made ready for production.

The cave line in the 50 ft. thick No. 3 seam must be kept well ahead of the 30 ft. thick No. 2 seam immediately below. Therefore, production from No. 2 seam will have to be slowed down by scheduling fewer shifts so as not to overtake and undermine the thicker No. 3 seam. The area mined per unit of time in the No. 1 seam will be less than the seams above due to its greater thickness, i.e., 90 ft.

To slow down the rate of mining in No. 2 seam to the same area as No. 3, about 40% less coal tonnage will have to be scheduled. This will be accomplished by working the No. 2 seam at the rate of 2 shifts per day; and, if necessary, by reducing the crew size.

With only 2 monitors deployed in each seam, as compared with 3 at the Balmer Mine, operating time is estimated at 65% vs. Balmer's 74%. Monitor production is then estimated as follows:

		Minutes
Available time/shift, portal-to-portal, 9.25 hrs.		555
Less mantrip time		75*
Time available at the face		480
Production time: 65%		312
Tons per minute, estimated:	4.75	
Tons per monitor shift:	1482	
Average tons/min. of total scheduled shift time: 1482 ÷ 480 =	3.09	
Tons per man shift on retreat: 1482 ÷ 7 =	212	

* Trip time for Scharf monorail (see following page)

Mantrip time for Scharf monorail:

	Length				Time in minutes for	
	Maximum	Average	Gradient	f.p.m.	Maximum	Average
In	11,000	11,000	-12° max.	580	19.0	19.0
	3,300	1,700	+ 7°	400	8.3	4.3
	2,300	1,500	+20°	220	10.5	6.8
Total					37.8	30.1
Out	2,300	1,500	-20°	460	5.0	3.3
	3,300	1,700	- 7°	580	5.7	2.9
	11,000	11,000	+12°	350	31.4	31.4
Total					42.1	37.6
Total round trip					79.9	67.7

For the purpose of cost estimates, 1-1/4 hours of overtime will be provided for face workers required on continuous operations, e.g., monitors, where shift changes will be at the face. For the sake of simplicity, transportation of men and materials is assumed to be entirely by monorail. Using a single mode of transportation eliminates transferring men or material from one vehicle to another underground. However, it is recognized that monorail cannot compete in speed with diesel powered rubber tired vehicles for the 11,000 ft. rock tunnels. In a more detailed study, the economics should be considered of transporting workers by diesel powered rubber tired buses at speeds of 20 to 30 mph in the rock tunnels, and then transferring them to monorail at the bottom of the slopes. This would save 30 to 40 minutes of travel time and would probably justify the capital cost for paving the road in one of the tunnels and buying the diesel buses.

Table 4 estimates the clean coal produced per year from the three seams.

TABLE 4
PRODUCTION

Rooms on 80 ft. centers

Seam:	No. 1	No. 2*	No. 3	Total
Seam Thickness	90	30	50	
Tons per ft. of panel				
In-place	576	192	320	
In-place with 12% dilution	645	215	358	
Recovered (assuming 60%)	387	129	215	
Mined on development	7	7	7	
Mined by monitor	380	122	208	
Average panel length	3,170	3,320	3,420	
Tons of raw coal per average panel mined by:				
development:	22,190	23,240	23,940	69,370
monitor:	1,204,600	405,040	711,360	2,321,000
Total	1,226,790	428,280	735,300	2,390,370
No. of shifts required to mine panel:				
development: ÷ 300	74.0	77.5	79.8	
retreat by monitor: ÷ 1482	812.8	273.3	480.0	
Total	886.8	350.8	559.8	
No. of shifts moving, delays, etc.	9	9	9	
Total shifts per panel, rounded	896	360	569	
Average raw tons per shift	1,369	1,190	1,292	
No. of shifts worked per day	3	2	3	
Tons/day w/o restricting #2 Seam	4,107	2,380	3,876	
*Reduced tonnage for seam #2	4,107	2,248	3,876	10,231
Days per year	365	365	365	365
Total raw production/year (000)	1,499	820	1,415	3,734
Time in days to mine one panel	299	190	190	
Total clean coal production per year at 82% recovery (000)	1,230	672	1,160	3,062
Rounded to 3 million tpy to allow for 7 idle days (000)	1,205	658	1,137	3,000

* Working shifts and tonnage are reduced in No. 2 seam to maintain the area extracted per year at the same or less than in No. 3 seam. No. 2 seam, having thickness of 30 ft. compared to No. 3 Seam's 50, slightly shorter rooms can only produce at 58% of the tonnage rate per day or per year of No. 3 Seam.

Ventilation

The ventilation system includes the following features:

- Independent ventilation system for each seam.
- A combination of blowing and exhaust with variable speed fans automatically controlled to provide atmospheric pressure at the face.
- Since little or no methane is anticipated, ventilation quantities adequate for dilution of diesel fumes, assumed to be 150 cfm per diesel hp. The monorail locomotive has 90 hp, and thus requires 13,500 cfm. 20,000 cfm per seam will be provided.
- High ventilation efficiency assumed at 80%, providing 16,000 cfm at the last open crosscut.
- Cross-sectional area in the rooms of 70 sq. ft., giving an air velocity of 228 ft. per minute.
- All worked out panels sealed as quickly as possible.
- Only one place producing coal at a time in each seam, thus only one split of air in each seam required.
- Unused crosscuts on main levels sealed with sand backfill to insure against leakage and sponcom. A 30-inch diameter steel tube and doors will be provided in each third crosscut for man travel between intake and return aircourses.
- Rock tunnels and screening-dewatering station ventilated on a separate split of air with its own fan.
- Continuous operation to minimize the time that coal lost in the gob is exposed to partial ventilation.

The ventilation is shown schematically in Figure 17. The system in each seam will consist of one fan blowing, delivering 20,000 cfm at an approximate water gauge of 0.5 inches, and one fan on exhaust drawing the same amount of air out of the mine at the same water gauge pressure. The fans will be equipped with variable speed drives. This "push-pull" system will be controlled to automatically provide atmospheric pressure at the retreat face line, thus minimizing the flow of air into or out of the gob area. An inert oxygen deficient atmosphere will be maintained in the gob as close to retreat faces as possible. With the round-the-clock working schedule, coal lost in the gob will be exposed to an oxygen level that would support sponcom for as short a period as possible.

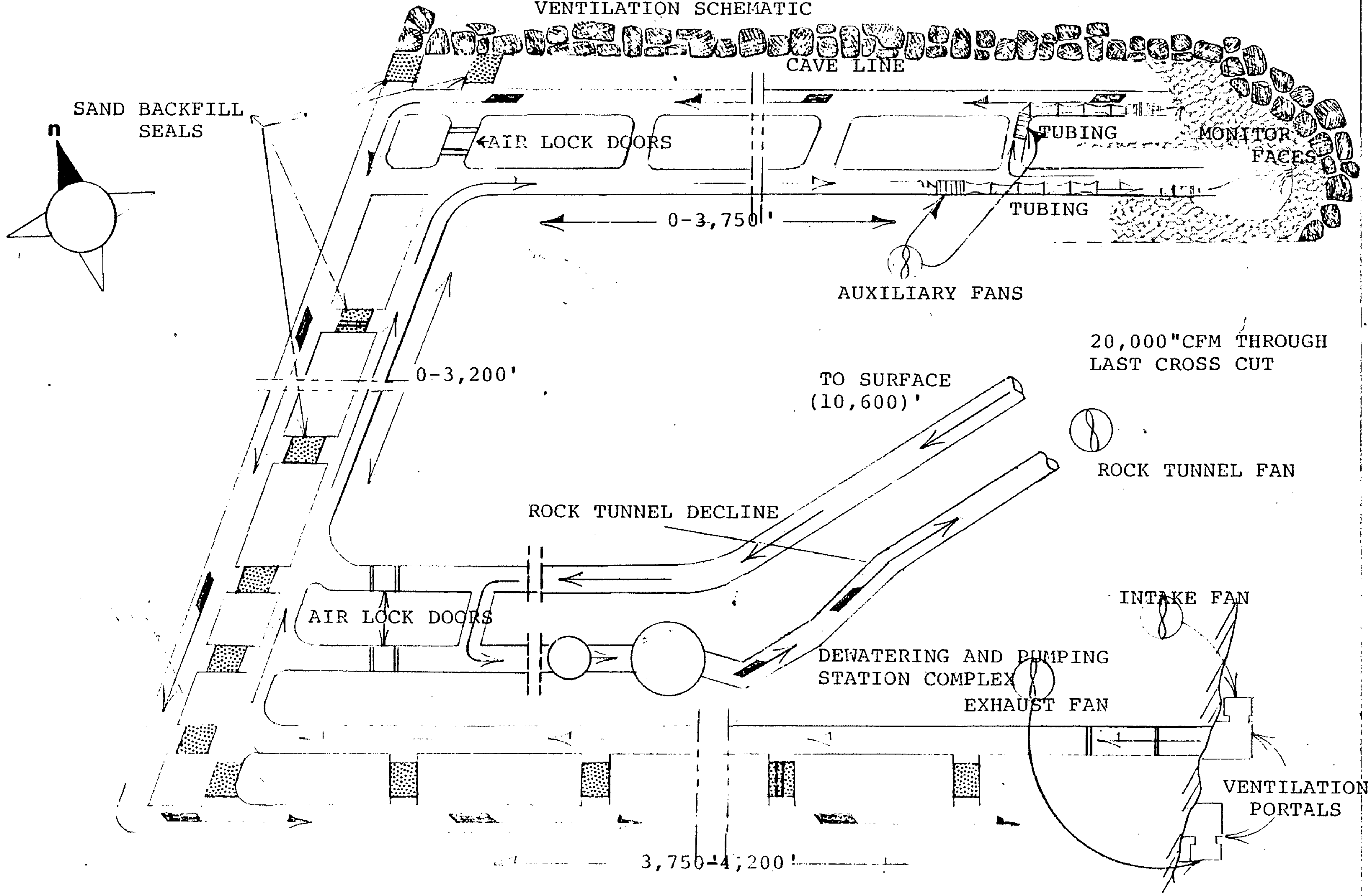
Auxiliary blower fans will be used to deliver air to the face both on development and retreat.

Upon completion of mining, each panel will be sealed with sand backfill.

This plan assumes that retreat mining will begin on the north end of the reserve block. The main aircourses will be driven to the surface on the south end of the block. The main levels in each seam will be driven to the south end and then up the pitch to the outcrop where the two fans for each seam will be installed. The fans, with motor and controls, as well as the steel airway connection to the portals, explosion door, air lock for man travel, etc. will all be designed and constructed as a prefabricated modular unit for ease of field installation. A 7 ft. diameter corrugated metal culvert will serve as the portal.

The ventilation air for driving the main levels to the south and the slope aircourses will be supplied by the rock tunnel ventilation fan. The coal produced in the main levels to the south will be 3200 x 9.3 = 30,000 tons. The slope aircourses to the fan site will produce 2,340 x 7 = 16,000 tons. Total tonnage produced in each seam, before being equipped with its own fans, would average 46,000 raw tons, equivalent to 38,000 clean tons. The production rate for all development is estimated to average 300 raw tons per shift, 900 tons per day, making the required time 51 days, equivalent to 1.7 months.

FIGURE 17
VENTILATION SCHEMATIC



WATER SYSTEMS

The water systems for the Kemmerer hydraulic mine include:

- Pumping of natural mine water inflow.
- High pressure water for extraction of top coal and pillar coal by jet.
- Low pressure water for flushing and fluming development coal.
- Pumping dilute slurry of minus 14 mesh coal and water to the surface.

Figure 18 is a schematic of the hydraulic system.

Natural Mine Water Inflow

In addition to the water pumped to the producing panels, ground water inflow will join the coal and water in the flumes and will collect in the sump at the dewatering station. From there, it will be pumped out of the mine by the slurry pumps.

The mine does not lie under any stream beds or alluvial formations. The majority of the reserves are in the saturated zone below 7,000 ft., which is estimated to be the water elevation. The old underground mines, when working below the water table, made up to 220,000 gallons per day (153 gpm). Since there will be hydraulic communication between the broken subsided overburden and the coal, these fractures, rather than porosities of the formations themselves, will contribute to the ground water inflow.

Accurately predicting the mine water inflow is beyond the scope of this study and should be evaluated in more detail by geohydrologic studies before mining. However, for this report, an inflow of 500 gpm will be assumed. This volume is representative of other western underground coal mines with similar depths, topography, and arid climates.

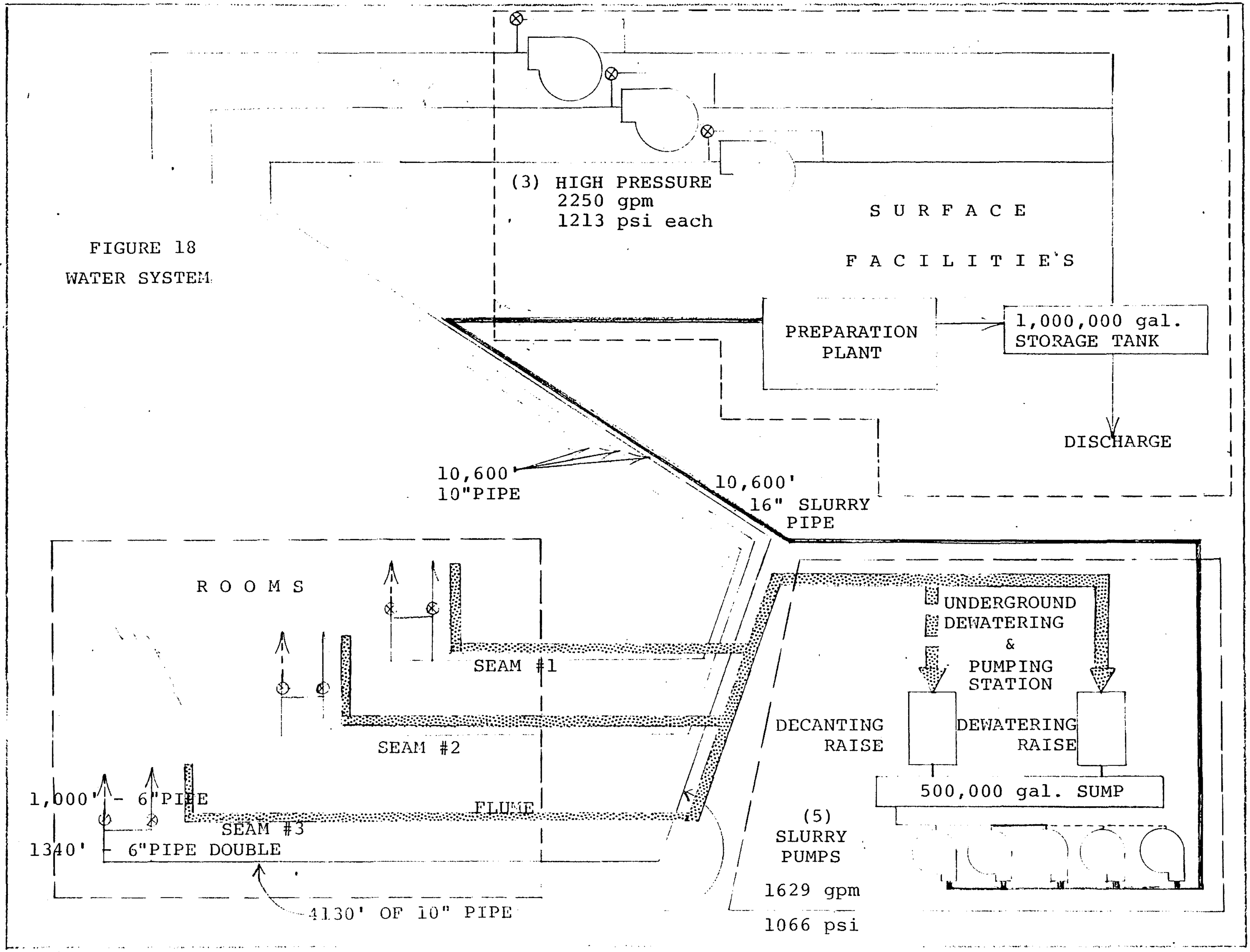


FIGURE 18
WATER SYSTEM

Slurry Pumping System

The slurry pumps will serve three purposes: mine dewatering, transportation of fine coal to the surface, and elevation of the circulating water to supply all of the hydraulic head needed for development fluming and part of the head required for mining by monitor. The total water to be pumped from the mine, along with the minus 14 mesh coal, is 4,888 gpm average over a full shift and 8,180 gpm maximum (see detail in Table below).

TABLE 5

DILUTE SLURRY PUMPING

	Avg. while oper. gpm	Oper. time as % of total time	Gals. per avg. shift (000)	Max. while oper. gpm	Max. oper. time as % of total time	Gal./ shift under max. conditions (000)	gpm Avg. over full shift	gpm Max.
Mine water inflow	500	100%	240	500	100%	240	500	500
3 monitors	6,750	65%	2,106	9,600	80%	3,686	4,388	7,680
Total	7,250		2,346	10,100		3,926	4,888	8,180

To estimate the volume of 14 mesh coal:

Average monitor production while running	4.75 tons/min.
Peak production per monitor: use 2 x 4.75	9.50
Monitors scheduled for operation	3
Estimated % of time monitor operates	65%
% of time all 3 monitors are running simultaneously: 0.65 x 0.65 x 0.65 =	27%
Potential peak production of all 3 monitors allowing no diversification factor: 3 x 9.50 =	28.5
Diversification factor, assume	80%
Total peak production: .80 x 28.5 =	22.8
	Use 23 tons/min.

To examine the reasonableness of the assumption of 80% diversification factor, assume that 20% of the time any single monitor would be running at 7.7 tons/minute or more, up to a peak of 9.5.

The probability of all three monitors running at this rate is: .27 x .20 x .20 x .20 = 0.0022 (= 0.22%), equivalent to 19 operating hours per year. Production of No. 2 seam, being thinner, must be paced in any event (see multi-seam mining and production); therefore, for this probable 19 hours per year, the production from this seam would be slowed down so as to stay within the 23 tons per minute of peak capacity provided in the underground dewatering station.

Peak production is limited in part by the water to coal ratio. At the high rates of 7.7 to 9.5 tons per minute, the ratio of water to coal would be low, in the range of 1.2 to 1.7.

The dilute minus 14 mesh slurry will be collected in an underground 500,000 gallon sump which will level out the peaks and valleys in dilute slurry being pumped out of the mine.

Based on several screen analyses of 1-5/8" x 0, it is estimated rom coal from the feeder-breakers will approximate:

6" x 1/2"	60%
1/2" x 14M	27%
14M x 0	13%

Volume per ton of raw coal is estimated at 23 $ft.^3$/ton.

With this data, minus 14 mesh coal inflow to be slurried is estimated:

	Avg. w/all monitors running	Avg. for 480 min. shift	Peak
Total production (tph)	855	556	1,380
% fraction minus 14 mesh	13%	13%	13%
14 mesh x 0 (tph)	111	72	179
Volume/weight factor, rom, ft^3/ton	23	23	23
Volume minus 14 mesh (cu. ft./hr.)	2,556	1,662	4,126
Volume minus 14 mesh (cu. ft./min.)	42.6	27.7	68.8
Volume minus 14 mesh (gpm): x 7.48	319	207	515
Total volume water (gpm)	7,250	4,888	8,180
Total volume of slurry (gpm)	7,569	5,095	8,695
Tph of water	1,812	1,222	2,045
Total tph of slurry	1,923	1,294	2,224
% solids in -14M slurry	5.8	5.6	8.0

	Average with all pumps running	Average over 480 minute shift	Peak
To determine specific gravity of slurry:			
Tons of water/minute	30.21	20.37	34.08
Tons of coal per minute	1.85	1.20	2.98
Tons of slurry per minute	32.06	21.57	37.06
Lbs. of slurry per minute	64,120	43,140	74,120
Volume of slurry per minute, $ft.^3$, (gpm/7.48)	1,012	681	1,162
Weight of slurry, $lbs/ft.^3$	63.36	63.35	63.79
Specific gravity of slurry, ÷ 62.4	1.015	1.015	1.022
Psi per ft. of head of slurry, x 0.4335	0.44	0.44	0.44

To pump this volume of slurry, four Wilson-Snyder 2000-SP slurry pumps, driven by automatically controlled 2,000 hp variable speed motors, have been chosen. Their performance capability and operating data are shown in Figure 19 and dimensions in Figure 20. Most of the operating time only two or three of the pumps will be working; the third and fourth will serve as standbys which will be operated only during periods of peak production or failure of one of the other pumps. The variable speed drives permit adjusting the volume pumped to match the variations in volume from the mine working faces. The fourth pump permits a rotating schedule for service and repair work.

The slurry will be transported through a 16-inch diameter pipe line which will travel up the rock tunnel decline parallel with the conveyor. The operating conditions and pump performance are listed as follows:

	Average over 480 min. shift	Peak (@ minimum head and 80% operating time)
Gpm of slurry	4,888	8,180
No. of pumps	3	4
Gpm required per pump	1,629	2,045
Piston size	11-3/4"	11-3/4"
Gpm per revolution	31.84	31.84
Rpm required (99% vol. eff.)	51	65
Static head	2,243'	2,243'
Friction head		
10,600'/100 x 1.36 x 1.25*	180	NAp
10,600'/100 x 3.69 x 1.25*	NAp	489
Total pressure required		
head	2,423	2,732
psi (head x .44)	1,066	1,202
Maximum pressure capability of pump (psi) w/2,000 hp motor	1,458	1,458
Hp applied per pump (88% mech. eff.)	2,000	2,000
Hp drawn per pump at peak	NAp	2,000
Average hp drawn per pump: 2,000 ÷ 65 x 51 x 1066 ÷ 1458	1,147	NAp
Total hp drawn	3,441	8,000

*25% factor of safety applied to provide for age of pipe, friction in pipe fittings and valves, and increased friction of dilute slurry over water.

FIGURE 19
SLURRY PUMP DATA

WILSON SNYDER 2000-SP

Type of Pump - Duplex Double Acting Piston
Stroke Length - 18 Inches
Piston Load Rating - 148923 lbs.
Piston Rod Diameter - 4.000 Inches

Fluid End Size	Piston Size In.	Maximum Pressure PSI	Gal. Per Rev.	Displacement, GPM @ RPM of - (99 Pct. Vol. Eff.) 15	25	35	45	55	65
No. 2	9 1/2	2305	20.13	299	498	698	897	1096	1296
	9 3/4	2178	21.31	316	527	738	949	1160	1371
	10	2061	22.52	334	557	780	1003	1226	1449
	10 1/4	1954	23.76	353	588	823	1059	1294	1529
	10 1/2	1854	25.03	372	620	867	1115	1363	1611
No. 3	10 3/4	1763	26.33	391	652	912	1173	1434	1694
	11	1678	27.66	411	685	958	1232	1506	1780
	11 1/4	1599	29.02	431	718	1006	1293	1580	1868
	11 1/2	1526	30.42	452	753	1054	1355	1656	1957
	11 3/4	1458	31.84	473	788	1103	1418	1734	2049
	12	1394	33.29	494	824	1154	1483	1813	2142
No. 4	12 1/4	1335	34.78	516	861	1205	1549	1894	2238
	12 1/2	1279	36.29	539	898	1258	1617	1976	2335
	12 3/4	1227	37.84	562	936	1311	1686	2060	2435
	13	1178	39.41	585	975	1366	1756	2146	2536
Brake Horsepower (88% Mechanical Efficiency)				462	769	1077	1385	1692	2000

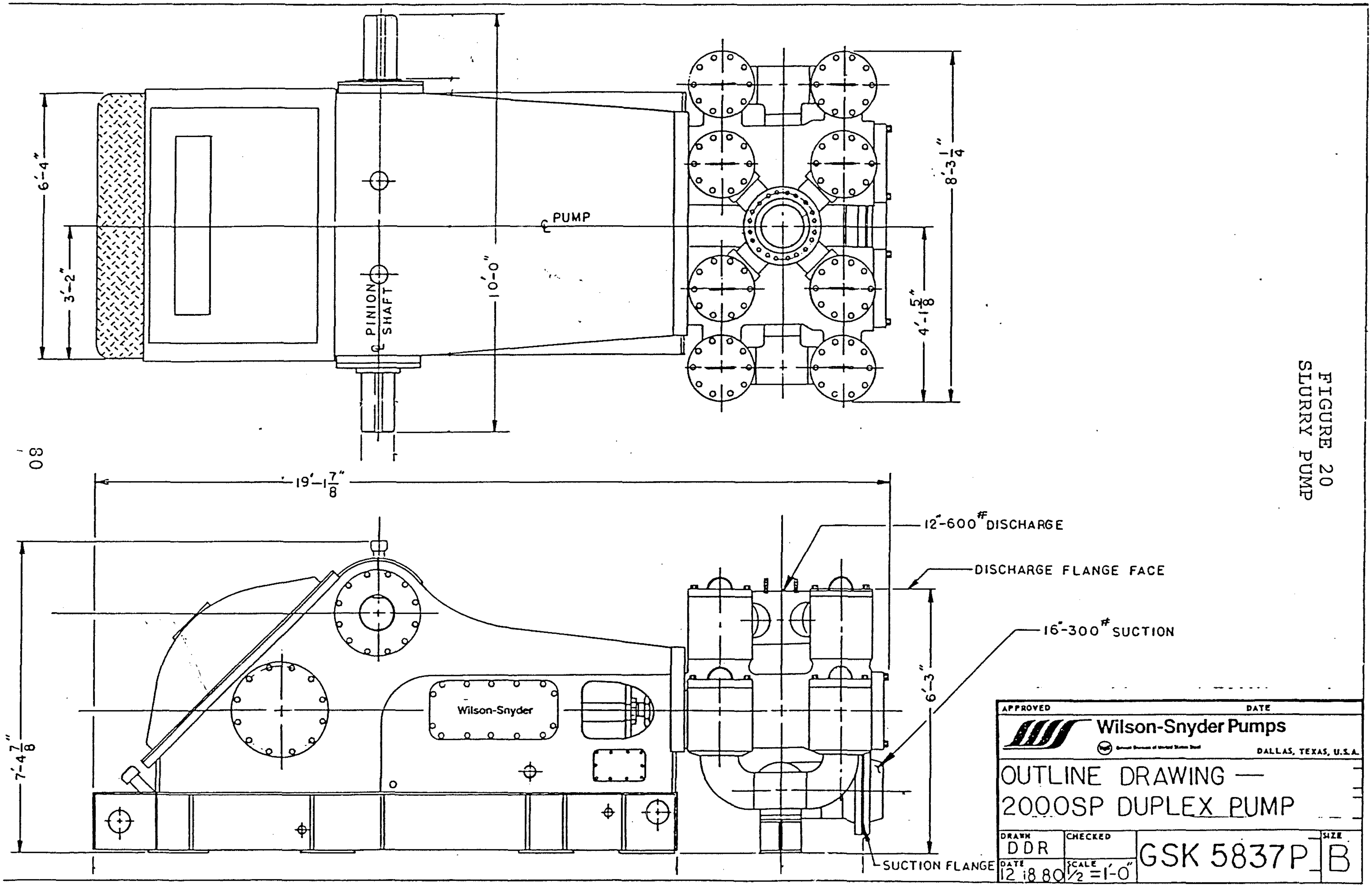


FIGURE 20
SLURRY PUMP

High Pressure System

This system is designed to supply to the retreating monitor 2000-3200 gpm of water at a minimum pressure of 1200 psi. Each of the three seams will be supplied through separate pipe systems, consisting of 10-inch Schedule 80 pipe, from the surface down the rock tunnel declines, up the inclines, and up the main levels. From the main level, a 6-inch Schedule 80 pipe will be laid up each of the two rooms in a pair. The two pipes will be cross-connected at every third crosscut. This will provide two pipes to carry the high pressure water to within 1,000 ft. of either face. For the remaining distance to either of the two faces, the water will alternately flow in a single line to one of the two monitors deployed. Six-inch pipe is used in the rooms for easier handling on development and monitor pullback, whereas the 10-inch line in the mains and rock tunnels is only installed and recovered once in the life of the mine.

Part of the energy supplied to the monitor is derived from the 2200 ft. lift of the underground slurry pumping system which moves the water and minus 14 mesh coal from the dewatering station to the surface. On the surface the clarified water is given further energy by three multistage centrifugal pumps, one for each seam.

The pumps selected for this application are 4-stage units, 2500 hp each, as manufactured by Ingersoll-Rand. A total of four pumps would be installed to provide for three operating units and one spare. The pump curves are shown in Figure 21 and pipe friction data in Figure 22. Following the figures, the centrifugal pump performance data, static and friction heads and resulting pressure at the monitor for various face locations are shown.

FIGURE 21
HIGH PRESSURE PUMP

INGERSOLL-RAND or equivalent Curve No.6X13DA1-3

Impeller: 335mm.

No. of stages: 4

Motor: 2500 hp.

No. of pumps: 3

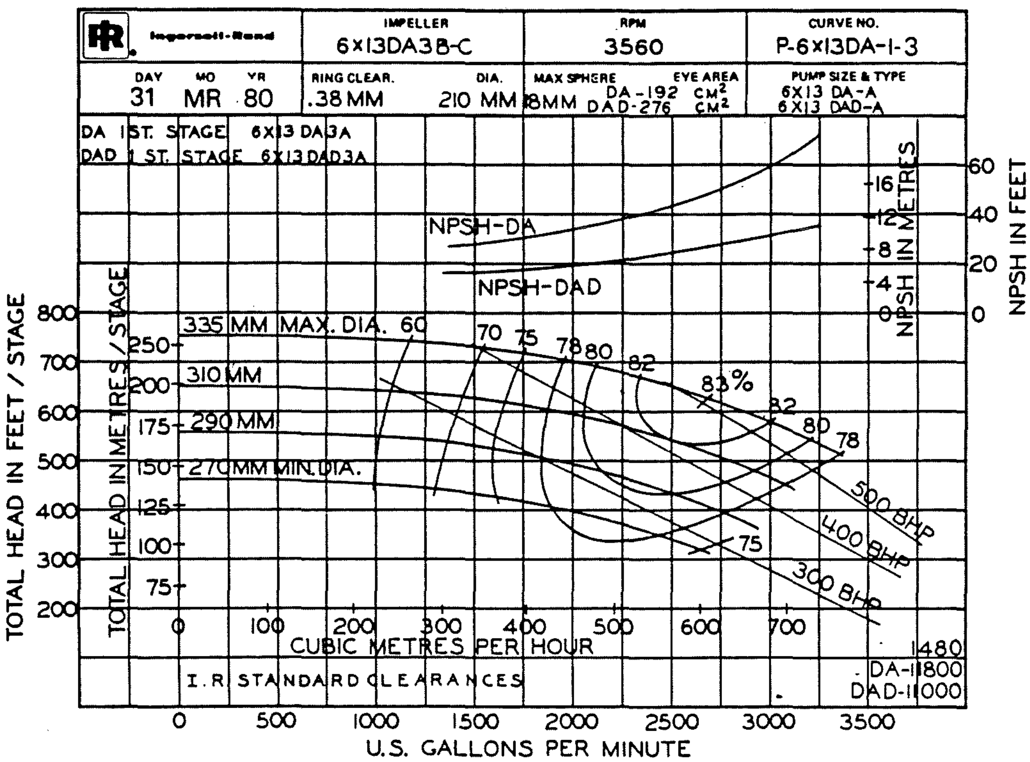


FIGURE 22

PIPE FRICTION

Note: No allowance has been made for age, difference in diameter, or any abnormal condition of interior surface. Any factor of safety must be estimated from the local conditions and the requirements of each particular installation. It is recommended that for most commercial design purposes a safety factor of 15 to 20% be added to the values in the tables

6"

Flow US gal per min	Extra strong steel sch 80		
	5.761" inside dia		
	Velocity ft per sec	Velocity head ft	Head loss ft per 100 ft
50	.62	.01	.032
60	.74	.01	.044
70	.86	.01	.058
80	.98	.01	.074
90	1.11	.02	.091
100	1.23	.02	.110
120	1.48	.03	.154
140	1.72	.05	.203
160	1.97	.06	.260
180	2.22	.08	.323
200	2.46	.09	.392
220	2.71	.11	.451
240	2.96	.14	.530
260	3.20	.16	.616
280	3.45	.19	.708
300	3.69	.21	.807
320	3.94	.24	.911
340	4.19	.27	1.02
360	4.43	.31	1.14
380	4.68	.34	1.26
400	4.93	.38	1.39
450	5.54	.48	1.74
500	6.16	.59	2.13
550	6.77	.71	2.55
600	7.39	.85	3.02
650	8.00	.99	3.52
700	8.63	1.16	4.06
750	9.24	1.33	4.64
800	9.85	1.51	5.25
850	10.5	1.7	5.90
900	11.1	1.9	6.60
950	11.7	2.1	7.33
1000	12.3	2.4	8.09
1100	13.5	2.8	9.74
1200	14.8	3.4	11.5
1300	16.0	4.0	13.5
1400	17.2	4.6	15.6
1500	18.5	5.3	17.8
1600	19.7	6.0	20.3
1700	20.9	6.8	22.8
1800	22.2	7.7	25.5
1900	23.4	8.4	28.4
2000	24.6	9.4	31.4
2200	27.1	11.4	37.9
2400	29.6	13.6	44.9

10"

Flow US gal per min	Schedule 80 steel		
	9.562" inside dia		
	Velocity ft per sec	Velocity head ft	Head loss ft per 100 ft
180	.804	.010	.027
200	.894	.012	.033
220	.983	.015	.039
240	1.07	.018	.046
260	1.16	.021	.053
280	1.25	.024	.061
300	1.34	.028	.069
350	1.56	.038	.092
400	1.79	.050	.117
450	2.01	.063	.145
500	2.34	.077	.177
550	2.46	.094	.211
600	2.68	.112	.239
650	2.90	.131	.277
700	3.13	.152	.319
800	3.57	.198	.410
900	4.02	.251	.512
1000	4.47	.310	.625
1100	4.92	.375	.749
1200	5.36	.446	.884
1300	5.81	.524	1.03
1400	6.26	.607	1.19
1500	6.70	.697	1.35
1600	7.15	.793	1.53
1700	7.60	.895	1.72
1800	8.04	1.00	1.92
1900	8.49	1.12	2.13
2000	8.94	1.24	2.36
2200	9.83	1.50	2.83
2400	10.72	1.79	3.35
2600	11.62	2.09	3.92
2800	12.51	2.43	4.52
3000	13.40	2.79	5.17
3200	14.30	3.17	5.87
3400	15.19	3.58	6.60
3600	16.08	4.02	7.38
3800	16.98	4.47	8.21
4000	17.87	4.96	9.07
4500	20.11	6.27	11.4
5000	22.34	7.75	14.1
5500	24.57	9.37	17.0
6000	26.81	11.15	20.1
6500	29.04	13.09	23.6
7000	31.28	15.18	27.3
7500	33.51	17.43	31.2

16"

Flow US gal per min	New steel schedule 40		
	15.000" inside dia		
	Velocity ft per sec	Velocity head ft	Head loss ft per 100 ft
500	.908	.013	.020
600	1.09	.018	.027
700	1.27	.025	.036
800	1.45	.033	.046
900	1.63	.041	.058
1000	1.82	.051	.070
1200	2.18	.074	.098
1400	2.54	.100	.130
1600	2.91	.131	.161
1800	3.27	.166	.201
2000	3.63	.205	.245
2500	4.54	.320	.374
3000	5.45	.460	.530
3500	6.35	.627	.712
4000	7.26	.819	.920
4500	8.17	1.04	1.15
5000	9.08	1.28	1.42
6000	10.89	1.84	2.01
7000	12.71	2.51	2.72
8000	14.52	3.27	3.53
9000	16.34	4.14	4.44
10,000	18.16	5.12	5.45
11,000	19.97	6.19	6.58
12,000	21.79	7.37	7.80
13,000	23.60	8.65	9.13
14,000	25.42	10.03	10.6
15,000	27.23	11.51	12.1
16,000	29.05	13.10	13.7
17,000	30.86	14.79	15.5
18,000	32.68	16.58	17.3
20,000	36.31	20.46	21.3
22,000	38.94	24.76	25.8
24,000	45.57	29.47	30.6
26,000	47.20	34.58	35.9
28,000	50.84	40.11	41.5
30,000	54.47	46.04	47.6
32,000	58.10	52.39	54.1
34,000	61.73	59.14	61.0
36,000	65.36	66.30	68.4
38,000	68.99	73.88	76.1

	Hydraulic Conditions		
	Maximum Distance	Average Distance	Minimum Distance
Gpm	2,000	2,250	3,200
Impeller diameter	335mm	335mm	335mm
No. of stages	4	4	4
Hp per stage	450	475	550
Total horsepower	1,800	1,900	2,200
Head per stage (ft.)	700	685	550
Total head supplied by pump (ft.)	2,800	2,740	2,200
Static head (ft.) from pump to monitor	890	1,290	2,190
Total head	3,690	4,030	4,390
Pipe friction			
Distance (ft.)			
10-inch single	14,730	13,070	11,410
6-inch double	1,340	1,300	0
6-inch single	1,000	600	100
Friction factor (in ft./100 ft.) (x 1.2) (See Figure 22)			
10-inch single	2.83	3.55	7.04
6-inch double	9.71	12.22	0
6-inch single	37.68	47.58	95.00
Friction head (ft.)			
10-inch single	416	464	804
6-inch double	130	159	--
6-inch single	377	285	95
Total	923	908	899
Pressure at monitor	2,767	3,122	3,491
Nozzle, psi (1 ft. of head = 0.4335 psi)	1,200	1,353	1,513

Low Pressure System

Water for flushing of development coal will be supplied through the high pressure pipe system. Due to the high elevation of the surface plant relative to the working faces, more than enough water will flow naturally to the roadheaders to flush and flume the coal from the face to the dewatering station. At the end of the line on the roadheader, the amount of water used will be throttled to the amount required for reliable fluming. Following are calculations for the "worst case" where static head is minimum and pipe friction is maximum:

Amount of water desired for flushing coal:		1500 gpm
Static head available from surface		896 ft.
Pipe friction	feet	
10-inch line		
14,730'/100 x 1.35 (1.2) =	239	
6-inch double line		
1340'/100 x 4.64 (1.2) =	75	
6-inch single line		
1,000'/100 x 17.8 (1.2) =	214	
4-inch hose		
50'/100' x 143 (1.2) =	86	
Total friction head	614	
Excess pressure in elevation head available for increasing fluming water supply		282

Coal Dewatering and Preparation

The coarse coal, plus 14 mesh, will be screened off in an underground dewatering and pumping station. The dewatered product, making up an estimated 87% of the raw coal, will be transported via a 48-inch belt conveyor up the rock tunnel 10,600 ft. to a raw coal stockpile ahead of the preparation plant. The 14 mesh x 0 coal and water will be stored in a 500,000 gallon underground sump, from which it will be pumped out of the mine by plunger pumps in a 16" diameter slurry pipeline parallel to the conveyor.

On the surface, the minus 14 mesh coal will be cleaned by "water only cyclones" and dewatered. The water will be clarified and returned to the working faces by centrifugal pumps.

The coarse raw coal, 6" x 14M, will be stored on the surface in an open pile. From there it will be washed in a Baum jig plant at the rate of 800 tph. The clean coal will be delivered to a clean coal stockpile and the refuse to a disposal area.

As to production rates, clean coal per year is estimated to average 3,000,000 tons from 3,659,000 tons of raw, equivalent to 10,219 tons of raw coal per average mine work day (assuming 358 days per year), or 425 raw tons per scheduled hour.

The average cutting rate per monitor is 4.75 tons per minute, or 855 tons per hour for the three monitors. However, when extracting caved coal, the monitors are expected to have a higher rate of 6 tons per minute. Any one monitor could produce as much as double this rate at times, 12 tons per minute, or at a water to coal ratio of 1.1:1. However, the diversification effect of three monitors will tend to level out the surges. Taking this into account, in the section on slurry pumping, the peak to the underground screening station has been estimated at 23 tons per minute. The maximum water volume to the three monitors combined is 9600 gpm, or 40 tons per minute. At peak conditions then, the water to coal ratio would be 1.74:1. Summarizing, the production of raw coal and water inflow is estimated as:

	Average over full year	Average with all 3 faces oper.	Peak with all 3 faces oper. at min. head
Coal (tph)	425	855	1,380
Water (gpm)			
Monitor	NAp	6,750	9,600
Mine water	NAp	500	500
		7,250	10,100

The dewatering flow sheet is shown on Figure 23. The advantage of this circuit is its relative insensitivity to surges in water and/or coal.

When the coal slurry reaches the dewatering station, the slurry will pass above the emergency decanting bunker. The flume will be equipped with a gate to automatically open during a power outage so that all the coal slurry in the flumes will flow to the decanting bunker.

The slurry entering the dewatering raise will be split into two streams and pass over two 8 ft. x 20 ft. single deck 1/2" scalping screens. The 6 x 1/2 inch coal, which is estimated to be about 60% of the raw coal, will be screened off for conveyor transport.

The 1/2" x 0 coal and water will be passed over 2 - 7 ft. wide, 80" radius, 60 degrees, 2 mm aperture sieve bends to remove most of the water. Each sieve bend will have a dewatering capacity of 650 gpm per ft. of width, or 4550 gpm.

Two vibrating screens, 8 ft. x 12 ft. with 14M decks, will follow the sieve bends, each handling a peak tonnage of 200 tph, and an average of about 80 tph. Two vibrating centrifuges will follow the vibrating screens, each capable of handling a peak of 190 tph and an average of 115 tph when all three monitors are producing. The centrifugally dried product, 1/2" x 14 mesh, will join the plus 1/2" on the belt.

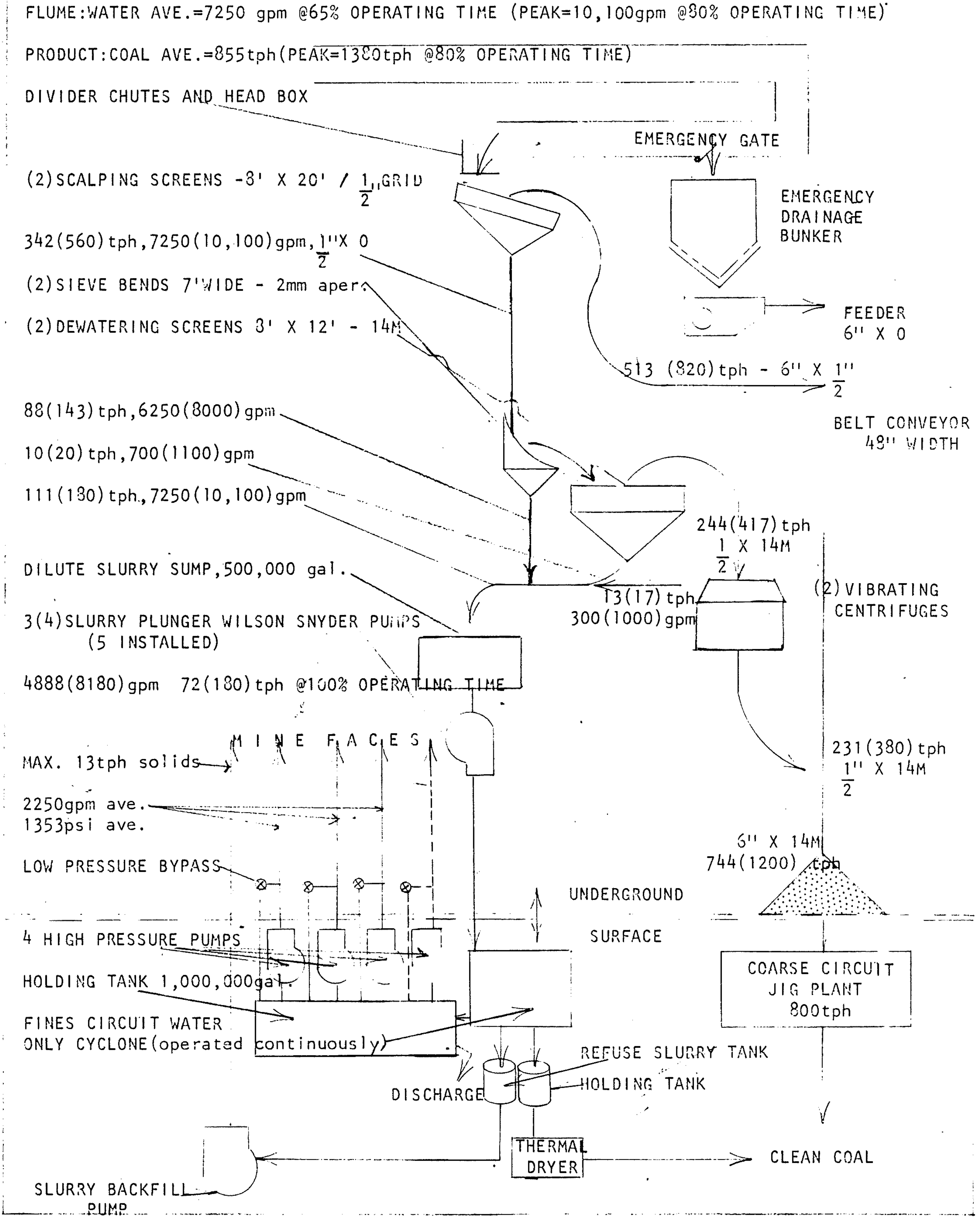
The minus 14 mesh and water will flow to the 500,000 gallon sump and be pumped to the preparation plant as a dilute slurry, containing about 3% solids during average and peak loads.

The 14 mesh x 0 coal will be then cleaned and dewatered in water-only cyclones. The clarified and treated water will be partly returned to the mine by the high pressure centrifugal pumps. Any surplus resulting from excess natural mine water inflow will be disposed of in an environmentally acceptable manner. In this arid region, this is not anticipated to be a significant problem.

In case of a power outage, the coal which enters the emergency decanting bunker will be drained of water slowly by percolation of the water through the coal and out to the slurry sump. When dewatered sufficiently for belt transport, and with the restoration of power, the gate in the bottom of the bunker will be opened and the coal will be fed onto the conveyor belt to the preparation plant.

FIGURE 23

DEWATERING STATION



MANPOWER ESTIMATE

By keeping the pillar line continually moving, the risk of spon com is reduced and safety is improved. Because of the high susceptibility to spon com at this property, it is assumed for the purpose of this study that a seven day per week working schedule can be negotiated with the United Mine Workers of America.

A total of 358 working days a year is assumed, allowing 7 days for the most important holidays. To permit a continuous operation, a "6 and 3" work schedule is planned. All workers required for production, dewatering, water clarification, pumping, material delivery and any other directly associated functions are scheduled to work six consecutive days, followed by three days off. This schedule requires 50% more workers on the payroll for these functions than are on the job each day.

An additional provision is made for 10% extra workers to permit annual vacations, holidays, and days off for all employees.

All underground workers in the face and on the screening and pumping station, are scheduled for 9-1/4 hour shifts portal-to-portal so that, aside from delay time and development time, the monitors can produce 24 hours per day. Other underground workers, e.g. on material delivery, ventilation, etc., are scheduled to work 8 hours per shift. Surface workers who are associated with cleaning and dewatering the minus 14 mesh coal pumped out of the mine, water clarification, and operating and maintaining the high pressure pumps, would be scheduled 8 hours per shift. Other surface hourly workers would be scheduled for the standard 7-1/4 hour shift. Salaried workers would be scheduled for 8 hour shifts.

Workers on jobs not required around the clock will work 239 days per year.

TABLE 6

HYDRAULIC MINE WORK FORCE

	Worker shifts per year 7-1/4 hours	8 hours	9-1/4 hours	Total
Underground hourly (Table 6-A)				
Panel workers	--	10,740	8,950	19,690
Dewatering/pumping and belt	--	--	6,444	6,444
General	--	11,702	--	11,702
Total underground	--	22,442	15,394	37,836
Surface hourly (Table 6-A)	7,887	5,370	--	13,257
Total hourly	7,887	27,812	15,394	51,093
Salaried employees (Table 6-B)				
Underground				8,474
Surface				3,703
Management and staff				7,288
Total salaried				19,465
Total shifts				70,558

Recapitulation:	Under-ground	Surface	Total
Hourly employees			
"5-day" work week	22	33	55
"6-3" schedule	91	15	106
Sub-Total per work day	113	48	161
Extra for covering "6-3" sched.	45	8	53
Total	158	56	214
Provision for vacation, etc.	16	5	21
Total hourly on payroll	174	61	235
Salaried employees	36	47	83
Total	210	108	318

Productivity = 3,000,000 tpy ÷ 70,558 man shifts/year = 42 tpm

TABLE 6-A

HOURLY WORK FORCE

	Workers per shift	Shifts per day	Workers per day	Work days per year	Hours per shift	Worker Shifts Per Year: 7-1/4 hours	8 hours	9-1/4 hours	Total
Panel workers									
Monitor/roadheader									
Seam #1	2	3	6	358	9-1/4	NAp	NAp	2,148	
Seam #2	2	2	4	358	9-1/4	NAp	NAp	1,432	
Seam #3	2	3	6	358	9-1/4	NAp	NAp	2,148	
Support crew									
Move-backs, etc.	10	3	30	358	8	NAp	10,740	NAp	
Mech./electricians	3	3	9	358	9-1/4	NAp	NAp	3,222	
							10,740	8,950	19,690
Dewatering/pumping/slope belt									
Operator	1	3	3	358	9-1/4	NAp	NAp	1,074	
Clean-up, lubrication	1	3	3	358	9-1/4	NAp	NAp	1,074	
Mech./electrician	3	3	9	358	9-1/4	NAp	NAp	3,222	
Belt clean-up, etc.	1	3	3	358	9-1/4	NAp	NAp	1,074	
								6,444	6,444
General underground									
Ventilation, backfilling, etc.	8	1	8	239	8	NAp	1,912	NAp	
Flume lines/pipe	6	1	6	239	8	NAp	1,434	NAp	
Material/supply	5	3	15	358	8	NAp	5,370	NAp	
Monorail maintenance	4	1	4	239	8	NAp	956	NAp	
Maint. roof, ribs	4	1	4	239	8	NAp	956	NAp	
Fireboss	1	3	3	358	8	NAp	1,074	NAp	
							11,702		11,702
Preparation plant									
Coarse circuit	6	3	18	239	7-1/4	4,302	NAp	NAp	
Fines circuit, pumps	2	3	6	358	8	NAp	2,148	NAp	
Backfill system	1	1	1	239	7-1/4	239	NAp	NAp	
Prep. plant maint.	2	3	6	358	8	NAp	2,148	NAp	
Warehouse	1	3	3	358	8	NAp	1,074	NAp	
Shop mech., welders, etc.	5	2	10	239	7-1/4	2,390	NAp	NAp	
General labor	4	1	4	239	7-1/4	956	NAp	NAp	
						7,887	5,370		13,257
Total hourly									51,093

At 239 shifts per employee, equates to 214 employees.

TABLE 6-B

SALARIED WORK FORCE

	Workers per shift	Shifts per day	Workers per day	Work days per year	Total worker shifts per year
Underground					
Monitors or development	3	3	9	358	3,222
Dewatering and pumping	1	3	3	358	1,074
Material delivery	1	3	3	358	1,074
Shift foremen	1	3	3	358	1,074
Maintenance foremen	1	3	3	358	1,074
General foreman	1	1	1	239	239
Assistant general foreman	1	1	1	239	239
Maintenance general foreman	1	1	1	239	239
Other underground foremen	1	1	1	239	239
(Total underground employees required: 36)					8,474
Surface					
Preparation plant general foreman	1	1	1	239	239
Quality control	2	1	2	239	478
Coarse preparation shift foreman	1	3	3	239	717
Fines circuit and pumping	1	3	3	358	1,074
Shop	1	1	1	239	239
Warehouse	1	1	1	239	239
General	3	1	3	239	717
(Total surface employees required: 16)					3,703
Management and staff					
General manager	1	1	1	239	239
Mine superintendent	1	1	1	239	239
Assistant mine superintendent	1	1	1	239	239
Preparation plant superintendent	1	1	1	239	239
Maintenance superintendent	1	1	1	239	239
Asst. maintenance superintendent	1	1	1	239	239
Accounting and payroll	4	1	4	239	956
Purchasing	2	1	2	239	478
Safety, training, sponcom	4	1	4	239	956
Personnel	2	1	2	239	478
Engineering	4	1	4	239	956
Reception and secretarial	4	1	4	239	956
Security	1	3	3	358	1,074
(Total management and staff required: 31)					7,288
Total salaried employees: 83					19,465

DEVELOPMENT AND PRODUCTION SCHEDULE

Pre-Development

Pre-development for a hydraulic mine at Pittsburg & Midway's Kemmerer Mine requires:

- Exploration drilling to determine:
  - Reserves
  - Mineable thicknesses
  - Structure
  - Mining conditions
- Environmental permits
  - Submission and approval (see Socioeconomics)
- Detailed engineering design

Following successful completion of these activities, development of the mine could proceed.

Initial Mine Development

For purposes of projecting economics, the project schedule shown in Table 7 has been prepared. Initial mine development will consist of the following components:

- Development of twin rock tunnel declines with a TBM for approximately 10,600 ft. to the site of the dewatering station, a curved level section to the top of the dewatering station, and two inclined tunnels through rock and coal approximately 800 ft. to intercept seams 2 and 3.
- Development of the dewatering raise, decanting raise, sump, pump room and connecting crosscuts and entries.
- Installation of underground dewatering and slurry pumping equipment.
- Development of main levels to the limits of the area to be mined in each seam.

- Development of initial room panels starting with the uppermost seam No. 3, followed by No. 2 seam, and finally the No. 1 seam.
- Initial retreat mining starting in the upper No. 3 seam, followed by No. 2 seam, and finally No. 1.
- Construction of surface facilities during development of the rock tunnels.

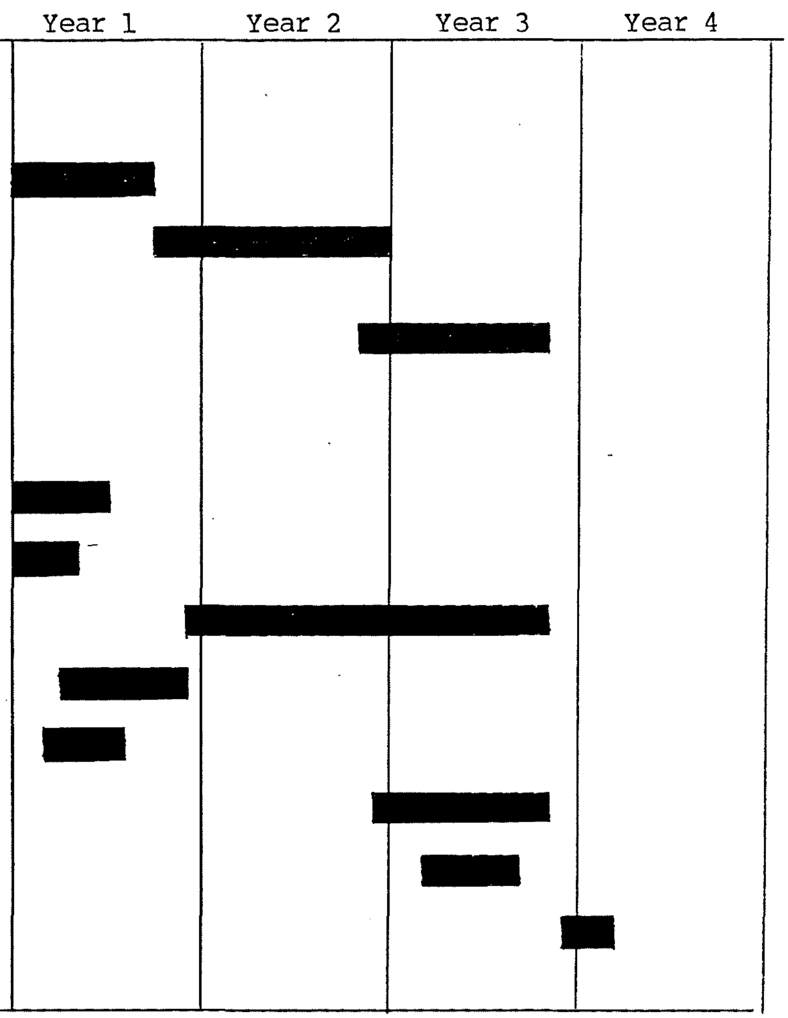
Tables 7 and 7-A are the construction and development schedules and show the estimated time allowed for each component until full productive capacity is reached. Table 8 shows the production schedule for the life of the property. Full production would be reached at the beginning of the 5th year and continue for 19 years, at which time seams 3 and 2 would be depleted. This production loss would probably be offset by development of reserves either to the north or south, or to the dip, with a second dewatering station; but the feasibility of reserves outside the area of this study is beyond the scope of this report. No. 1 seam reserves will last an estimated 31 years.

Assuming the north limit of the mining area is the same for all three seams, in order to avoid abutment pressures from the upper seams affecting the rooms in the lower seams, the first 2-room panel in the No. 3 seam would be mined out before starting development of the first panel in No. 2 seam. This requires that mining in No. 3 seam starts about 190 days ahead of No. 2 seam. In turn, No. 1 seam mining would be started 190 days after No. 2 seam.

TABLE 7

CONSTRUCTION AND DEVELOPMENT SCHEDULE

Description	Months
Rock Work and Construction	
TBM delivery	9
Rock tunnel driving	15
Dewatering station construction	11
Surface Construction	
Site preparation, mobile	6
Powerline	4
Preparation plant	23
Bathhouse/shop	8
Substation	5
Stockpiles and belts	11
Loadout	6
Plant start-up	3

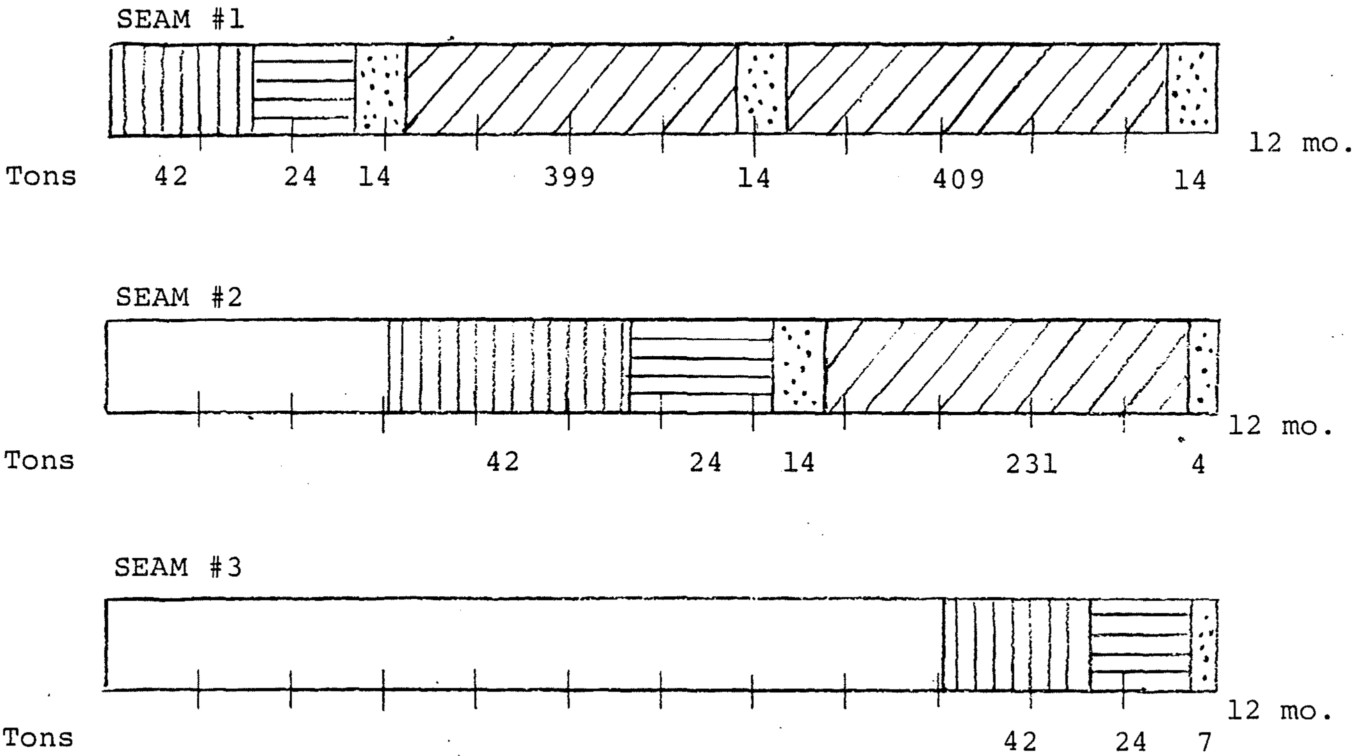


*Full capacity reached at beginning of 5th year.

TABLE 7-A

INITIAL PRODUCTION PERIOD (YEAR FOUR)

TONS (000)



PRODUCTION FIGURES

TONS CLEAN COAL (000)

SEAM 1	73
SEAM 2	315
SEAM 3	916
TOTAL	1304

FULL PRODUCTION STARTING YEAR 5 @ 3,000,000 TPY

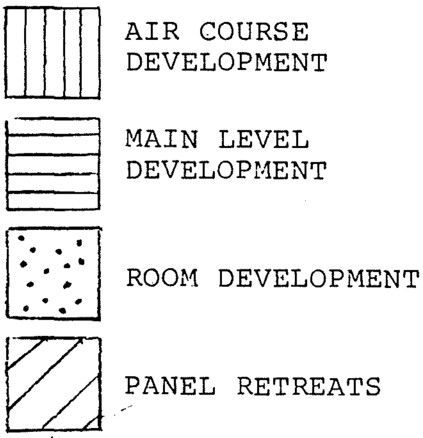


TABLE 8

PRODUCTION SCHEDULE

Year	Production (000's Tons of Clean Coal) Seam 1	Seam 2	Seam 3	Total
1	-	-	-	-
2	-	-	-	-
3	-	-	-	-
4	73	315	916	1,304
5	1,205	658	1,137	3,000
6	1,205	658	1,137	3,000
7	1,205	658	1,137	3,000
8	1,205	658	1,137	3,000
9	1,205	658	1,137	3,000
10	1,205	658	1,137	3,000
11	1,205	658	1,137	3,000
12	1,205	658	1,137	3,000
13	1,205	658	1,137	3,000
14	1,205	658	1,137	3,000
15	1,205	658	1,137	3,000
16	1,205	658	1,137	3,000
17	1,205	658	1,137	3,000
18	1,205	658	1,137	3,000
19	1,205	658	1,137	3,000
20	1,205	658	1,137	3,000
21	1,205	658	1,137	3,000
22	1,205	658	1,137	3,000
23	1,205	658	1,137	3,000
24	1,205	436	46	1,687
25	1,205	0	0	1,205
26	1,205	0	0	1,205
27	1,205	0	0	1,205
28	1,205	0	0	1,205
29	1,205	0	0	1,205
30	1,205	0	0	1,205
31	1,205	0	0	1,205
32	1,205	0	0	1,205
33	1,205	0	0	1,205
34	1,205	0	0	1,205
35	1,205	0	0	1,205
	37,428	13,253	22,565	73,246

FINANCIAL FORECAST

The economic and financial forecast for the hydraulic mine at Pittsburg & Midway's Kemmerer coal property includes:

- Operating Costs
  - Direct
  - Indirect
- Capital Expenditures
- Cash Flow Forecast and Internal Rate of Return

Operating costs per ton of clean coal are summarized in Table 9 and are based on the mine plans and manpower presented in this study. Manpower rates are the current 1983 UMWA rates; other costs have been extrapolated from experience at other coal mines. The direct cost per ton for labor, salaries, material and power are assumed to be 50% over normal for the start-up period, until full production is reached. Indirect costs include general and administrative, local taxes, insurance and royalty costs. The indirect costs are for the most part variable with tonnage, except for a few minor items, such as insurance and a part of general and administrative.

Capital expenditures are estimated in 1983 dollars and charged against cash flow based on the development and construction schedule, Table 7, the production schedule, Table 8, and the capital expenditure schedule, Table 10. Initial capital expenditures are summarized in Table 10 and are those expenditures required in the first five years of mining. Future expenditures required for rebuilding or replacement of worn out equipment are assumed to be expensed. Capital in this study does not account for any costs already incurred, such as purchase cost of the property, existing rail spurs, power lines, etc.

The cash flow forecasts, which are summarized in Table 12, have been prepared for three sales prices: $21.00, $23.50 and $26.00 per ton. The cash flow forecasts combine production, sales prices, operating costs, capital costs and income tax treatment, on a year-by-year basis for the life of the mine.

The internal rate of return (IRR) is the rate of return (discount factor) calculated for the series of unequal cash flows to give a present value at the beginning of the operation equal to zero.

TABLE 9

OPERATING COSTS

Production: 3,000,000 tons of clean coal per year

DIRECT COSTS		Cost/ton	
Payroll, payroll taxes, Table 9-A		$4.04	
Materials & supplies, excluding preparation plant, Table 9-B		1.47	
Repairs and maintenance, Table 9-C		0.44	
Power costs, Table 9-D		0.66	
Coal preparation, drying & loading, Table 9-E		0.93	
Other Direct, Table 9-F		0.15	
			$ 7.69
INDIRECT COSTS			
General & Administrative, Table 9-G		$0.50	
Black Lung tax, Table 9-H (1)		1.00	
OSM tax, Table 9-H		0.15	
UMWA Health & Welfare levy, Table 9-H (2)		1.70	
Misc. taxes & fees (excl. income), Table 9-H		0.15	
Insurance, Table 9-H		0.15	
Hydraulic royalty, Table 9-I		0.30	
			3.95
Sub-Total of operating costs before Wyoming state taxes			$11.64
Mineral royalties; not applicable on fee land			NAp
Coal price f.o.r.	$21	$23.50	$26
Wyoming state severance and ad valorem taxes for underground coal after January 1984 (Table 9-H), 13.75%	$ 2.89	$ 3.23	$ 3.58
Total operating cost (excludes capital)	$14.53	$14.87	$15.22

(1) The two components in Black Lung cost are: (1) hourly contribution, which is approximately 4% of direct wages; and (2) a "per ton" contribution, which for 1983 is $1.00.

(2) The two components which make up the Health & Welfare cost are: (1) hourly contribution of approximately $1.007 per man hour; and (2) a "per ton" contribution of $1.695.

TABLE 9-A
WAGES AND SALARIES
1983 Dollars

	Hourly Wage Earners				
	UMWA 7-1/4 hours /shift	UMWA 8 hrs. /shift	UMWA 9-1/4 hrs. /shift	Salaried employees	Total
Payroll cost per average worker shift					
Direct wage or salary, avg.	$93.26	$97.34	$120.30	$145.00	
Shift differential (23-1/4¢/hr.)	1.86	1.86	2.15	NAp	
Holiday pay and birthday pay	4.89	4.89	5.65	NAp	
Vacation pay (20 days average)	9.36	9.77	11.30	NAp	
Sick, bereavement pay (6 days avg)	2.13	2.22	2.57	NAp	
Unscheduled overtime (3% - 8 hr.)	2.80	2.92	NAp	NAp	
Clothing allowance	0.63	0.67	0.67	NAp	
Sub-Total	$114.93	$119.67	$142.64	$145.00	
Payroll taxes, benefits, etc.					
FICA (.067 of wages)	7.70	8.02	9.56	9.72	
FUI and SUI (3.96% of first $7,000 p.a. = $277/year)	1.16	1.16	1.16	1.16	
Workmen's Compensation	5.00	5.21	6.44	7.68	
Black Lung, hourly component (1)	3.73	3.89	4.81	5.80	
Sub-Total payroll taxes	$17.59	$18.28	$21.97	$24.36	
Insurance and Benefits					
UMWA H&W Fund, hourly cost (2)	7.30	8.06	9.31	NAp	
Life and medical insurance	10.00	10.00	10.00	10.00	
Accident insurance	0.70	0.70	0.70	1.00	
Dental insurance	1.00	1.00	1.00	1.00	
Pension	NAp	NAp	NAp	7.00	
Savings plan	NAp	NAp	NAp	1.50	
Sub-Total Insurance & Benefits	$19.00	$19.76	$21.01	$20.50	
Total Payroll Cost per Worker Day	$151.52	$157.71	$185.62	$189.86	
Worker shifts/year (Table 6)	7,887	27,812	15,394	19,465	70,558
Annual payroll cost (000's)	$1,195	$4,386	$2,857	$3,696	$12,134
Payroll cost per ton: ÷ 3,000,000 =					$4.04

(1) The two components in Black Lung cost are: (1) hourly contribution, which is approximately 4% of direct wages; and (2) a "per ton" contribution, which for 1983 is $1.00.

(2) The two components which make up the Health and Welfare cost are: (1) hourly contribution of approximately $1.007 per man hour; and (2) a "per ton" contribution of $1.695.

TABLE 9-B

MATERIALS AND SUPPLIES

	Useage /day	Cost per Unit	Cost per Day	$/ton
Roof support materials in rooms				
Bolts and crossbars used per day in rooms:				
Roof bolts	8	8	64	0.01
Crossbars, fiberglass or aluminum, reuseable	4	150	600	0.07
Aluminum props: total investment for full operations, 1,000 units @ $900 = $900,000. Assume 20%/year lost or worn out, @ $180,000				0.06
General mine roof support material				0.05
Flume and pipe replacements:				
Total initial capital cost for full production is $2,924,000; assume average life is 7 years, or 21,000,000 tons of clean coal. Cost per ton:				0.14
Rock dust, 1.0 lbs. @ $0.0125/lb.				0.01
Ventilation				
Stoppings and overcasts				0.10
Ventilation tubing				0.05
Bits, augers, tools				0.07
Explosives				0.01
Diesel fuel and lubricants				0.07
Safety supplies, cap lamps, oxygen apparatus, etc.				0.15
Dewatering and pumping station supplies:				
Total capital cost for all pumps, screens, and centrifuges is $5,313,000. Most of the investment is in the slurry pumps, which are inherently long wearing. The centrifugal pumps, having few moving parts, are also low in maintenance. Annual cost for parts for this group is assumed at 10% of the capital cost, or $531,000 per year				0.18
Other general supplies and miscellaneous				0.40
Freight and drayage				0.10
Total Materials and Supplies				$1.47

TABLE 9-C

REPAIRS AND MAINTENANCE
OTHER THAN LABOR

Development of rooms for hydraulic mining does not require shuttle cars, belt conveyors, or the complex equipment involved in longwall mining. To support a production of 3 million tons per year, only six simplified boom cutter type machines are required for driving the rooms and crosscuts. Having no gathering head or conveyor, these machines are comparatively low in maintenance. Moreover, only about 3% of the total tonnage is produced by these six machines, about 110,000 tons of raw coal per year, equivalent to 18,000 tons per machine per year. Hydraulic retreat mining also uses very simple, long-life equipment, i.e., monitor, feeder-breaker, high pressure pumps, and standard dewatering equipment. The boom cutters on the machines are replaced with monitors when the development of a pair of rooms is completed. The machine then becomes a crawler mounted feeder-breaker-monitor.

	Cost per Year	Cost per Clean ton
Repair and maintenance cost excluding preparation plant:		
Mining machines, et. al.	$330,000	0.11
Other mobile equipment, incl. monorail	300,000	0.10
Underground dewatering/pumping station	400,000	0.13
Rebuilds of other machinery	300,000	0.10
		$0.44

TABLE 9-D

POWER & OTHER COSTS

Excludes preparation plant, stockpiling and load-out

	Cost per clean ton
Assumes power is taken at 69 kv	
Average cost per kwh estimated at $0.03, the same as the average for the current surface mine. Most of the load for hydraulic mining would be pumps which would run comparatively steady compared to the open pit.	
Pumps:	
Underground slurry pumps: 3441 hp x .746 x 24 hrs/day x 358 x $0.03 ÷ 3,000,000 =	0.22
Low pressure water supplied by slurry pumps to the surface followed by gravity flow to faces.	
High pressure: 3 x 1900 hp x .746 x 24 hrs/day x 65% utilization x 358 x $.03 ÷ 3,000,000 =	0.24
Dewatering station: 600 hp x .746 x 24 hrs/day x 358 x 70% utilization x $.03 ÷ 3,000,000 =	0.03
Belt conveyor to the surface: 1700 x .746 x 24 x 358 x .03 ÷ 3,000,000 =	0.11
Face equipment: 6 continuous miner units x 300 hp each x .746 x 24 x 358 x .50 x .65 x .30 load factor x $.03 ÷ 3,000,000 =	0.01
Ventilation fans:	
6 fans at 10 hp each x .746 x 24 hrs/day x 365 days x $.03 ÷ 3,000,000 =	0.01
6 auxiliary fans, 10,000 cfm, 10 hp x .746 x 24 hrs/day x 365 days x $.03 ÷ 3,000,000 =	0.01
Underground lights and miscellaneous, assume: 50 kw x 24 hrs/day x 365 x $.03 ÷ 3,000,000 =	0.01
Surface, bathhouse, shop and other, assume 500 kw x 20 hrs/day x 240 days x $.03 ÷ 3,000,000 =	0.02
Total	$0.66

TABLE 9-E

PREPARATION, STOCKPILING AND LOADING
EXCLUDING LABOR

	Cost per clean ton
Materials and supplies	$ 0.23
Repair and maintenance supplies	0.15
Rebuilds and replacements	0.10
Plant power = 10 kwh/ton x $.03/kwh	0.30
Other	0.15
Total	$0.93

TABLE 9-F

OTHER DIRECT COSTS

	Cost/ton
Small vehicle use, private road maintenance, equipment rentals, utilities other than electric power, etc.	$0.15

TABLE 9-G

GENERAL AND ADMINISTRATIVE

A provision of $0.50/ton has been made in the estimate to cover:

	Cost/ton
Auto expense	
Contributions	
District office	
Ecology	
Consultants	
Legal and central accounting	
Marketing	
General, Administrative and Marketing Costs	$0.50

TABLE 9-H

TAXES, INSURANCE, HEALTH & WELFARE LEVY AND ROYALTIES

Taxes

Taxes, other than income taxes, are estimated as follows:

	Cost per ton
Black Lung tax	$1.00
OSM tax	0.15
Miscellaneous tax and fees, provision	0.15
Wyoming state taxes	

Sales price f.o.r.	$21	$23.50	$26
Severance tax*, effective January 1984, 7.25%	$1.52	$1.70	$1.89
Ad valorem tax, 6.5%	1.37	1.53	1.69
Total	$2.89	$3.23	$3.58

Health & Welfare levy, UMWA	1.70

Insurance

Includes insurance and any other taxes. These costs are affected by volume, book value and others. Based on experience with other operations, for the purpose of this report, an average of $0.15 is assumed to be representative.	$0.15

Royalties

The coal is owned in fee; therefore, no mineral royalty is applicable. A royalty to use B. C. Coal's proprietary know-how on hydraulic mining is assumed at 30¢ per ton	$0.30

* The 1983 session of the Wyoming Legislature effectively reduced the severance tax on underground coal from 10.5 to 7.25% effective January 1, 1984. Reference: Chapter 173, Original Senate File No. 88, "Mine Products Tax on Underground Coal." This Act became law on March 9, 1983 without the Governor's signature.

TABLE 10
CAPITAL EXPENDITURES

		Total ($000s)
Rock tunnels (Table 10-A)		
Tunnel boring machine, 14 ft. diam.	3,956	
Approx. 24,000 ft. tunnel	3,480	
Portals, fan, site preparation, etc.	150	
Enlarging 550 ft. for pump room and sump	187	
		7,773
Dewatering & pumping station (Table 10-B)		
Machinery	452	
Construction	1,325	
Rock excavation for raises	354	
		2,131
Hydraulic system (Table 10-C)		
Pumps	4,870	
Pipes	2,060	
Flumes	864	
		7,794
Mining equipment (Table 10-D)		
Development/retreat equipment	7,000	
Material supply, transp. & clean-up	375	
Main slope conveyor	2,330	
General	350	
		10,055
Monorail system (Table 10-E)		2,160
Electric power (Table 10-F)		
Surface	805	
Underground	872	
		1,677
Ventilation, compressed air, communications and monitoring systems (Table 10-G)		
Ventilation	480	
Compressed air	45	
Communications and monitoring	200	
		725
Coal preparation plant (Table 10-H)		
Preparation plant complete	13,100	
Rail loop, 2 miles	2,000	
Raw coal handling	490	
Clean coal handling and load-out	1,950	
Roads, fencing & other	850	
		18,390
Shop, bathhouse & ancillary facilities (Table 10-I)		1,450
Sub-Total Equipment & Construction		52,155
Management & engineering, exploration (10%)		5,216
Contingency (15%)		7,823
Operating capital, inventories, etc.		6,000
Total Capital Expenditures		71,194
Interest during construction @ 12% over 2 years		17,087
Total Capital (including interest during construction)		88,281

NOTE: Expenditures include freight and sales taxes as appropriate.

TABLE 10-B
CAPITAL EXPENDITURES
Dewatering Coarse Coal Underground*

	Unit	($000) Unit Cost	($000) Total	($000) Group Total
Machinery				
Head box & splitter, 2 ways	1	4	4	
S.D. vibrating screens, 8' x 20', 1/2" grid	2	45	90	
Sieve bends, 60° arc, 80" radius, 7' wide, 2 mm aperture	2	6	12	
Dewatering screens, 8' x 12', 1 mm grid	2	30	60	
Centrifuges (Wemco 1100 or equal)	2	38	76	
Overhead monorails & hoists, 20 ton	2	65	130	
Decant gate	1	25	25	
Feeder for unloading emergency sump	1	25	25	
Diverter for emergency decanting sump	1	30	30	
				452
Construction				
Concrete			100	
Steel: material and fabrication			200	
Erection of steel and equipment			200	
Piping material and erection, including connection to slurry pumps			250	
Electrical, etc.			350	
Ground support for raises			225	
				1,325
Rock Excavation				
Tunnels to be driven with TBM. The cost for this and their enlargement for the pump room and sump are included in Rock Tunnels, Table 10-E.				
Dewatering raise: 50' high by 40' avg. radius = 2327 cu. yds. at $70 per cubic yard			163	
Decanting raise			163	
Ventilation raise above pump room Raise: 6'x6'x70', @$400/ft.			28	
				354
Total Underground Dewatering				$2,131

*Fine coal dewatering included with preparation plant.

TABLE 10-C

CAPITAL EXPENDITURES

Hydraulic System - Pumps, Pipes & Flumes

	Units	Unit Cost ($000)	Total ($000)	Group Total
Pumps				
Slurry pumps, underground	5	650	3,250	
High pressure pumps, surface, Ingersoll-Rand, 6x13 DA-1-3, 4 stage, 2,500 HP	4	190	760	
Pump installations				
Slurry pumps	5	120	600	
Centrifugal pumps	4	65	260	
				4,870
Pipe		($)	($000)	
16" elec. weld, 5LX-X-52; 0.375" wall, in avg. lengths of 55 ft.	11,000'	12	132	
16" welds, X-rays, incl. installation	200	600	120	
10", Sched. 80, seamless, 40' lengths	42,000'	24	1,008	
10" couplings, Victaulic HP-70ES	1,050	80	84	
10" welds, incl. X-rays, installation	42,000'	10	420	
6", Sched. 80	18,000'	10	180	
6" couplings, Victaulic HP-70ES	900	27	24	
3" backfill line, Sched. 40, w/coup.	20,000'	3	60	
Compressed air line, 3" diam., with couplings	16,000'	2	32	
				2,060
Flumes				
Lined, main levels	9,600'	90	864	864
Total Pumps, Pipe & Flumes				7,794

TABLE 10-D

CAPITAL EXPENDITURES

Mining Equipment

	Units	Unit Cost ($000)	Total ($000)	Group Total ($000)
Development and retreat equipment				
Mining machines with bolters and feeder-breakers; interchangeable boom cutters and monitors	6	1,000	6,000	
Aluminum hydraulic jacks, 22 ton	1,000	.9	900	
Tools			100	
				7,000
Material supply, transportation & clean-up				
Diesel pick-ups	7	15	105	
Diesel material supply trucks	2	50	100	
Load-haul-dump (LHD - Eimco 913)			170	
				375
Main slope conveyor				
Terminal groups, 48", 1600 hp			500	
Structure and belting, 11,000' @ $130/ft.			1,430	
Delivery stacker to surface stockpile, 300 ft.			400	
				2,330
General equipment				
Tools			100	
Safety equipment, cap lamps, self-rescuers, etc.			250	
				350
Total mining equipment				10,055

TABLE 10-E

CAPITAL EXPENDITURES

Monorail System

		($000)		
	Units	Unit Cost	Total	Group Total
Equipment				
Diesel locomotives	4	195	780	
Shunting trolleys for short hauls	3	15	45	
Diesel powered manrider, seats 9	6	16	96	
Operating cabs	4	6	24	
Safety brakes	5	8	40	
Drawbars (9 ft.)	16	0.3	5	
Drawbars (4 ft.)	20	0.1	2	
Carrying trolley, w/3 ton air hoist	24	10	240	
Lifting beam, w/air hoist, 20 ton			40	
Containers	20	0.7	14	
Other			100	
				1,386
Monorail				
Surface yard: 1,000 ft. @ $115/ft.			115	
Rock tunnels: double trace, 24,000 ft. @ $15/ft.			360	
Dewatering and pumping station: 1,000 ft. @ $15/ft.			15	
Main levels: 3 x 3200 ft. = 9600 ft. @ $15/ft.			144	
Curved sections: 200 - 15 degree curve sections @ $150 each			30	
Switches: 22 @ $5,000 each			110	
				774
Total monorail system				2,160

TABLE 10-F
CAPITAL EXPENDITURES
Electric Power

	Unit	Unit Cost ($000)	Total ($000)	Group Total ($000)
Surface				
Substations				
69/13.8 kv; 20,000 kva, 2 outlets			250	
13.8/4.16 kv; 7500 kva, 4 outlets			75	
13,800/440 v; 2,000 kva, 10 outlets			40	
Power line, 69 kva, 1 mile			40	
Surface power dist., w/lines to fans			150	
Stand-by diesel generator, 7200 v, 1400 kw for mine dewatering			250	
				805
Underground				
Cable: insulated mine feeder, 15 kv w/coupling, installed; 11,000' @ $15/ft.			165	
#2 Panel feeder, w/couplers; 27,000' @ $6/ft.			162	
Controls			100	
Control system			100	
Power centers				
1500 kva, 7200/4160 v, 1 outlet, slurry pumps	5	50	250	
750 kva, 7200/440 v, dewatering station			35	
400 kva, 7200/440 v, roadheader/ feeder-breaker	3	20	60	
				872
Total electric power				1,677

TABLE 10-G
CAPITAL EXPENDITURES
Ventilation, Compressed Air, Communication and Monitoring

	Unit	Unit Cost ($000)	Total ($000)	Group Total ($000)
Ventilation				
Main fans, variable speed, 10 hp ea., installed	6	20	120	
Face ventilation, aux. fans w/tubing	6	10	60	
Overcasts, installed	6	20	120	
Air lock doors, installed	6	30	180	
				480
Compressed Air				
150 hp, 600 cfm compressor, installed at underground dewatering station			45	45
Communication and monitoring system				200
Total ventilation, compressed air, communication and monitoring				725

TABLE 10-H
CAPITAL EXPENDITURES
Coal Preparation

	($000) Total	Group Total
Coal preparation plant		
800 tph jig plant for 6" x 14M, 5 days/wk. operation	6,000	
200 tph fines (14M x 0) plant for 7 days/wk. operation	4,000	
Heat dryers	2,000	
Backfill storage tank and pump	500	
Water tank for industrial water	600	
		13,100
Rail loop		
Grading, ties and ballast, 2 miles @ $1,000/mile		2,000
Raw coal handling		
Transfer point, magnet, etc.	100	
6,000 ton coal pile with reclaimer	150	
1,000 ft. conveyor and structure w/terminal	100	
Terminal groups, 42", 2 @ $70,000 each	140	
		490
Clean coal handling and loadout		
Compacted dead storage pile, 200,000 tons, incl. equipment	1,000	
Unit train loadout, 30,000 ton live storage, w/conveyor	850	
Sampling system	100	
		1,950
Other		
Roads, paving and fencing	100	
Bulldozer, refuse truck, front-end loader	500	
Environmental, diversions, ponds, refuse	250	
		850
Total coal preparation		18,390

TABLE 10-I
CAPITAL EXPENDITURES
Shop, Bathhouse, Warehouse, Office, etc.

	($000) Total	Group Total
Bathhouse/office/warehouse/shop	1,200	
Domestic water tank, pipe, fire protection, and misc.	150	
Other	100	
Total shop, bathhouse, warehouse, office, etc.		1,450

TABLE 11

CAPITAL EXPENDITURE SCHEDULE
($000)

YEAR:	1	2	3	4	TOTAL
Rock tunnels					
TBM	3,956	-	-	-	3,956
Tunneling	480	3,000	-	-	3,480
Portals, fan, site prep.	150	-	-	-	150
Enlarging pump room	-	187	-	-	187
Dewatering & pumping station					
Machinery	-	-	452	-	452
Construction	-	-	1,325	-	1,325
Rock excavation	-	200	154	-	354
Hydraulic system					
Pumps	-	870	4,000	-	4,870
Pipes	-	0	1,500	560	2,060
Flumes	-	-	-	864	864
Mining equipment					
Development & retreat	-	-	7,000	-	7,000
Mat. supply, transportation and clean-up		175	200	-	375
Main slope conveyor	-	2,330	-	-	2,330
General	100	100	100	50	350
Monorail system	-	953	1,207	-	2,160
Power System					
Surface	300	100	405	-	805
Underground	-	100	700	72	872
Ventilation and compressed air					
Ventilation	-	-	480	-	480
Compressed air	-	45	-	-	45
Communication & monitoring	-	-	200	-	200
Coal preparation	3,000	6,000	9,390	-	18,390
Shop, bathhouse, warehouse, office, etc.	700	750	-	-	1,450
Sub-Total Capital	8,686	14,810	27,113	1,546	52,155
Management & engineering, 10%	869	1,481	2,711	155	5,216
Contingency	1,303	2,221	4,067	232	7,823
Operating capital	200	800	2,000	3,000	6,000
Total Capital Expenditures	11,058	19,312	35,891	4,933	71,194

Cash Flow Forecast

The cash flow forecast is based on sales revenue, operating costs, capital expenditures and the development schedule. Three cases for sales prices are used: $21, $23.50 and $26 per ton f.o.r. The depreciation, depletion and deferred tax deductions, such as loss carry forward, are non-cash cost deductions for tax purposes only. Cash flow is the sum of the net profit, depreciation, depletion, amortization and deferred deductions, or, conversely, cash flow equals the sales revenues minus operating costs, income tax, and capital expenditures. The cash flow is the after-tax money available to the company to pay off long and short term debts, pay for the use of the capital invested, and invest in new projects.

A detailed cash flow statement is shown for a sales price of $23.50 per ton (Table 12-C-1). Only summaries are shown for the other two sales prices.

Production

The cash flow analysis assumes that the saleable tonnage produced will remain constant for 19 years after full production is reached at the beginning of the 5th year. Thereafter, it will be reduced when Seams No. 2 and 3 are mined out in the 24th year.

Operating Costs

For each of the three sales price cases, operating costs, as shown in Table 8, vary due to Wyoming ad valorem and severance taxes. In year 4, 50% has been added to direct costs since the retreat/development ratio is about 4:1 instead of 34:1 when full production is reached.

Depreciation

Depreciation is estimated on the "accelerated cost recovery system" (ACRS) introduced by the Economic Recovery Tax Act of 1981, for qualifying assets. The IRS Guideline for Asset Depreciation Ranges, allows depreciation rates lower than 5 years; however, in this study the ACRS depreciable assets are written off over 5 years. Depreciation used in this study assumes 1981-84 rates as follows:

Five year ACRS percentages from year placed in service

Year 1	15%
Year 2	22%
Year 3	21%
Year 4	21%
Year 5	21%

Straight line depreciation over the life of the applicable reserves is used for assets not qualifying for 5 year ACRS depreciation. Salvage value is assumed to be nil at the end of the depreciation life and only operating capital is recovered in the last year.

Depletion

The tax deduction that allows an operator to recover the value of coal mined from his property is the depletion allowance. Percentage depletion, usually larger than "cost depletion," is used for this cash flow. Percentage depletion is 10% of the gross income after royalties from the sale of the coal during the tax year, but the deduction for depletion under this method cannot exceed 50% of taxable income from the property, after all deductions are allowed except the deduction for depletion.

Income Taxes

The tax rates used in this cash flow are the Federal Tax Rates. Wyoming has no State Income Tax. The tax rates used are:

	Federal
1st $25,000	17%
2nd $25,000	18%
3rd $25,000	30%
4th $25,000	40%
Over $100,000	46%

Tax Credit

The investment tax credit (ITC) allowance of 10% of the amount of the investment must have the following qualities:

(1) Be depreciable.

(2) Have a useful life of at least 3 years.

(3) Be tangible property.

(4) Be in service during the year.

For property depreciated under ACRS for 5 years, 100% of the investment qualifies for the 10% ITC. The investment tax credit does not affect the depreciation basis of an asset and is deducted directly from the income tax. Any unused credit can be carried forward up to seven years.

Capital Expenditures

Initial capital expenditures, as well as interest during development, are shown in Table 10. However, capital without interest is used in calculating the cash flows and rate of return.

Cash flow, as used in these calculations, does not include either interest charged on the negative flows, or interest earned on positive flows and the tax treatment thereof. This is the standard approach most commonly used for comparing alternate capital investments.

Rate of Return

The discounted cash flow rate of return (DCFROR; same as IRR) is usually defined as the rate of return after "present worthing" all cash outflow and cash inflow. IRR is the most common method of economic evaluation. The cash flows and IRR are based on three coal prices: $21, $23.50 and $26 per ton.

The internal rate of return is calculated on the bottom of the cash flow tables.

TABLE 12
SUMMARY OF CUMULATIVE CASH FLOWS
(without allowing for effects of interest)
($000)

Year	Sales Price F.O.R. Mine: Table 12-A $21/ton	Table 12-B $23.50 /ton	Table 12-C $26/ton
1	<11,058>	<11,058>	<11,058>
2	<30,370>	<30,370>	<30,370>
3	<66,262>	<66,262>	<66,262>
4	<67,778>	<64,948>	<62,144>
5	<48,368>	<39,058>	<29,942>
6	<28,958>	<13,351>	1,752
7	< 9,552>	12,098	25,669
8	9,592	30,257	47,253
9	26,907	47,827	68,651
10	40,633	65,396	90,048
11	54,358	82,966	111,446
12	68,084	100,535	132,844
13	81,809	118,105	154,241
14	95,534	135,675	175,639
15	109,260	153,244	197,036
16	122,985	170,814	218,434
17	136,711	188,383	239,832
18	150,436	205,953	261,229
19	164,161	223,523	282,627
20	177,887	241,092	304,024
21	191,612	258,662	325,422
22	205,338	276,231	346,820
23	219,063	293,801	368,217
24	226,933	303,832	300,401
25	232,653	311,097	389,203
26	238,373	318,361	398,005
27	244,093	325,625	406,806
28	249,813	332,889	415,608
29	255,533	340,153	424,410
30	261,253	347,417	433,212
31	266,973	354,681	442,014
32	272,693	361,946	450,816
33	278,413	369,210	459,618
34	284,133	376,474	468,420
35	295,853	389,738	483,222
Internal Rate of Return	18%	23%	27%
Payback period after full production reached	3.5 yrs.	2.5 yrs.	1.9 yrs.

TABLE 12-A
CASH FLOW FORECAST
Sales Price: $21/ton
($000)

Year	Shipments, Tons (000)	Revenues ($21.00 per ton)	Operating Costs ($14.53/ton) (1)	Operating Profit <Loss>	Total Taxes	Operating Cash Flow	Capital Expenditures	Cash Flow (2)	Cumulative Cash Flow
1	-0-	-0-	-0-	-0-	-0-	-0-	<11,058>	<11,058>	<11,058>
2	-0-	-0-	-0-	-0-	-0-	-0-	<19,312>	<19,312>	<30,370>
3	-0-	-0-	-0-	-0-	-0-	-0-	<35,892>	<35,892>	<66,262>
4	1,340	27,384	23,968	3,416	-0-	3,416	< 4,932>	< 1,516>	<67,778>
5	3,000	63,000	43,590	19,410	-0-	19,410	-0-	19,410	<48,368>
6	3,000	63,000	43,590	19,410	-0-	19,410	-0-	19,410	<28,958>
7	3,000	63,000	43,590	19,410	<4>	19,406	-0-	19,406	< 9,552>
8	3,000	63,000	43,590	19,410	<266>	19,144	-0-	19,144	9,592
9	3,000	63,000	43,590	19,410	<2,095>	17,315	-0-	17,315	26,907
10	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	40,633
11	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	54,358
12	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	68,084
13	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	81,809
14	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	95,534
15	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	109,260
16	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	122,985
17	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	136,711
18	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	150,436
19	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	164,161
20	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	177,887
21	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	191,612
22	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	205,338
23	3,000	63,000	43,590	19,410	<5,685>	13,725	-0-	13,725	219,063
24	1,687	35,427	24,512	10,915	<3,045>	7,870	-0-	7,870	226,933
25	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	232,653
26	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	238,373
27	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	244,093
28	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	249,813
29	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	255,533
30	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	261,253
31	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	266,973
32	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	272,693
33	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	278,413
34	1,205	25,305	17,509	7,796	<2,076>	5,720	-0-	5,720	284,133
35	1,205	25,305	17,509	7,796	<2,076>	5,720	6,000 (3)	11,720	295,853

(1) Year 4, $18.38.
(2) Without allowing for effects of interest.
(3) Year 35 - $6,000,000 operating capital recovered.

Internal Rate of Return = 18%
Payback period after full production reached: 3.5 years

TABLE 12-B
CASH FLOW FORECAST
Sales Price: $23.50/ton
($000)

Year	Shipments, Tons (000)	Revenues ($23.50 per ton)	Operating Costs ($14.87 ton) (1)	Operating Profit <Loss>	Total Taxes	Operating Cash Flow	Capital Expenditures	Net Cash Flow (2)	Cumulative Cash Flow
1	-0-	-0-	-0-	-0-	-0-	-0-	<11,058>	<11,058>	<11,058>
2	-0-	-0-	-0-	-0-	-0-	-0-	<19,312>	<19,312>	<30,370>
3	-0-	-0-	-0-	-0-	-0-	-0-	<35,892>	<35,892>	<66,262>
4	1,304	30,644	24,398	6,246	-0-	6,246	< 4,932>	1,314	<64,948>
5	3,000	70,500	44,610	25,890	-0-	25,890	-0-	25,890	<39,058>
6	3,000	70,500	44,610	25,890	<183>	25,707	-0-	25,707	<13,351>
7	3,000	70,500	44,610	25,890	<441>	25,449	-0-	25,449	12,098
8	3,000	70,500	44,610	25,890	<7,731>	18,159	-0-	18,159	30,257
9	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	47,827
10	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	65,396
11	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	82,966
12	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	100,535
13	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	118,105
14	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	135,675
15	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	153,244
16	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	170,814
17	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	188,383
18	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	205,953
19	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	223,523
20	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	241,092
21	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	258,662
22	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	276,231
23	3,000	70,500	44,610	25,890	<8,320>	17,570	-0-	17,570	293,801
24	1,687	39,645	25,086	14,559	<4,527>	10,031	-0-	10,031	303,832
25	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	311,097
26	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	318,361
27	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	325,625
28	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	332,889
29	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	340,153
30	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	347,417
31	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	354,681
32	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	361,946
33	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	369,210
34	1,205	28,318	17,918	10,399	<3,135>	7,264	-0-	7,264	376,474
35	1,205	28,318	17,918	10,399	<3,135>	7,264	6,000 (3)	13,264	389,738

(1) Year 4, $18.71.
(2) Without allowing for effects of interest.
(3) Year 35 - $6,000,000 operating capital recovered.

Internal Rate of Return = 23%

Mine payback period after full production reached: 2.5 years

TABLE 12-B-1 (Part I)
CASH FLOW FORECAST
Sales Price: $23.50/ton
($000)

Year:	1	2	3	4	5	6	7	8	9	10	11	12
Shipments, Clean Tons (000)	-0-	-0-	-0-	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000
Revenues @ $23.50/ton	-0-	-0-	-0-	30,644	70,500	70,500	70,500	70,500	70,500	70,500	70,500	70,500
Operating Cost @$14.87/ton(1)	-0-	-0-	-0-	24,398	44,610	44,610	44,610	44,610	44,610	44,610	44,610	44,610
Operating Profit <Loss>	-0-	-0-	-0-	6,246	25,890	25,890	25,890	25,890	25,890	25,890	25,890	25,890
Federal Taxes(2)												
Depreciation	1,154	3,182	7,609	9,507	9,400	7,923	6,185	1,114	708	708	708	708
Tax Loss Forward	-0-	1,154	4,336	11,945	18,270	8,830	2,482	-0-	-0-	-0-	-0-	-0-
Percentage Depletion	-0-	-0-	-0-	3,064	7,050	7,050	7,050	7,050	7,050	7,050	7,050	7,050
50% Limit	-0-	-0-	-0-	-0-	-0-	4,568	8,612	12,388	12,591	12,591	12,591	12,591
Taxable Income	-0-	-0-	-0-	-0-	-0-	4,568	10,174	17,726	18,132	18,132	18,132	18,132
Income Tax	-0-	-0-	-0-	-0-	-0-	<2,081>	<4,660>	<8,134>	<8,320>	<8,320>	<8,320>	<8,320>
Investment Tax Credit	1,086	1,851	3,389	193	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
ITC Forward	-0-	1,086	2,937	6,326	6,519	6,519	4,621	403	-0-	-0-	-0-	-0-
ITC Limit	-0-	-0-	-0-	-0-	-0-	1,898	4,219	403	-0-	-0-	-0-	-0-
Total Taxes	-0-	-0-	-0-	-0-	-0-	<183>	<441>	<7,731>	<8,320>	<8,320>	<8,320>	<8,320>
Operating Cash Flow	-0-	-0-	-0-	6,246	25,890	25,707	25,449	18,159	17,570	17,570	17,570	17,570
Capital Expenditures	<11,058>	<19,312>	<35,892>	4,932	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Net Cash Flow(4)	<11,058>	<19,312>	<35,892>	1,314	25,890	25,707	25,449	18,159	17,570	17,570	17,570	17
Cumulative Cash Flow	<11,058>	<30,370>	<66,262>	<64,948>	<39,058>	<13,351>	12,098	30,257	47,827	65,396	82,966	100,535

(1) Year 4 - $18.71
(2) No Wyoming tax.
(3) Year 35 - $6,000,000 operating capital recovered.
(4) Without allowing for effects of interest.

Internal Rate of Return = 23%

Payback Period After Full Production Reached: 2.5 years

TABLE 12-B-1 (Part II)
CASH FLOW FORECAST
Sales Price: $23.50/ton
($000)

Year:	13	14	15	16	17	18	19	20	21	22	23	24
Shipments, Clean Tons (000)	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	3,000	1,687
Revenues @ $23.50/ton	70,500	70,500	70,500	70,500	70,500	70,500	70,500	70,500	70,500	70,500	70,500	39,645
Operating Cost @$14.87/ton (1)	44,610	44,610	44,610	44,610	44,610	44,610	44,610	44,610	44,610	44,610	44,610	25,086
Operating Profit <Loss>	25,890	25,890	25,890	25,890	25,890	25,890	25,890	25,890	25,890	25,890	25,890	14,559
Federal Taxes (2)												
Depreciation	708	708	708	708	708	708	708	708	708	708	708	708
Tax Loss Forward	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Percentage Depletion	-0-	-0-	-0-	-0-	3,064	7,050	7,050	7,050	7,050	7,050	7,050	3,964
50% Limit	12,591	12,591	12,591	12,591	12,591	12,591	12,591	12,591	12,591	12,591	12,591	6,925
Taxable Income	18,132	18,132	18,132	18,132	18,132	18,132	18,132	18,132	18,132	18,132	18,132	9,886
Income Tax	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<4,527>
Investment Tax Credit	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
ITC Forward	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
ITC Limit	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Total Taxes	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<8,320>	<4,527>
Operating Cash Flow	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	10,031
Capital Expenditures	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Net Cash Flow (4)	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	17,570	10
Cumulative Cash Flow	118,105	135,675	153,244	170,814	188,383	205,953	223,523	241,092	258,662	276,231	293,801	303,832

(1) Year 4 - $18.71
(2) No Wyoming tax.
(3) Year 35 - $6,000,000 operating capital recovered.
(4) Without allowing for effects of interest.

Internal Rate of Return = 23%

Payback Period After Full Production Reached: 2.5 years

TABLE 12-B-1 (Part III)
CASH FLOW FORECAST
Sales Price: $23.50/ton
($000)

Year:	25	26	27	28	29	30	31	32	33	34	35
Shipments, Clean Tons (000)	1,205	1,205	1,205	1,205	1,205	1,205	1,205	1,205	1,205	1,205	1,205
Revenues @ $23.50/ton	28,318	28,318	28,318	28,318	28,318	28,318	28,318	28,318	28,318	28,318	28,318
Operating Cost @$14.87/ton (1)	17,918	17,918	17,918	17,918	17,918	17,918	17,918	17,918	17,918	17,918	17,918
Operating Profit <Loss>	10,399	10,399	10,399	10,399	10,399	10,399	10,399	10,399	10,399	10,399	10,399
Federal Taxes (2)											
Depreciation	708	708	708	708	708	708	708	708	708	708	708
Tax Loss Forward	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Percentage Depletion	2,832	2,832	2,832	2,832	2,832	2,832	2,832	2,832	2,832	2,832	2,832
50% Limit	4,845	4,845	4,845	4,845	4,845	4,845	4,845	4,845	4,845	4,845	4,845
Taxable Income	6,859	6,859	6,859	6,859	6,859	6,859	6,859	6,859	6,859	6,859	6,859
Income Tax	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>
Investment Tax Credit	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
ITC Forward	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
ITC Limit	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Total Taxes	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>	<3,135>
Operating Cash Flow	7,264	7,264	7,264	7,264	7,264	7,264	7,264	7,264	7,264	7,264	7,264
Capital Expenditures	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	6,000 (3)
Net Cash Flow (4)	7,264	7,246	7,264	7,264	7,264	7,264	7,264	7,264	7,264	7,264	13,264
Cumulative Cash Flow	311,097	318,361	325,625	332,889	340,153	347,417	354,681	361,946	369,210	376,474	389,738

(1) Year 4 - $18.71
(2) No Wyoming tax.
(3) Year 35 - $6,000,000 operating capital recovered.
(4) Without allowing for effects of interest.

Internal Rate of Return = 23%

Payback Period After Full Production Reached: 2.5 years

TABLE 12-C
CASH FLOW FORECAST
Sales Price: $26/ton
($000)

Year	Shipments, Tons (000)	Revenues ($26.00 per ton)	Operating Costs ($15.22/ton) (1)	Operating Profit <Loss>	Total Taxes	Operating Cash Flow	Capital Expenditures	Cash Flow (2)	Cumulative Cash Flow
1	-0-	-0-	-0-	-0-	-0-	-0-	<11,058>	<11,058>	<11,058>
2	-0-	-0-	-0-	-0-	-0-	-0-	<19,312>	<19,312>	<30,370>
3	-0-	-0-	-0-	-0-	-0-	-0-	<35,892>	<35,892>	<66,262>
4	1,304	33,904	24,854	9,050	-0-	9,050	< 4,932>	4,118	<62,144>
5	3,000	78,000	45,660	32,340	<137>	32,203	-0-	32,203	<29,942>
6	3,000	78,000	45,660	32,340	<646>	31,694	-0-	31,694	1,752
7	3,000	78,000	45,660	32,340	<8,423>	23,917	-0-	23,917	25,669
8	3,000	78,000	45,660	32,340	<10,756>	21,584	-0-	21,584	47,253
9	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	68,651
10	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	90,048
11	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	111,446
12	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	132,844
13	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	154,241
14	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	175,639
15	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	197,036
16	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	218,434
17	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	239,832
18	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	261,229
19	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	282,627
20	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	304,024
21	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	325,422
22	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	346,820
23	3,000	78,000	45,660	32,340	<10,942>	21,398	-0-	21,398	368,217
24	1,687	43,862	25,676	18,186	< 6,002>	12,184	-0-	12,184	300,401
25	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	389,203
26	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	398,005
27	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	406,806
28	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	415,608
29	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	424,410
30	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	433,212
31	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	442,014
32	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	450,816
33	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	459,618
34	1,205	31,330	18,340	12,990	< 4,188>	8,802	-0-	8,802	468,420
35	1,205	31,330	18,340	12,990	< 4,188>	8,802	6,000 (3)	14,802	483,222

(1) Year 4, $19.06 per ton.
(2) Without allowing for effects of interest.
(3) Year 35 - $6,000,000 operating capital recovered.

Internal Rate of Return = 27%

Payback period after full production reached: 1.9 years

MARKET

Factors which may influence stronger demand for this type of coal in the future are:

1. Acid rain legislation, if enacted, would make the use of high sulfur Midwestern coals less economical. Kemmerer coal would then enjoy increased demand.
2. The present oversupply of steam generation plants in the U.S. will be absorbed, increasing the market for this type of coal.
3. The economy will recover, and the consumption of electric power and cement and, in turn, coal will grow.
4. The oversupply and consequent low prices of oil and gas are seen by most energy experts to be temporary. They are forecasting, over the long term, reduced world reserves, higher prices in real terms and an increase in the switchover to coal and alternate forms of energy.
5. The western and southwestern areas of the United States are growing at a faster rate than the rest of the country.
6. The coal product from this mine would be a washed, thermally dried product, low in ash, low in sulfur, with medium btu. Following is a typical analysis: ash, 2.5%; sulfur, 0.3%, btu, 10,000.
7. The coal, while not so high in btu, otherwise is an ideal thermal coal product for utilities, the fastest growing market segment in tonnage in the U.S.

Pittsburg & Midway's marketing department keeps abreast of the market for their coal. For the purpose of this feasibility study, it is their opinion that under current economic conditions and those anticipated for the next few years, the market price for a washed, thermally dried coal, having the analyses expected, would range in 1983 dollars from $21 to $26, with $23.50 per ton being the mid-point.

SOCIOECONOMICS

Introduction

The following discussion addresses the socioeconomic, environmental and permitting considerations for development of a hydraulic mine at the Kemmerer, Wyoming site.

The Kemmerer Coal Company, which was established in 1897, was instrumental in the founding of the town which bears its name. Prior to 1954, coal was mined from as many as 10 underground mines.

With the decline in use of coal during the postwar years, the area's economy declined. In 1950, surface mining was initiated at the Elkol Mine. In 1963, the Sorensen tipple was established to supply coal to Utah Power & Light Company's 715,000-kw mine-mouth Naughton power plant. Pittsburg & Midway Coal Company, a subsidiary of Gulf Oil Corporation, purchased Kemmerer Coal Company in 1980. The company's headquarters are located in the original company town of Frontier, just north of Kemmerer.

The present Kemmerer surface operation includes the 1-U-D pit, or "Big Pit", the deepest and one of the largest surface coal mines in the United States. There, up to 12 major coal seams are mined. Current production from the South Block (Elkol Sorenson) is ranging from 3 to 4 milllon tons per year.

The hydraulic mining site is located adjacent to the ultimate highwall, as currently planned, down dip from the 1-U-D pit. The site lies a few miles southwest of Kemmerer in Lincoln County, Wyoming.

The towns of Kemmerer, Diamondville and Frontier represent the population center of the area. The population of the Kemmerer-Diamondville area is estimated at over 4,272.

If hydraulic mining were to be implemented at Kemmerer, there are two possibilities for its use:

- The underground hydraulic mine could be phased in gradually to replace the surface mine, as reserves with low stripping ratios are depleted. In this case, there would be no socio-economic impact since productivity and employment would be about the same.
- If the demand for coal increased enough to warrant underground mining in addition to the current surface mining, employment would increase by about 100 persons for each one million tons of added capacity.

In the event that hydraulic mining were added to present operations, the following conditions would apply.

Labor Requirements

There are approximately 525 employees at the Kemmerer Coal Company. Of these, the majority of hourly employees are represented by the United Mine Workers of America (UMWA).

Assuming an increase of 318 permanent employees for the hydraulic mine and assuming multipliers of .76 for indirect employment and an average of 2.89 people per family, the project would increase the area population by 1,617 people.

The following population projections by the 1982 Lincoln County Community Development Study and the Wyoming Planning Report are extrapolated to include the additional population:

	1980	1990	1990 with Project
Lincoln County	12,177	15,392	17,009, 11% increase
Kemmerer-Diamondville	4,273	6,736	7,949, 18% increase
County Mine Employment	1,350	1,605	1,923, 20% increase

The county's unemployment rate in 1982 was 10.9%, and in July 1983, 12.7%. In general, development of Wyoming's energy resources is expected to continue, which may make labor availability in 1986-1990 a concern. However, in light of the present industrial layoffs within other industries in the intermountain area and the current depressed state of uranium mining in Wyoming, labor availability for the near term should not be a deterrent to selection of the Kemmerer site.

Availability of Trade Contractors, Construction Materials and Supplies

In Lincoln County, there are 395 people employed in the construction sector (down from 568 in 1981) and many small trade contractors.

With other underground mines in the area, trona and coal, mine supplies are readily available.

Construction material is available in the area for surface facility construction.

Infrastructure Conditions

To date, growth in the Kemmerer-Diamondville area has been absorbed with little problem as compared to Rock Springs or Gillette. The Lincoln County population grew by 40.9% between 1970 and 1980 and kept pace with Wyoming, which experienced a 41.6% growth rate. The Kemmerer-Diamondville area grew by 53.8% in population between 1970 and 1980. Services have generally kept pace with this growth, and apparently the cities, county and State are successful at accommodating the impact. The expansion due to additional mine development may be more easily absorbed in this area, where state and local governments are experienced in overcoming these impacts, than in other parts of the country where there is little experience.

The following are assessments of public service systems from the Lincoln County Development Study of March 1982:

- Lincoln County

  The county provides the following major services: Sheriff's Department, solid waste disposal, library and roads. Human services are provided by the county as well as private groups. The county services are generally in good condition. Where there are deficiencies, these are in the process of being improved (new land fill sites, relatively new jail and public safety building, new library under construction, etc.) Road and bridge funds are not keeping pace with costs, and, until recently, inflation has been strongly felt by all agencies responsible for road and bridge maintenance. There are no county expenditures for park and recreation, housing or utility services and facilities. The county policy is to rely on the incorporated areas to provide urban type services to urban density development.

- Kemmerer

  Kemmerer is the county seat of Lincoln County. The basic utility systems of water and sewer, which also serve Diamondville, are in relatively good condition. Some improvements to the water system are needed, but most are already funded from a Farm Loan Board grant and city funds. Both systems, after the projected improvement, could serve up to 10,000 people, which would cover development even beyond the addition of a new hydraulic mine. The major facility needs are for municipal office space and for the police department. There is a need to develop a north-south major arterial on the west side of town which would open considerable area for development. There is partial funding ($500,000) from the State Highway Department for this route. The town has two parks, a swimming pool, museum, three ball fields and a 9-hole golf course.

- Diamondville

  The town of Diamondville is contiguous with Kemmerer to the southeast. Politically, the towns are separate, but services are integrated. Diamondville has one police officer who uses Lincoln County facilities in Kemmerer. The town has one park, picnic facilities and a ball field. None of the roads in Diamondville is paved.

Environmental Conditions and Control Measures Required

The Kemmerer surface mine has been permitted in accordance with PL 95-87, the Surface Mining Control and Reclamation Act (SMCRA). While underground mining would cause minimal environmental impact, some of the possible concerns are:

- Surface subsidence in the area of hydraulic mining.
- Surface facilities near the proposed rock tunnels, including a preparation plant, stockpiles and train loadout.
- More potential coal dust emissions due to more coal production.
- Possible discharge of ground water from the mine, coupled with the drawdown of the ground water aquifers.

These concerns are of minor consequence when compared with the impact of existing surface operations. Little surface disturbance will be required for the surface facilities proposed. If the underground hydraulic mine was developed to replace all or part of the surface mine, the net effect on the environment would be beneficial.

Subsidence

The removal of 170 ft. of coal from three seams will cause subsidence of the surface. However, such subsidence can be accomplished with only minor disturbance to the soil and vegetation. Some recontouring, reseeding and filling of fissures can be expected for full reclamation after subsidence is complete, and, with modest effort, the surface can be left in better condition than before mining. There are no known improvements, such as roads, pipelines, or power lines which would require relocation. Fencing and posting of approximately one square mile would be in order while mining is underway.

Surface Facilities and Preparation Plant

The surface facilities will require full reclamation following mining. Other than surface drainage control, the only other concern during operations is preparation plant refuse disposal. A substantial portion of the fines will be returned underground to be used for sealing old workings in the mine; the remainder of the refuse can easily be buried in backfilling operations at the surface mine.

Air Quality

The raw coal from the dewatering station will contain enough surface moisture to reduce the usual dust emissions from belt and transfer points and raw coal storage piles. The thermally dried clean coal would add only a minor amount.

Water Quality and Ground Water

Moderate inflows of water are expected with underground mining; however, the hydraulic system does consume some water, which is lost with the raw coal. Since some discharge may be expected, the water would be passed through sedimentation ponds, evaporated or treated as required for NPDES and state standards. Surplus water, if any, could be used to irrigate the affected surface. Because of the naturally arid climate, such irrigation would greatly increase the forage for wild game or domestic grazing stock, probably four to five fold.

Housing and Transportation of Personnel

It is assumed that most new miners will locate in the Kemmerer-Diamondville area. The Community Development Study of March 1982, by Briscoe, Maphis, Murray & Lamont, Inc. describes the county-wide housing growth in 1960, 1970 and 1980. This study shows that housing has grown at a more rapid pace than population, due to vacation homes in the northern part of the county. Between 1970 and 1980 housing in Kemmerer increased by 508 units, 5.1%, while population increased by 1,981, 3.6%. Diamondville's population increased 7.5% and housing increased at 6.3%. If the addition of a hydraulic mine at Kemmerer was to increase overall production, the projected 1990 population growth would accelerate by as much as 18%. In this event, additional housing units should be planned early in the project.

Laws and Regulations

Permitting of an underground hydraulic mine at Kemmerer would either require a new permit pursuant to the Surface Mining and Reclamation Act, or a modification of the existing permit to incorporate the underground mine. Permitting requirements for a hydraulic mine would require the following major permitting activities:

- Revision of all permits required by EPA and Wyoming Department of Environmental Quality, DEQ.
- Development of a mining reclamation plan submitted to DEQ.
- MSHA mine plans including plans for roof control, ventilation, training, etc.

While the required permitting is routine and time-consuming, there is no known reason for denial. The time required would probably depend, at least partly, on the economic conditions at the time and the desire of state and federal officials to increase economic activity and employment.

Conclusion

If general economic conditions recover slowly and steadily, and, specifically, if energy demand improves, the Kemmerer-Diamondville area will be favorably impacted. The area is well positioned to absorb healthy economic growth. The hydraulic mine planned herein would have an especially beneficial effect if it became the means of sustaining mining activity after the depletion of the surface mine reserves, thereby forestalling economic collapse. This report proposes a new system of mining to not only more fully recover the thick seams of the vast underground reserves, but also to improve the economics of underground mining.

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