

**A mining research contract report
JUNE 1983**



UNDERGROUND HIGH VOLUME HYDRAULIC COAL MINING

**Volume II
Price River Coal Company
Castlegate Mine**

OFR 86-35B

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

**BUREAU OF MINES
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REPORT DOCUMENTATION PAGE	1. REPORT NO.	2.	3. Recipient's Accession No.
4. Title and Subtitle Underground High Volume Hydraulic Coal Mining Volume II, Price River Coal Company Castlegate Mine			5. Report Date June 1983
7. Author(s) R. G. Heers, Mining Consultant, for B. C. Coal, Ltd. (Westar)			6.
9. Performing Organization Name and Address Prime Contractor: B. C. Coal, Ltd. (Westar Mining) 1176 West Georgia Street Vancouver, British Columbia Canada V6E 4B8 Sub-Contractor: R. G. Heers & Associates, Consultants 1425 Beneficial Life Tower Salt Lake City, Utah 84111			8. Performing Organization Rept. No.
12. Sponsoring Organization Name and Address Department of Energy 2401 East Street, N.W. Washington, D.C. 20241 U. S. Bureau of Mines Twin Cities Mining Research Center 5629 Minnehaha Avenue, South Minneapolis, Minnesota 55417			10. Project/Task/Work Unit No.
			11. Contract(G) or Grant(G) No. (C) J0233911 (G)
			13. Type of Report & Period Covered Mining Research Contract February 1982-Feb. 1984
15. Supplementary Notes Extension of: Underground High Volume Hydraulic Coal Mining, Interim Phase I Report, Kaiser Resources (US) Ltd., January 1981. D.O.E. Contract DE-AC01-79ET14232.			14.
16. Abstract (Limit: 200 words) This volume, one of five in this study, examines the technical and economic feasibility of hydraulically mining a thick coal lens, from 12-25 ft., in the "A" bed on the west side of Price River Coal Company's property in Carbon County, Utah. Although the coal is hard and strong and pitches only 7-8°, hydraulic mining at a rate of 1,154,000 tpy was determined to be highly feasible using an induced caving system. With a selling price range of \$25-\$35 per ton f.o.r., the estimated rate of return on the incremental capital of about \$25 million would be 17%-41%. Overall productivity would be 29.1 clean tons per man day. The coal would be developed by continuous miners in widely spaced dual rooms driven up the full pitch. About 83% of the production would be extracted from pillars and top coal by hydraulic monitors in the rooms, supported by yieldable aluminum arches. In the average seam thickness of 20 ft., it is estimated that the tonnage mined from arched headings will constitute only 3.4% of the total. Monitor production is estimated to average over 1500 tons/shift. Most of the coal breaking in the retreat mining faces would be caused by induced caving, using strata and abutment pressures in a controlled manner. The coal would be dewatered underground. Safety would be enhanced by: controlling the roof and ribs with lightweight arches, remote controlled extraction of pillars and top coal, reduction of airborne dust, and positive ventilation of pillar faces. Both mining recovery and productivity would be improved over any other system used for such thick seams.			
17. Document Analysis a. Descriptors HYDRAULIC MINING COAL MINING UNDERGROUND MINING b. Identifiers/Open-Ended Terms MINE PLANNING WATER JETS BITUMINOUS c. COSATI Field/Group 0809 HYDRAULIC MINING			
18. Availability Statement Unlimited		19. Security Class (This Report) Unclassified	21. No. of Pages 152
		20. Security Class (This Page) Unclassified	22. Price

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 is a summary of the work recently completed as a part of this contract during the period February 1, 1982 to February 29, 1984. This report was submitted by the authors in February 1984.

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., China, U.S.S.R., and Japan, 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 system, 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. This work was extended to study in more depth five favorable sites in the U.S., three bituminous in the West and two anthracite in Pennsylvania. This volume reports on the feasibility of hydraulically mining a thick reserve of coal (12 to 25 ft.) in the A-seam of the Price River Coal Company holdings at Castlegate, Utah.

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INTRODUCTION

The subject of this report is the feasibility of hydraulically mining a thick coal seam, ranging up to 26 ft., in a portion of the Price River Coal Co. property near Helper, Utah.

Numerous old coal properties and contiguous virgin coal land centered around the old mines near Helper and Castlegate, Utah have been amalgamated and modernized by the Price River Coal Co., a subsidiary of American Electric Power. The overall property comprises 27,000 acres, controls 15 miles of outcrop, and contains 700-800 million tons of coal in place, of which 270-300 million is estimated by independent consultants to be proven, clean recoverable. The coal is high quality, low sulfur steam coal with a Btu content of 12,400 per lb. after washing. Wash plant yield for the west side of the property, where the thick coal lens is located, is estimated to be 86%.

The property is located in the Book Cliffs coal field in Carbon County, 8 miles north-northwest of Price, Utah and 110 miles southeast of Salt Lake City. This area is the most developed part of the Utah coal fields and is on the main line of the D&RG railroad and the main highway between Price and Salt Lake City, Utah.

The Price River divides the property into two parts, the area to the west comprising the largest reserves and the thickest seams. Before the recent economic slowdown, the coal seams in the west side were being mined by continuous miner and longwall methods at a rate of approximately one million tons per year.

To rationalize the transportation of men and material, and to greatly increase the ventilation capacity on the west side, two large diameter shafts have recently been completed to a depth of 1425 ft. These shafts, located in Crandall Canyon, penetrate, in descending order, three workable seams of coal: the Castlegate

"D", the "A" and the "Sub 3." An especially thick lens of coal, ranging up to 26 ft. thick, occurs in the A-seam. In this one lens alone and within the present Government lease boundaries, the reserves that have a thickness of more than 12 ft. amount to an estimated 30,000,000 tons in place. Additional reserves in the same thick lens lie down dip to the north of the present lease. To provide a more economical layout of the mine, some of these reserves have been included in the plan. There are other thick lenses on the property and contiguous federal land.

The southwestern edge of this block of thick coal lies about 2500 ft. from the new Crandall Canyon shafts, and the center of the lens is about 9,000 ft. to the east-northeast.

Reserves above 12 ft. thick are not normally treated as recoverable because, in the past, there has not been a proven way in the U.S. of mining such thick seams in deep cover with acceptable recovery. The success of the hydraulic mine in Canada, operated by B. C. Coal at Sparwood, B.C., has changed the prospects for thick seam mining in gradients of 6 degrees or more.

Because the coal at this site is hard, under deep cover, and overlain by strata containing many strong, thick, massive sandstones, the hydraulic mining method developed in this study combines the best and most applicable elements of several other methods:

- Advance room and pillar mining with continuous miners;
- Undercut block caving, and
- Induced caving.

The key role of hydraulic mining for this site is transporting, with water, the broken coal from the mining face in the pillars to the well-protected fluming room. In the extraction of pillar and top coal, men and machinery would not be required to work beyond the narrow, well-supported rooms used for fluming.

Abutment pressure would be the primary means of breaking top coal and pillar coal on retreat.

Development and undercutting would be with narrow places, using continuous miners and water transportation; no shuttle cars or belt conveyors would be used. The narrow places would eliminate most of the need for artificial roof support, except for the fluming rooms, which must be maintained in a safe condition under the abutment pressures.

The high ratio of retreat coal, broken by induced caving, to coal mined on advance would result in high productivity and low material costs. High recovery and greater safety would also be achieved.

Unless otherwise specified, all references to tons throughout this report refer to short tons (2,000 lbs.).

TABLE 1
SUMMARY

Annual Production (Table 4)	1.154 million clean tons		
Working Schedule:	3 shifts per day, 5 days per week, 240 days per year		
Recoverable Reserves (Table 2)	25.658 million tons		
Recovery Estimated			
Mining	80%		
Preparation	86%		
Mine Life			
Initial Construction and Development	40 months		
Full Production	21 years		
Total Life	25 years		
Production Units			
Dev., dual rooms, hyd. haul.			
Deployed	2 continuous miners		
Producing each unit	3 unit shifts per day		
Dev., undercut rooms, hyd. haul.			
Deployed	1 continuous miner		
Producing	3 unit shifts per day		
Monitor retreat mining			
Deployed	3 feeder-breaker-monitors		
Producing	3 unit shifts per day		
Productivity in Mining Sections for average height of coal of 20 ft. (Table 4)	<u>No. of Workers</u>	<u>Tons Per Unit Shift</u>	<u>Tons Per Man Shift</u>
Retreat by monitor & moving back	6	1556	259
Arched rooms	5	63	12.6
Ventilation rooms	2	63	31.5
Undercut rooms	<u>2</u>	<u>181</u>	90.5
	15	1863	124.2 Avg.
Retreat:development ratio, room and pillar:	5:1		
Water:coal ratio for monitor	3:1		
Percent of total coal mined by monitor:	83.5%		
Percent of coal mined from arched headings, average:	3.4%		
Total mine employees (Table 5)	173		
Productivity, overall (Table 5)	29.1 clean tons per man day		

Financial Forecast

	Selling price per ton f.o.r.		
	<u>\$25</u>	<u>\$30</u>	<u>\$35</u>
Operating cost, including shared facilities (Table 7)	\$19.79	\$20.19	\$20.59
Capital cost, hydraulic mining only, including interest during construction and development loss carried forward (\$000) (Table 8)	\$26,508	\$25,628	\$24,748
Rate of return, hydraulic mine only (Table 10)	17%	30%	41%
Payback period after 3.3 years of construction, hydraulic mine only (Table 10)	4.6 yrs	2.2 yrs	1.4 yrs

Safety Features:

- All coal is cut under water jet or spray, allaying virtually all dust.
- All top coal and pillar coal is extracted remotely by high pressure, high volume jet, requiring no men or machinery to go into the immediate mining area.
- All work carried out in the abutment zone is by workers protected by yieldable aluminum arches, thus providing the ultimate in protection from roof and ribs.

GEOLOGY

The Book Cliffs coal field lies along the southern boundary of the Uintah Basin, which has been the most intensively mined field in the state of Utah. All the coal is contained in the Blackhawk Formation and is of Upper Cretaceous Age. The coal bearing formation ranges up to 1300 ft. thick in the region of this property. The rank of the coal is high volatile "B" bituminous.

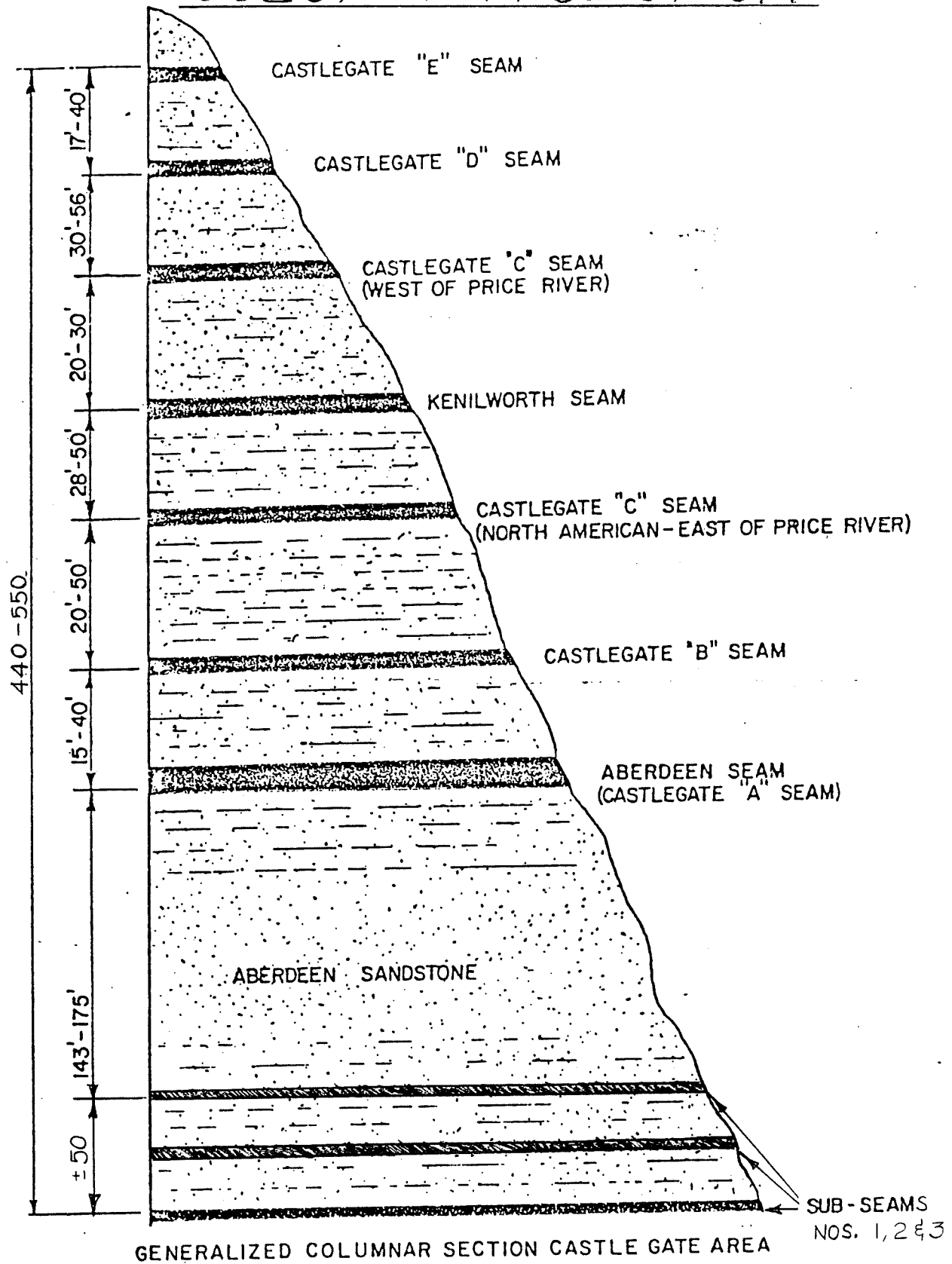
The structure is a broad monocline that dips approximately 7° to the north in the area of interest. The structure in this general area is undisturbed and unusually reliable. Mining has been done for 100 years in the vicinity, and there have been virtually no problems with faults or rolls. There is no evidence of igneous activity, no dikes or sills.

There are 10 seams of coal as shown on a generalized section in Figure 1. In the area of this study, three of the seams have mineable thicknesses, ranging from 5 to 26 ft.

The coal seams are separated and overlain by strong sandstones and shales. The thickest cliff-forming sandstones are the Castlegate, Aberdeen, and Starpoint. The Castlegate Sandstone is most prominent over the property and reaches a thickness of 500 ft. It forms massive cliffs on either side of the Price River where it cuts through the middle of the property; hence, the name Castlegate.

The Castlegate Sandstone overlies the Blackhawk formation, about 800 ft. above the A-seam. It is one of the thicker, more massive, and stronger rock formations to overlie a coal-bearing formation in North America. Above the Castlegate is the Price River formation, which contains several more sandstone beds, some as much as 100 ft. thick.

FIGURE 1 COLUMNAR SECTION



The topography has been influenced by these strong massive sandstones and the erosion of the Price River and tributaries, which cut through the middle of the property. The result is a precipitous terrain (see photograph in Figure 11) which causes the depth of overburden to vary widely in a short distance.

The thickness and strength of these massive sandstones affect the mining conditions.

MINING CONDITIONS

The conditions for mining the thick lenses of coal at Price River Coal are favorable for hydraulic mining, considering the:

- thickness of the seam;
- 7° gradient;
- strong roof and strong floor;
- self-supporting top coal roof in narrow places;
- severity of the abutment pressures on pillars sized to yield without violent bumping.

Seam Thickness

There are several major lenses of coal on the Price River Coal property which exceed 12 ft., the maximum height generally considered feasible for longwall mining. The thickest lens is in the A-seam on the west side, about 2500 ft. from the new Crandall Canyon shafts. Here, the thickest section drilled was 26 ft.

In order to have logical boundaries between the coal reserved for longwall mining, i.e., below 12 ft. thick, and the thick coal to be mined hydraulically, the actual boundary would follow a longwall panel layout and would generally fall between the 10 and 12 ft. isopach lines. In this study, hydraulic mining is planned to take all of the coal thicker than 10 to 12 ft. The average thickness is estimated to be 20 ft.

Seam Pitch

The pitch of the seam is very uniform, varying between 7 and 8 degrees. While a bit steeper pitch would be preferred for hydraulic mining, the pitch here is adequate. Fluming rooms can be driven up the full pitch, and pillars and top coal can be extracted on both sides. A crosscut, or a pillar extraction line, 55 degrees off of full pitch, would provide a 4 degree

fluming gradient, sufficient to bring out the broken coal, given an adequate flow of water.

The pitch of 7 to 8 degrees is gentle enough to permit efficient travel by rubber tired equipment for men and material. Also, the pitch is flat enough to eliminate caved rock sliding into the working place and causing dilution and/or blockage of the monitor.

Roof

The roof over the seam is sandstone, shale and siltstone, and is expected to be similar to the roof in the other seams with similar rock. It is believed that the rock will stand up long enough to recover at least 80% of the coal seam before caving in.

The experience with top coal roof in the A-seam at the Kenilworth Mine and the New Peerless was good. The top coal made a safe, competent roof with spans of 16 ft., and occasionally up to 30 ft. No artificial roof support was required. There are miles of entries in the old workings, with only top coal for support, which have stood without maintenance for up to 50 to 60 years. Many of these workings can be safely traveled to this day.

The undercut rooms, which are to be 11 ft. wide, can be driven safely under top coal without support.

Floor

The main floor is the Aberdeen Sandstone, a strong sandstone which ranges from 143 to 180 ft. thick. This member is very competent and should make an ideal rock in which to excavate the required openings for the underground dewatering and pumping station. Little support, if any, should be necessary for the spans required.

Between the top of the Aberdeen and the bottom of the A-seam, there is usually several ft. of hard shale, siltstone or bone. From experience, it is expected that this immediate floor will stand up long enough to extract the room before deteriorating under the effect of the water.

Faults, Rolls, Intrusions

No faults, rolls or intrusions are in evidence in the entire property, except at the west end, about 5 miles from the proposed site. None of the extensive old workings in the Kenilworth, Castlegate, Royal, Peerless and Spring Canyon Mines show any faulting or other geological disturbances.

Overburden

The overburden ranges from 1300 to 2700 ft., with only about 10% of the reserves over 2400 ft. While this depth is adverse for extraction of pillars in normal room and pillar mining, it provides an advantage for hydraulic mining.

Unlike normal pillar recovery, which requires men and machinery to work in the pillars being extracted, hydraulic mining sends only a jet of water into the pillar, no men or machines. The monitor-feeder-breaker unit is positioned in a narrow fluming room, supported well enough to withstand the abutment pressures and the small bumps from the yielding pillars on either side.

The advantage of the 1300 to 2700 ft. overburden in this mining method is that the pillars can be designed to a size that the abutment pressure in the zone of extraction will be sufficient to crush them, making recovery by the water jet very productive.

Coal Hardness

The Hardgrove index is reported to be in the range of 43-47. Uniaxial compressive strengths have been reported to range from 2600-4300 psi. One Protodyakonov test showed a value of 1.72. The coal is strong and massive, and, as explained earlier, when under normal overburden pressures, makes a very strong, self-supporting roof. A coal this strong is not readily amenable to cutting with a hydraulic jet. Therefore, application of hydraulic mining to the coal in this plan utilizes continuous miners to partially undercut the coal and abutment pressure to break the remaining coal pillars and top coal. The water jet would need to do little cutting.

Multiple Seams

There are at least three mineable seams in the area of interest. Above the A-seam, 190 to 220 ft., is the Castlegate "D". Presumably, this seam, ranging in thickness from 9 to 18.6 feet, will be mined ahead of the A-seam. Assuming large pillars are not left in the "D", there should be no significant adverse effects on the "A" seam.

If the "D" is mined by longwall, pillars could be completely eliminated by use of monolithic packwalls in place of coal pillars between panels. Such packwalls can easily be blasted out as the longwalls are retreated, thus eliminating any pillars which might cause pressure concentrations in the underlying seams.

Some 220 to 250 ft. below the A-seam is the "Sub 3" seam. Where the mineable areas of the "A" and "Sub-3" overlap, the "Sub 3" ranges in thickness from 5 to 8 ft. To avoid pressure concentrations in the "Sub 3", the "A" should be mined as completely as possible, leaving no uncrushed pillars.

Rock Splits

Of nine drill holes in the thick A-seam reserve, only one, "MC 5", shows a rock split of any consequence. Here, the seam is split by 3.1 ft. of rock with 8.7 inches of coal above, and 14.8 ft. of coal below. Drill hole "2" shows only a 6 inch layer of bone between two coal layers, a 12'6" upper and an 11'2" lower.

The 3.1 ft. layer of rock, cored in hole MC-5, contains about 400,000 tons of rock which constitutes about 1.4% of the total in-seam reserve of raw material. The 14% reject loss provided in the cleaning plant estimates should easily be enough to include this in-seam impurity.

Moreover, the recovery of rock from the broken pillar coal by the water jet will be less than that of coal. The rock, being harder and heavier than coal, will not be flushed out of the pillared area as readily as the coal. Rock breaks into larger pieces than coal and requires more water and steeper gradients to be moved by the flow of water.

Gas

All the seams on this property contain gas. Core samples have been tested, using the USBM method for determining the amount of gas contained in the coal. The data from these tests indicate that the A-seam coal in this area contains an average of about 200 cu. ft. per ton in place. The coal is quite impermeable and does not bleed off its gas readily. Only the first 8 to 10 ft. of rib bleeds off. Also, since the solid coal gives up its gas slowly, much of the gas will be carried out with the coarse fraction of the raw coal mined. Experience has shown that 30,000 cfm is ample to ventilate a continuous miner working a 5 entry system. Also, experience has shown that 100,000 cfm on a longwall panel producing 4,000 tons per day is adequate, with

50,000 coursing the face and 50,000 going to the bleeders through the old entries along the panel gob.

No. 5 mine, when producing 6,000 tons per day, liberated 1,800,000 cu. ft. of gas per day, equivalent to 300 cu. ft. per ton.

Spontaneous Combustion

Where bleeder entries were used requiring return air to pass over the gob, heatings and fires have occurred occasionally in the "Sub 3" and Kenilworth seams. The A-seam coal is chemically and physically very similar. Therefore, it is planned that the air will not travel through the gob. Bleeder entries will be provided, as explained under "Ventilation," to draw off the return air from the gob near the working faces.

Bumps and Outbursts

Because of the thick and strong overburden, there will be unusually high abutment pressures. The mining plan must be designed to cope with these pressures without disastrous effects. This can be done by:

- (1) Use of small yielding pillars which cannot store enough energy to cause serious violence during failure. These pillars will be designed to stand under normal strata load, and to gently crush and yield under abutment loads.
- (2) Use of yielding arches (aluminum) in the active fluming rooms which must be supported well enough to withstand the abutment pressures and then yield when the adjacent pillars yield.
- (3) More complete recovery of pillars than in a normal room and pillar system, thus more complete caving.
- (4) Open end pillar extraction by the water jet.
- (5) Use of small cross-section fluming roads which are inherently easier to support.

The phenomena of abutment pressure will be used to advantage in this plan. Actual experience at Price River Coal has demonstrated, as it has at other deep mines in Utah, that thin pillars yield or fail without serious violence. Also, as they yield, they have less tendency to punch holes in or shear the roof. Their yielding permits relatively gentle convergence of the roof and floor. Such pillars do not blow up as they fail.

COAL RESERVES

The reserves planned herein for hydraulic mining are centered on the extra thick coal lens in the A-seam in the vicinity of the abandoned New Peerless Mine. While the final boundaries would be made compatible with the projected longwall panels surrounding this block of coal, the reserves for this plan have been calculated two ways:

- (1) Including all coal within the 12 ft. isopach line, to the south, east and west, and to the north boundary (see Figure 2). Partial extraction (50%) is assumed for the area 1200 ft. wide under Price River Canyon, which serves as a corridor not only for the river, but also for the State Highway and the main line of the D&RGW Railroad. Thus, significant subsidence of this area will be avoided.
- (2) Same as above, but including sufficient Federal coal to the north to permit more efficient recovery of the thick seam, and less frequent moves of the dewatering and pumping station.

The north boundary of the reserve in (1) above is the present lease line. It is close to the strike of the coal, and is about 16,000 ft. long. In order to collect the hydraulically flumed coal, without use of excessive water, a gradient of 3 degrees or more is needed. To mine all of this coal hydraulically and avoid a series of closely spaced dewatering stations, it is planned that additional land, approximately one-half mile to the north, would be leased in the thickest part of the lens. This would permit approximately 25 million tons of clean coal to be produced from the first dewatering station site.

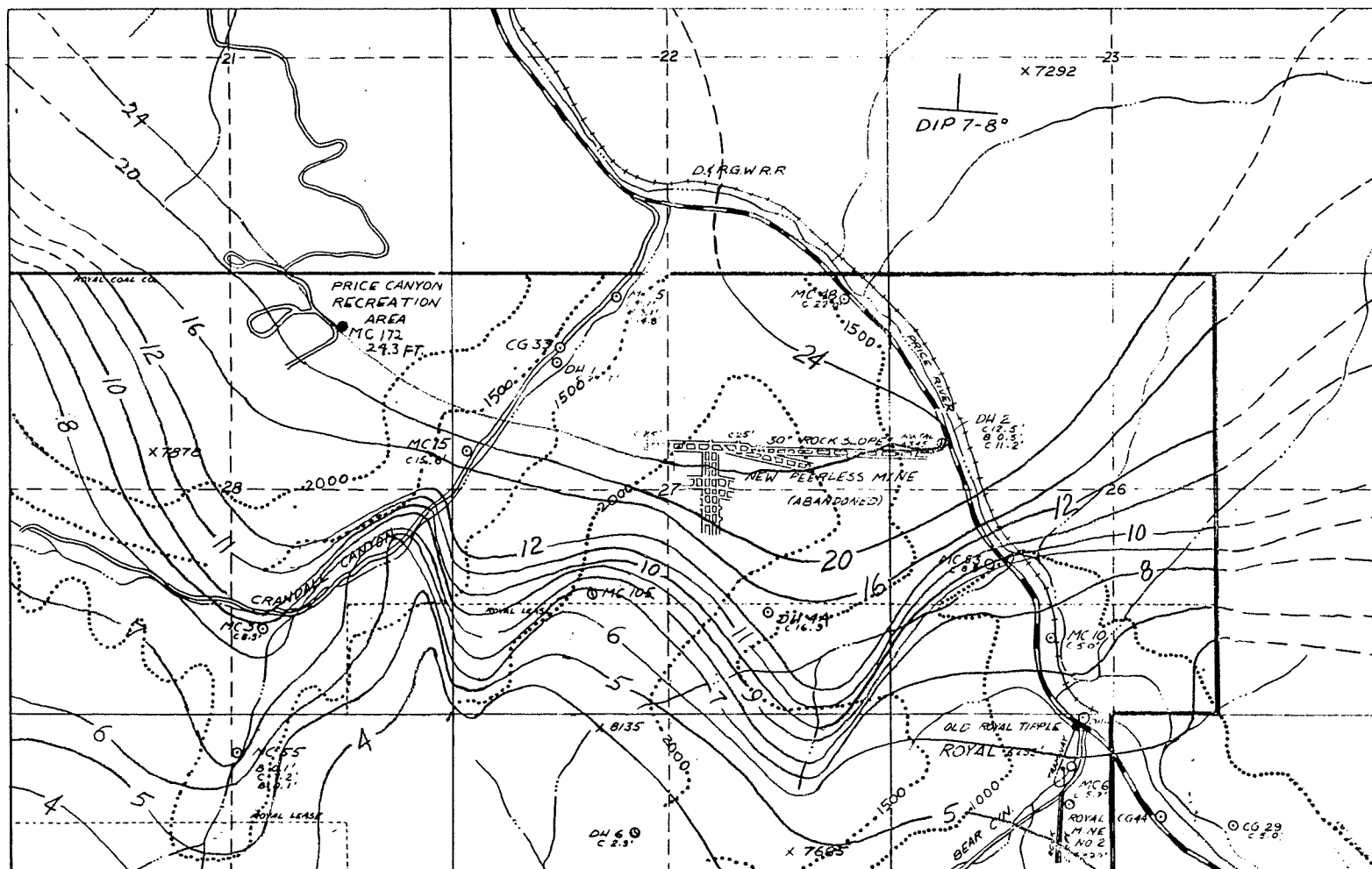
The Federal Government should logically be receptive to an application to lease out more land for the purpose of employing a mining system that would increase the recovery of thick deep coal reserves to a higher level than any experienced heretofore.

TABLE 2

Hydraulic Mining Reserves, A-Seam

<u>Coal Thickness</u>		<u>Tons in Place (000s)</u>		
<u>Range</u>	<u>Average</u>	<u>Present Lease</u>	<u>Additional Federal Lease</u>	<u>Total</u>
12-16	14'	5,912	654	6,566
16-20	18'	5,052	6,904	11,956
20-24	22'	13,041	422	13,463
24-26	24'	6,266	-	6,266
	20.18'	30,271	7,980	38,251
Less coal sterilized under Price River		1,848	1,550	3,398
Net reserves in-seam		28,423	6,430	34,853
Assume mining recovery 80%		22,738	5,144	27,882
Dilution from out-of-seam material, adds 7%		24,330	5,504	29,834
Clean coal recoverable (prep. plant 86%)		20,924*	4,734	25,658
Production of clean coal per year		1,154	1,154	1,154
Life of reserve in A-seam in years		18.1	4.1	22.2

*Of this amount, about 8,000,000 is the increment over 12 ft. thick which, with longwall or other types of mining, would be considered as not recoverable.



— SECTION LINES
 - - - - - QTR. SECTION LINES
 ——— PROPERTY BOUNDARY
 - - - - - LEASE BOUNDARY
 ——— ISOPACHS
 - - - - - OVERBURDEN ISOPACHS

——— PRINCIPAL HIGHWAY
 ——— SECONDARY HIGHWAY
 ——— WATERWAY
 + + + + + RAILROAD
 ○ DRILL HOLE
 ○ TRAMWAY HOIST

FIGURE 2
CASTLE GATE
A-SEAM



SCALE: 1"=2000'

Coal Quality

The analysis of the core from drill hole MC53 is typical of the A-seam. This hole is just off the southeast edge of the block planned for hydraulic mining. The raw coal analysis of A-seam is shown in comparison with typical analyses of other mineable seams that occur on the property.

<u>Seam</u>	<u>Hole No.</u>	<u>Moisture</u>	<u>Ash</u>	<u>Volatile Matter</u>	<u>Fixed Carbon</u>	<u>Btu</u>	<u>Sulfur</u>	<u>MAF Btu</u>
A	MC53	1.64	7.59	42.48	48.29	13,387	0.43	14,748
D	MC16	4.26	5.66	42.32	47.76	13,173	0.59	14,508
Sub-3	MC41	2.15	10.35	42.09	45.41	12,739	0.45	14,559
K	MC47	1.96	7.59	42.40	48.05	13,085	0.50	14,467

In recent years only the "Sub-3" and "D" seams have been mined. A representative test of the quality of this raw coal as mined has been:

<u>Moisture</u>	<u>Ash</u>	<u>Sulfur</u>	<u>Btu</u>
6.0	18.2	0.5	11,000

A washability study of rom coal from a mix of these two seams shows a recovery of 91.3% at 1.60 sp. gr. with a float coal ash of 8.16%. At the separation gravity of 1.60 the near gravity material is less than 4%. Actual quality of washed coal shipped in 1980 was:

<u>Moisture</u>	<u>Ash</u>	<u>Sulfur</u>	<u>Btu a.r.</u>
7.08	8.77	0.51	12,363

An in-depth feasibility study of the property has used an 86% yield from all seams feeding to the preparation plant for the west side. A review of the 16 drill holes which are in the hydraulic mining reserve or influence the assessment thereof, shows that rock splits will be encountered near to the periphery of the reserve area. For example, visually identified in-seam impurities in drill hole MC 172 constitute about 11% of the total in-seam material mined; likewise, the core of MC-5 shows about 17% reject from in-seam material. The yield of 86% for the A-seam reserves as envisaged in this report, would produce a 7-1/2% ash, after allowing for a raw coal dilution factor of 14% from interbedded rock splits and out-of-seam impurities, and washer efficiency of 98%.

MINE PLAN

The plan for hydraulically mining the 12 to 25 ft. thick A-seam coal will be integrated and sequenced into Price River's overall plans for the property. It is assumed that mining in the overlying D-seam will be carried out far enough ahead of the A to avoid undermining virgin coal in the D and, thereby, inviting roof problems there.

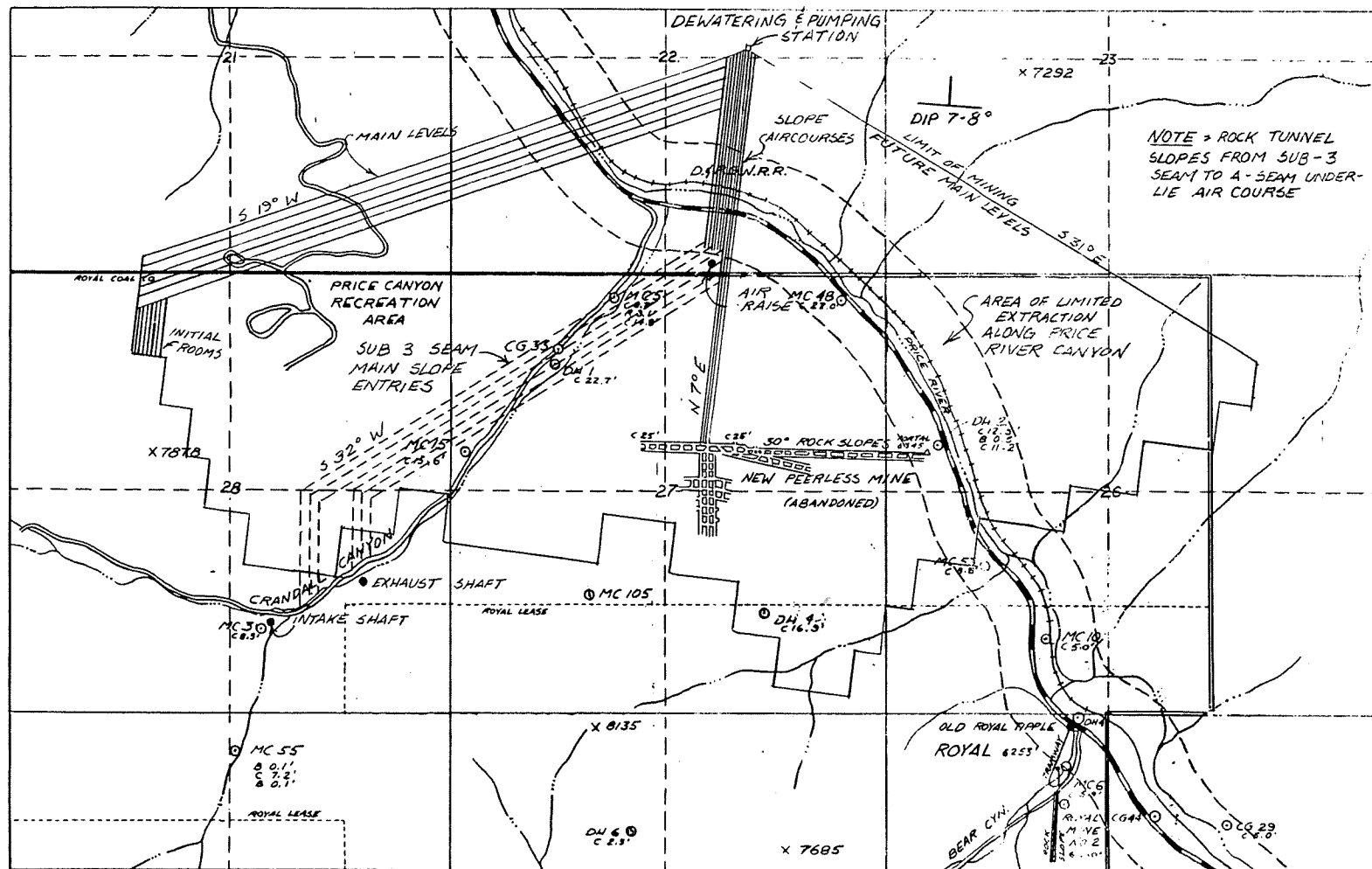
Mine Development

Location of the Dewatering and Pumping Station

This facility will be constructed in the Aberdeen sandstone on the down dip side of the thickest part of the coal lens in Federal coal land not yet leased (see Figure 3). Locating here will permit the recovery of over 25 million tons of clean coal, which, if mined at the rate of 1.154 million tons per year, would last 22 years. This location, reserve, and period of time have been used for the purpose of estimating the feasibility of this system.

Main Slopes and Rock Tunnels

See Figure 3. The approach to the A-seam reserve is planned via the Sub-3 seam, approximately 230 ft. below the A-seam. Entries will be driven from the bottom of the projected Sub-3 slopes, approximately along the 5 ft. isopach. They will run 6,000 ft. about N 55° E on a decline gradient of about 9%, and 2500 ft. N 30° E through rock on a near level gradient. This route, under about 2,000 ft. of overburden, partly in low coal, 4-1/2 to 5 ft., and partly in rock, will avoid sterilizing reserves in either seam that are more favorable to mining. Six entries will be driven, each with a cross-sectional area of 90 sq. ft.



Since the distance in coal is 6,000 ft., the amount of coal thus mined will be approximately 205,000 raw tons (176,000 clean tons). The out-of-pocket loss on this development drivage is assumed to vary from zero to \$10 per clean ton, depending on the selling price. The total loss will then range from zero to \$1,760,000.

The time to drive the 9% slopes is estimated as follows:

Assume the continuous miner used in this work produces 175 tons per shift.

205,000 tons ÷ 175 tons/unit shift = 1171 unit shifts.

2 continuous miners working 3 shifts per day would perform this leg of the development in 195 days.

Total time required, assuming a 240 work day year: 9.7 months; use 10 months.

During the driving of these entries, one of them will carry the belt conveyor, three intake air, and two return air.

After completion of all other development work, including connections to the New Peerless slopes, one of the 6,000 ft. entries in the Sub-3 will be used to carry the belt conveyor; the other five will serve as intakes, one of which will be brushed to provide a high speed roadway for delivery of men and materials.

The roadway will be constructed concurrently with the 6,000 ft. drivage in the Sub-3 coal seam. Brushing the roadway to a depth of 4-1/2 ft. will cost an estimated \$2.00 per cubic foot, or \$473,000. The roadway surfacing of compacted, crushed limestone or slag, is estimated at \$20 per ft. for 8500 ft., for a total of \$170,000.

Approximately 2,500 ft. from the site of the dewatering station, where the coal in Sub-3 seam becomes too thin to proceed further,

two rock tunnels, about 2500 ft. long, will be driven (see Figure 3). One of them will be inclined at 1% to intersect the A-seam at the top of the dewatering station site; the other will decline at 2.6% to intersect the bottom of the dewatering station. Thus, access will be provided to both the top and bottom levels of the station. The lower tunnel will carry the conveyor. The rock tunnels are planned to be 9 x 18 ft. in cross-section, and are estimated to cost \$400 per ft., for a total of \$2,000,000. (Tunnels are currently reported to be going for as low as \$310 per ft. in this coal field.) The rock tunnels are estimated to progress at 15 ft. per day and require 167 days of driving, plus 1 month for mobilization, for a total of 8.3 months; use 9 months.

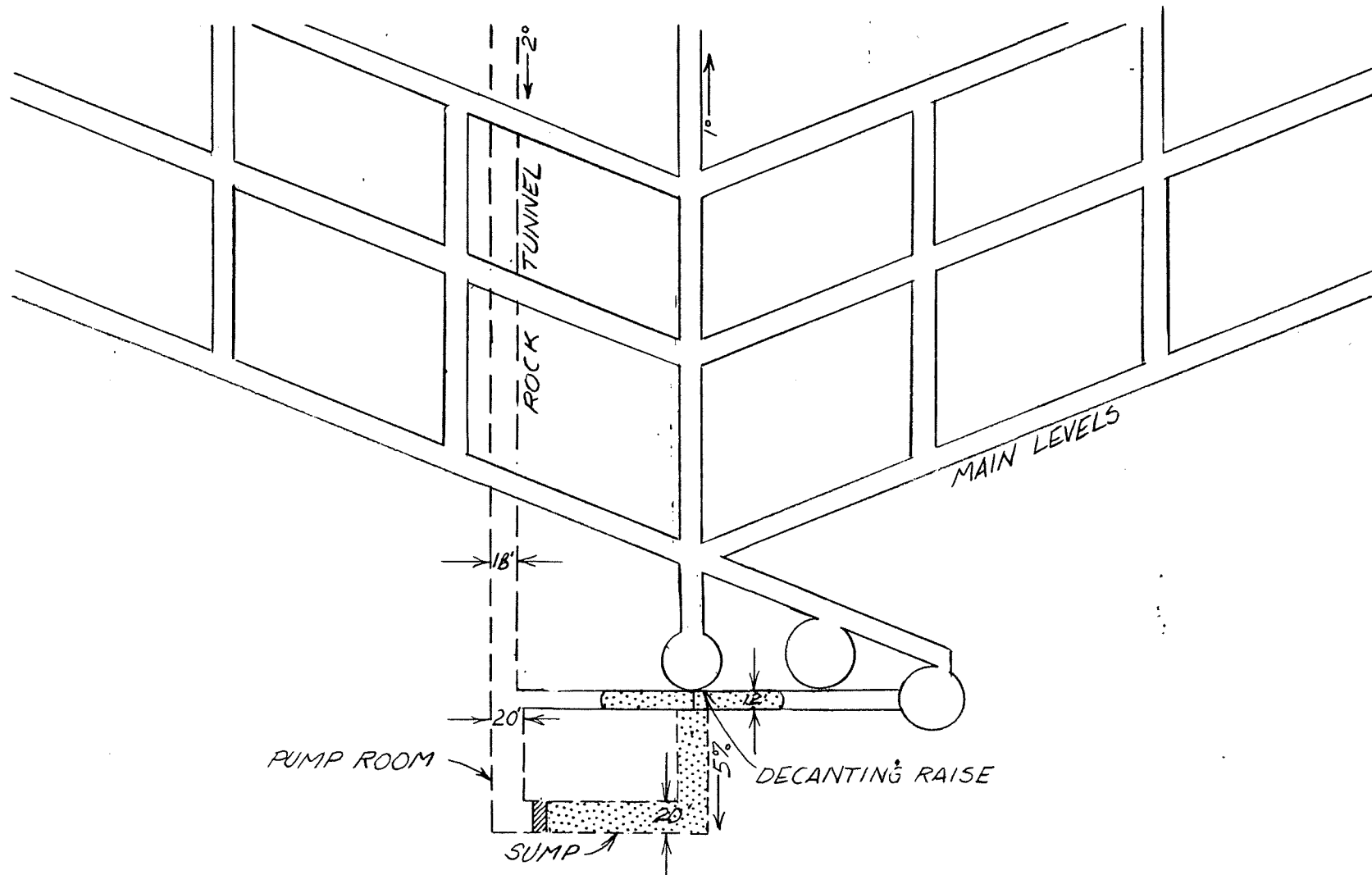
A 230 ft. intake air shaft, 16 ft. in diameter, will be raised from the Sub-3 entries connecting to the A-seam approximately 2500 ft. directly up pitch from the dewatering station (see Figure 3). This shaft will connect with 6 intake slopes in the A-seam which conduct the air down to the dewatering station and the main levels. The cost of the shaft is estimated at \$116,000 (approximately \$500 per ft., or \$67 per cu. yd.)

Dewatering and Pumping Station

This facility, constructed 2,000 ft. below the surface in the Aberdeen Sandstone, will contain the equipment for pumping fluming water to the faces of all development headings for pumping and high pressure water to the monitors mining the crushed pillars and broken top coal (see Figures 4 and 5). Also housed in the station are the screens, centrifuges, thickener and filters for dewatering the coal so that it can be conveyed out of the mine. The thickener and flocculating system will clarify the water so that it can be recycled through the high pressure pumps without intolerable wear.

FIGURE 4

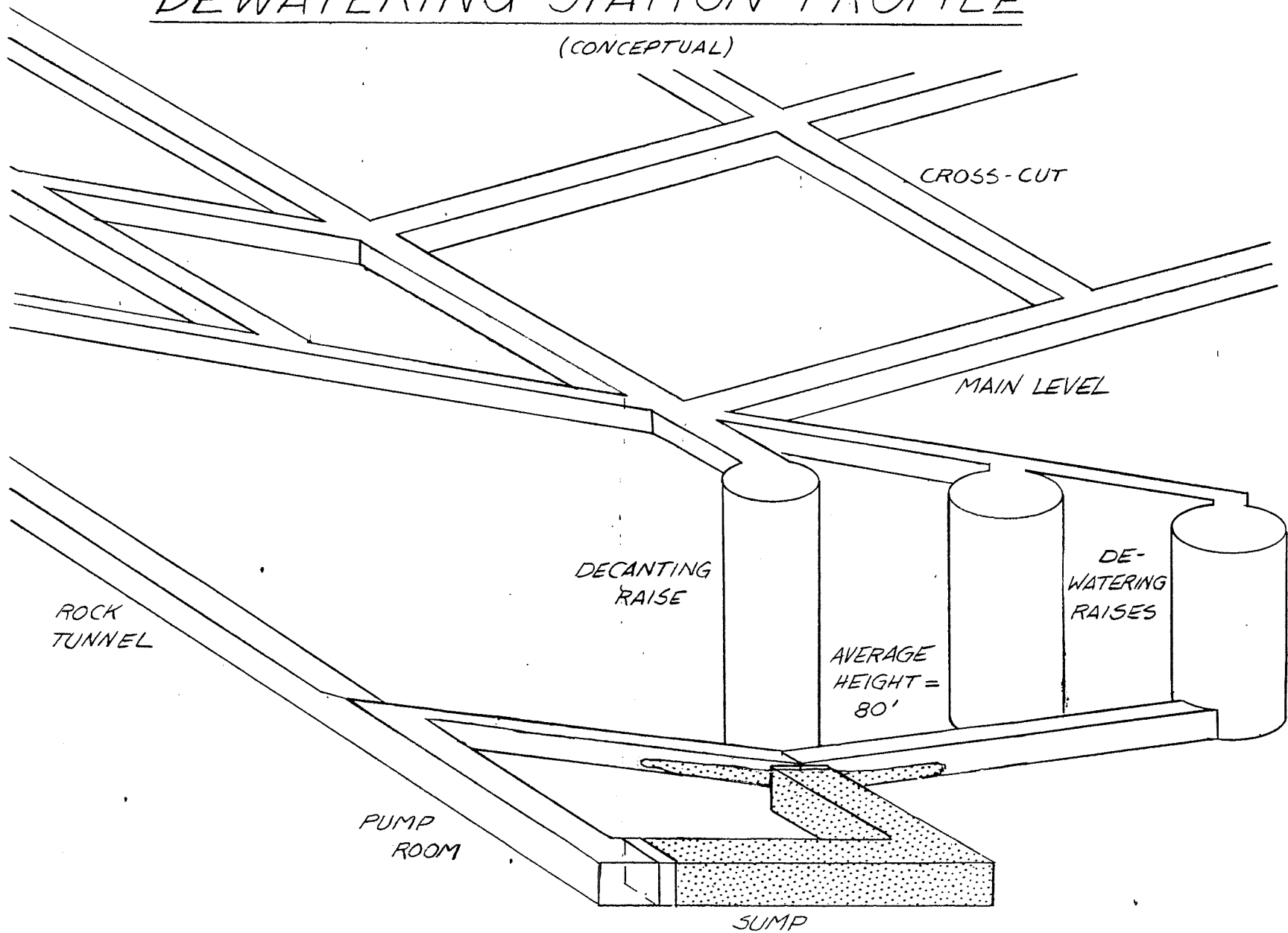
DEWATERING STATION PLAN



SCALE: 1" = 100' (approx.)

FIGURE 5 DEWATERING STATION PROFILE

(CONCEPTUAL)



The processing equipment will all be installed in 2 raises in the Aberdeen sandstone just below the A-seam. They will approximate 80 ft. in height and 40 ft. in diameter at the widest horizon where the thickener is housed. A third raise, 90 ft. high and 40 ft. in diameter, will provide an emergency decanting sump in case of power failure or breakdown of a key item of equipment in the dewatering facility.

A 150 ft. extension of the lower rock tunnel, with a 15 x 20 ft. cross-section, will provide a 300,000 gallon sump and positive head to the pumps (see Figures 4 and 5).

The construction of this facility will require an estimated 10 months.

	<u>Months</u>
Mobilization	1
Raises: (2 x 80 ft. + 90 ft.) + 5 ft./shift	
+ 40 shifts per month =	1.25
Rock tunnel sump: 150 ft. + 7-1/2 ft./shift	
+ 40 shifts per month	0.5
Construction	6
(The 230 ft. air shaft will be bored concurrently.)	
Contingency	<u>1</u>
	9.75
Use	10

Upon completion of the dewatering station, development of the air courses to the bottom of the New Peerless rock slopes will commence. Development of the main levels will begin at the same time.

Return Air Courses to New Peerless Rock Slopes

After the rock tunnels have reached the A-seam and the dewatering station has been completed, 10 entries will be driven up the full

pitch to the air shaft. Six of these will be intakes and will extend 2500 ft. from the dewatering station to the 230 ft. intake shaft to the Sub-3 seam. The other 4 will be extended 2,000 ft. further to connect with the bottom of the two existing New Peerless rock slopes. These slopes are 2500 ft. long and lead to the surface. They will be used for return air courses.

The 4500 ft. returns and 2500 ft. intakes will be 7 ft. x 11 ft. in cross-section, on 37.5 ft. centers, and will be driven under self-supporting top coal. The coal will be cut with a continuous miner and flumed to the dewatering station. Since the production from these development headings will be very efficient, the coal should be sold at a profit. Therefore, there should be no development cost or "loss carried forward" on this production.

This total drivage will produce 134,000 tons of raw coal (115,000 tons of clean), and will be done concurrently with the main levels over a 9 to 10 month period. Work will be on 3 shifts per day, requiring about 235 raw tons per shift. Since no shuttle cars or roof bolts are required, the productivity per man shift will be high, and not more than a 3-man crew is estimated to be needed.

To avoid sterilizing some of the thickest, most attractive reserves in a later phase of the mine, the four return air courses will be replaced with either a return air shaft or a new connection to the "Crandall Canyon 20 ft. shaft." The four air courses will then be retreated as normal rooms in the manner explained under "Retreat Mining."

Main Level Development

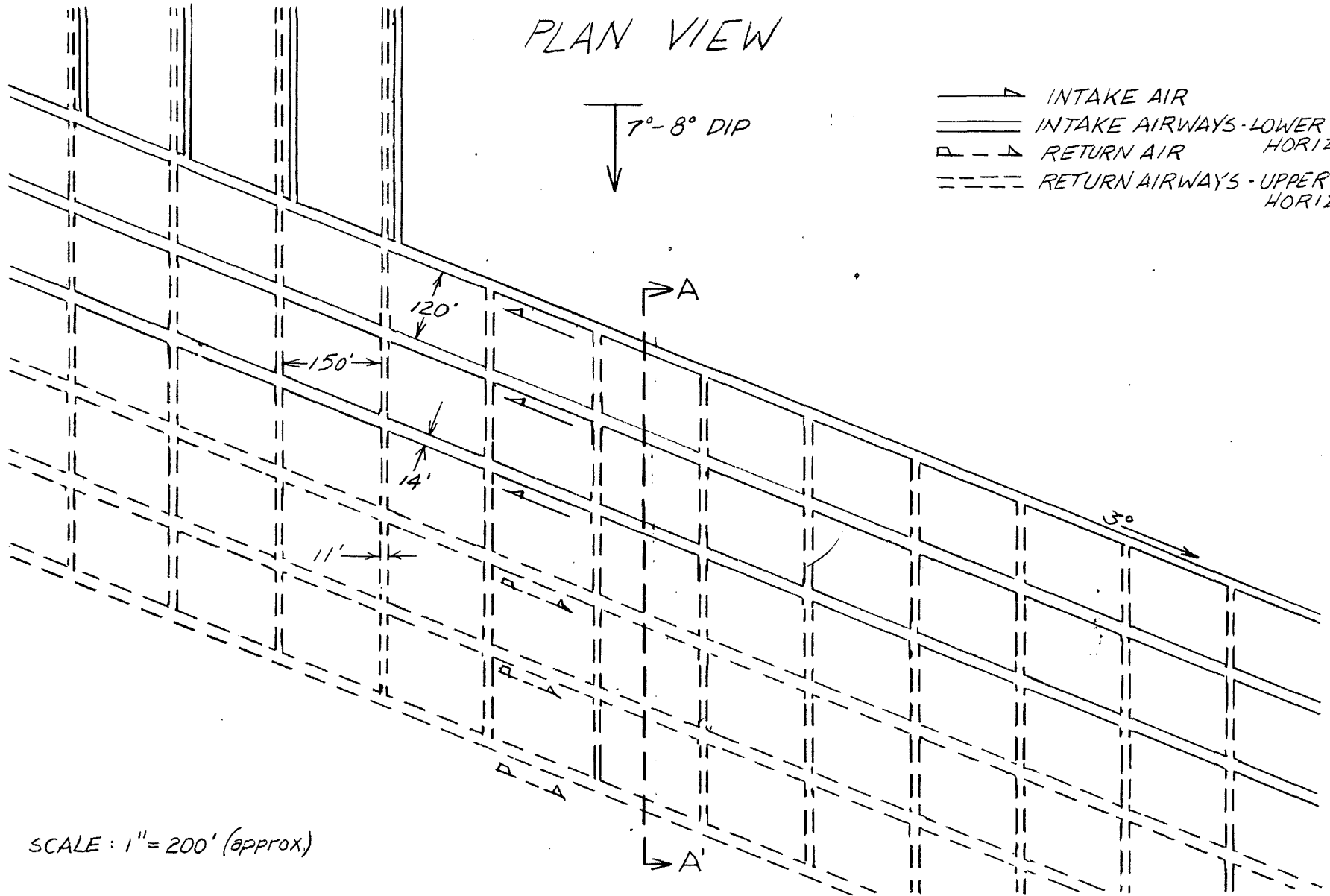
From the dewatering station, main levels to the southwest and to the southeast will be driven on a 3 degree minimum fluming gradient (see Figures 3, 6 and 7). Six entries will ultimately be driven in each direction from the station, three intakes and three returns.

FIGURE 6 MAIN LEVEL DEVELOPMENT

PLAN VIEW

7°-8° DIP
↓

- INTAKE AIR
- ==== INTAKE AIRWAYS - LOWER SEAM HORIZON
- ▢ - ▢ RETURN AIR
- RETURN AIRWAYS - UPPER SEAM HORIZON

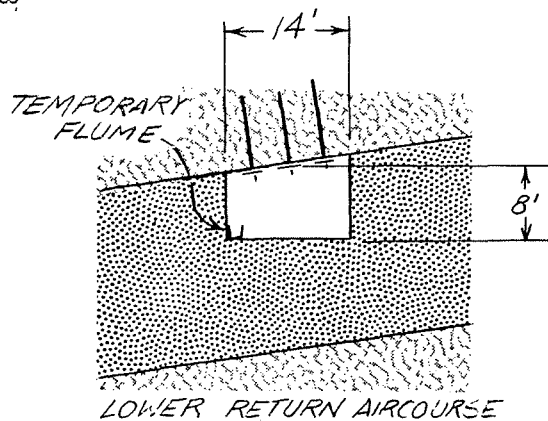
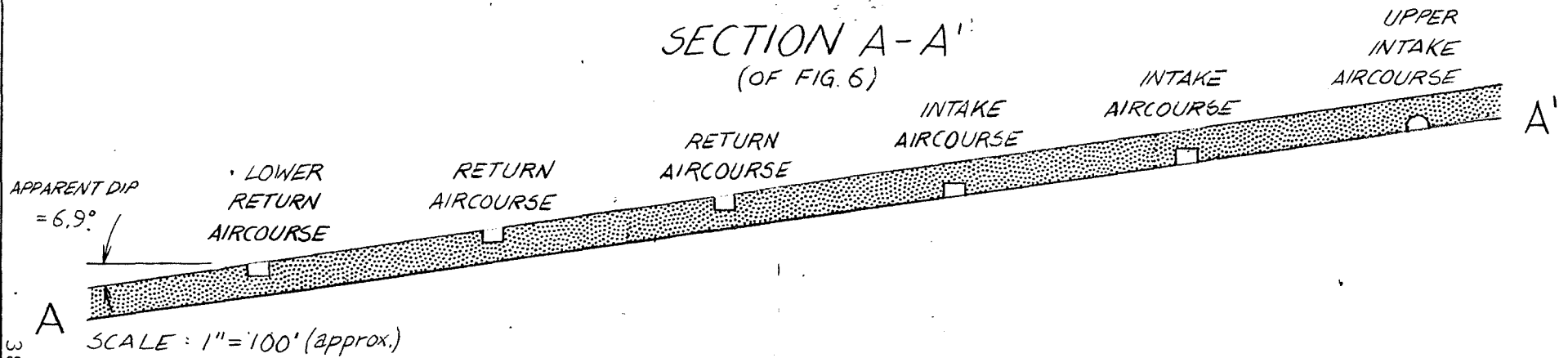


SCALE: 1" = 200' (approx)

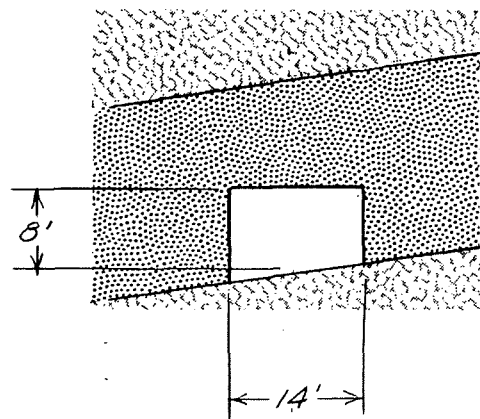
FIGURE 7

MAIN LEVEL DEVELOPMENT PROFILE

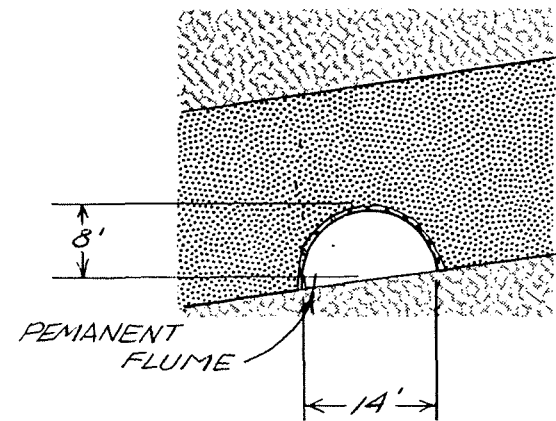
SECTION A-A' (OF FIG. 6)



SCALE: 1" = 20' (approx.)



INTAKE AIRCOURSE



UPPER INTAKE AIRCOURSE

The main levels will be on 120 ft. centers with a cross-section of 8 ft. x 14 ft. Crosscuts will be on 150 ft. centers. With these dimensions, 22% of the area in plan view is extracted. This is considerably less than the 34% area extraction ratio used in the Kenilworth Seam about 6 miles to the east of the proposed site for this study. The old Kenilworth workings are still standing after more than 10 years with no maintenance. (See photographs in Figure 11.)

The intakes will be on the up-pitch side of the main level corridor and will be driven on bottom with top coal for support during the initial drivage. The uppermost intake, from which fluming rooms are turned, will be arched after the initial drivage and before incurring abutment pressures. A flume, constructed of high molecular weight polyethylene (HMWPE) with a very low coefficient of friction, will be installed on the floor.

The three main levels used for return air will be against the roof, 8 ft. x 14 ft. in cross-section, and on 120 ft. centers. The return farthest down the pitch will contain a temporary flume of HMWPE.

In the thicker parts of the reserve, no artificial overcasts are necessary. The great thickness of the coal permits the return air courses to be driven over the top of the intakes with a separation or bridge of coal between the two. In the thinner parts of the reserve, overcasts can be constructed, but without excavation of rock. The intervening coal, if any, will have to be supported from the lower place.

The main level layout is designed so that the pillars and top coal can be economically recovered on the retreat. This will be accomplished by undermining the pillars with narrow places on 120 ft. centers driven before the abutment zone gets too close and adds to the normal strata pressure. These places will be driven with a boom type continuous miner (roadheader) and

FIGURE 8

ROOM DEVELOPMENT

PLAN VIEW

DIP 7°-8°

UNDERCUT ROOMS
(6½' X 11')

UNDERCUT ROOM
CROSSCUTS (6½' X 7')

VENTILATION ROOMS DRIVEN
AGAINST THE ROOF
(8' X 11')

FLUMING ROOMS
(8' X 11')

CROSSCUTS
(6' X 7')

NOTE = SOLID LINES
SHOW WORKINGS IN
LOWER HORIZON SEAM.
DOTTED LINES SHOW
ROOM IN UPPER HORIZON
SUPERIMPOSED OVER
AND OFFSET FROM
A' LOWER FLUMING ROOM.

SECTION A-A'

lighter or heavier cover, and based on actual underground experience, the cross-section dimensions and spacing can be varied to optimize the monitor productivity with the cost of development, roof support and ventilation.

Fluming rooms will be developed in the bottom horizon, up the full pitch of 7 to 8 degrees, on 150 ft. centers. From each of these rooms, coal will be extracted, on full retreat, 75 ft. on either side. These rooms will be driven with an arched cross-section, 8 ft. x 11 ft., by a boom type of continuous miner.

While the top coal over these rooms should be self-supporting under normal strata pressures (see Figure 11), it will be supported by arches. They are estimated to be needed on 3 ft. centers for complete roof and rib protection from the abutment pressure that will occur near the cave line.

It will not be necessary to set the arches at the face ahead of the mining machine as is the case in most arched headings. Because the top coal is self-supporting before abutment pressures are imposed, the arching can be carried on close behind the boom header, or as far back as is convenient for scheduling the work.

Aluminum arches, while initially more expensive than steel, are only about one-half the weight for equal strength, and therefore can be carried, erected and retrieved with about half of the manpower. These more expensive arches are justified where arch recovery and re-use is high, as is planned herein. In the Lujiatuo hydraulic mine in China, under similar cover, the recovery of arches has been developed to a fine art; they report recovering and reusing 99.9% of the arches. For the purpose of this estimate, 90% recovery and reuse is assumed.

A flume 24" wide and 12" deep will be cut in the floor in the center of the place to contain a low friction, long wearing,

HMWPE flume liner. By keeping the flume depressed in the middle of the floor, the roadway can more easily be kept clean and smooth for vehicular travel. Small bridges are easy to construct at the intersections with the main level flume road.

An example of the boom type of mining machine which would be suitable for driving the arched fluming roads is the Paurat E-195, which, in the standard form, has a height of 6.2 ft., a width of 8.1 ft., a cutting head power of 134 hp, and a weight of approximately 35 tons. Inasmuch as hydraulic transport of the coal eliminates the need for a gathering head, conveyor, and rear boom, the machine for this application would be narrowed down to between 6 and 7 ft. Cutting reach in height is 14.6 ft., and in width, without moving the machine off center line, 16 ft. (See Figure 9 for a conceptual outline of the machine adapted for hydraulic mining). Other models are available from Dosco, Anderson-Mavor, Voest-Alpine, AEC, Eickhoff, Westphalia, etc.

Water to the heading will be served by two 6" diameter Schedule 80 pipes. For ventilation of each arched fluming room and the area being extracted by it, a parallel room, approximately 8 ft. x 11 ft., will be driven against the roof above the fluming room and, depending on the thickness of the coal, offset from 15 to 20 ft. from the bottom room. These rooms will be driven by a continuous miner, such as a Joy 12-CM (see Figure 10), but equipped for hydraulic haulage. This modified continuous miner would consist of only a set of crawlers, chassis, cutter head, and roof bolter. No gathering head or conveyor is necessary, because the coal is cut under a flow of water and flumed on the floor to the first open crosscut, where it drops down to the lower fluming room.

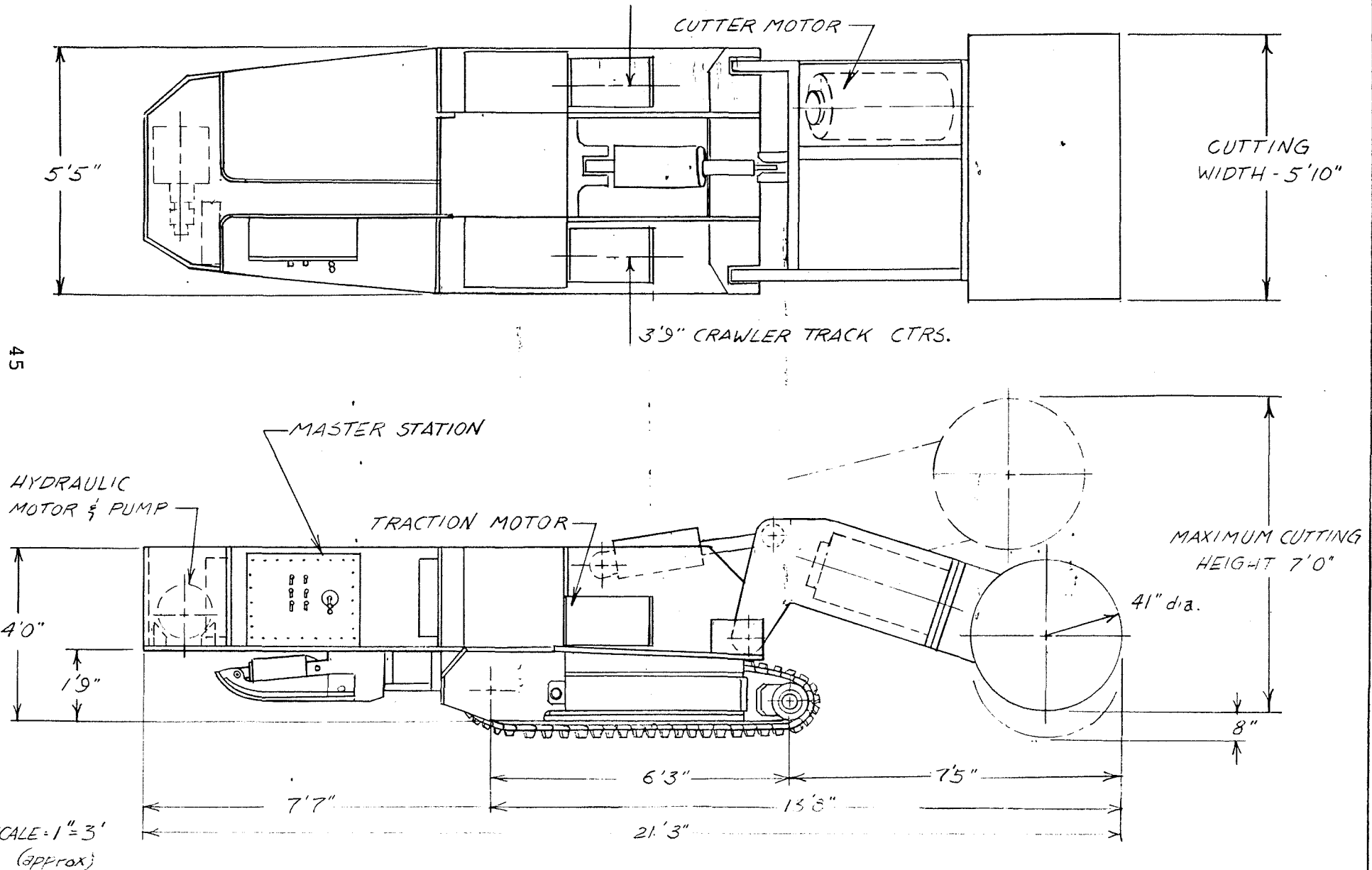
Steep, short crosscuts of small cross-section, 6 ft. x 7 ft., on 60 ft. centers will be driven partly by the boom header in the lower place, and partly by the continuous miner in the upper place. The closely spaced crosscuts will minimize the length of

MODIFIED FOR HYDRAULIC HAULAGE



FIGURE 10

CONCEPTUAL OUTLINE OF DRUM HEAD CONTINUOUS MINER



brattice lines, and the small cross-section will minimize the cost of stoppings.

The upper level rooms will provide excellent bleeder ventilation from the top of the gobs at the cave line where the pillars and top coal are recovered. The arched fluming room with the overlying, offset return air room will hereafter be referred to as a "dual-room" (the combined width of extraction is 22 ft.).

To start full retreat production with the monitor, 3 dual-rooms for fluming need to be developed and 6 undercut rooms, 12 places in total. The length of these initial rooms is estimated to be 600 ft., corresponding to the length of a longwall face in the adjacent workings.

The advance rate of the dual-rooms, and undercut rooms can be controlled by the size of the crew assigned. The development work will be spread over three shifts per day, the same schedule as the monitor, in order to provide a flow of coal and water as uniform as possible to the dewatering station. Thus, the development advance rate and production will be keyed to the retreat rate of the monitor. See Table 3, page 51.

If the operation of the two mining machines is separated from the installation of arches, the advance rate will depend mainly on the rate at which the ventilation room can be roof bolted. Experience in mines using side mounted bolters on continuous miners is typically 100 ft. per shift, equivalent to 50 bolts with a 2-bolt pattern on 4 ft. centers, or 75 bolts on a 3 bolt pattern.

Each lineal foot of arched room produces 3.5 tons of raw coal; likewise, each lineal foot of upper return air room produces 3.5 tons.

Under average conditions the monitor retreats 18 ft. per shift, and, in turn, requires this much development work. It is estimated this can be accomplished by a 5-man crew in the arched heading, and 2 men in the ventilation room.

The productivity for this work is estimated at: 18 ft. x 2 places x 3.5 tons/ft. = 126 tons per shift. Raw coal per man shift is: $126 \div 7 = 18$. For the arched room productivity is $63 \div 5 = 12.6$; and for the ventilation room $63 \div 2 = 31$.

The time to develop the initial dual-rooms required to begin retreat mining by monitor is estimated at approximately 10 days.

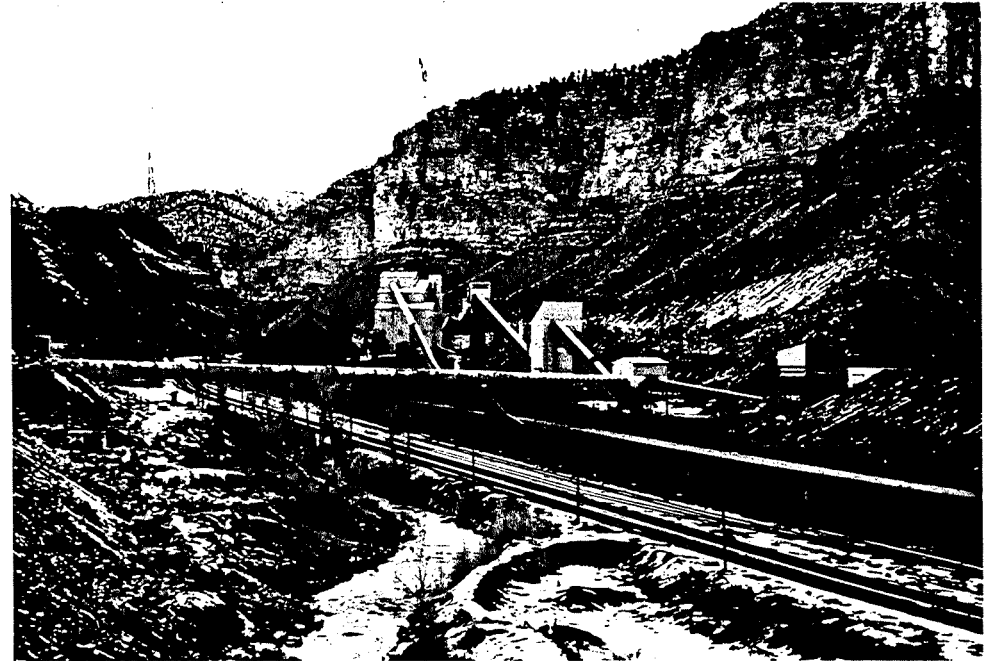
Between fluming rooms, in the bottom horizon, is a 139 ft. wide pillar which will be partially undercut by 3 rooms, having a cross-section of 6-1/2 ft. high by 11 ft. wide. "Undercut rooms" will have 6-1/2 ft. by 7 ft. wide crosscuts on 80 ft. centers. Thus, under typical conditions, the spacing of rooms will be 37.5 ft. and the intervening pillars, 26.5 ft.

The three "undercut rooms" are driven under self-supporting top coal for the purpose of reducing the pillars to a size that will crush under the abutment pressure, but will support the top coal roof under normal strata pressure. (See Figure 11, which shows some old Kenilworth mine openings standing securely under self-supporting top coal after 50 to 60 years without maintenance.) These dimensions are considered to be applicable to the typical condition of the planned area, i.e., 20 ft. of coal thickness and 2,000 ft. of cover. They are put forward as the best judgment based on long experience of working in similar conditions of overburden depth, coal strength, roof strength and overburden strata strength.

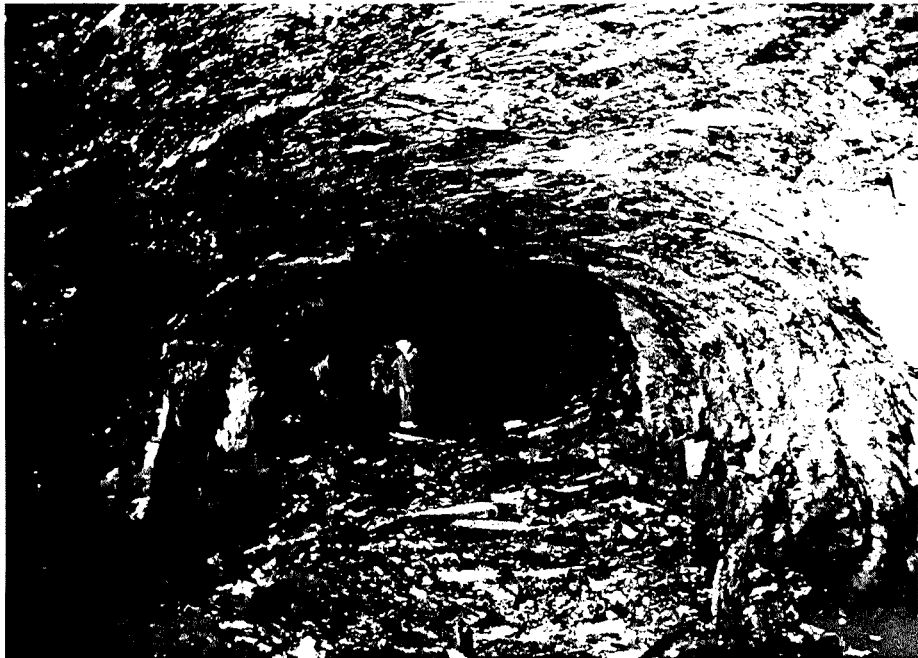
Each set of three undercut rooms between the fluming rooms, including crosscuts, produces 10 tons per ft. These rooms will be driven by a continuous miner under top coal with virtually no impediments to continuous production other than extending pipe,

FIGURE 11

PHOTOGRAPHS OF SITE AND OLD WORKINGS



ABOVE: Surface facilities at Price River Coal Co. showing typical terrain.



LEFT: Old workings at Kenilworth Mine illustrating the strength of the top coal in unsupported headings after standing 60 years without maintenance.

power line and brattice. Coal will be cut at an estimated average rate of 3 tons per minute, although much higher peak rates could be attained if more fluming water were supplied to the face. Production is conservatively estimated at about 180 tons per unit shift using a 2-man crew. This rate requires about 60 minutes of cutting time, leaving 320 minutes for hanging brattice, extending pipe and cable, cleaning up, rock dusting, building stoppings, etc. If a row of safety props were required, there would be ample time to set them. However, past experience would indicate that they will not be needed. Productivity is estimated at 90 tons per man.

When the coal is thinner and more undercut development is needed, the crew will be increased to provide more time for cutting (see Table 3).

In order to get the initial pillars crushed sufficiently to make for easy mining, the upper end of the initial undercut rooms will be widened out and the pillars thinned down to induce caving. Actual field experience will quickly determine the amount of widening needed.

The total area extracted on room development on the bottom horizon will be 34%, the same as that in the bottom of the Kenilworth Mine, where only "first mining" was done under 1300 to 2200 ft. of cover. There, as mentioned before, many of the older workings (20 to 50 years old) are supported with top coal only; some more recent workings, 10 to 15 years old, are roof bolted. Moreover, the Kenilworth workings are about 10 ft. high and 16 to 20 ft. wide. For the most part they are standing very well today after more than 10 years with no maintenance.

Development Ratios

In thick seams and deep cover where the fluming rooms require arches, as is believed to be the case here, the cost of development for such rooms is a major factor in the economic feasibility. The selection of room cross-section and room spacing, as shown in Figure 8, attempts to minimize the ratio of development coal from arched headings to low cost coal from the monitor. Compared to retreating one side only, as is necessary in steep seams, retreating both sides of the monitor room effectively doubles the amount of monitor coal per foot of arched heading. As mentioned earlier, the overburden will vary from 1300 to 2700 ft. and the coal thickness from 12 to 25 ft.; therefore, the dimensions should be adjusted to match the conditions. A more detailed report would comprehend the variations in conditions and match the dimensions to suit them, area by area, and year by year.

Table 3 shows for various seam thicknesses the proportion of coal mined by the several types of work. The percentage of coal mined from arched headings calculated on the table ranges from 2.7 to 5.3% and averages 3.4%. The coal mined outside of the arched room, whether by continuous miner or by monitor, is comparatively low in cost. This means that 28 tons of low cost coal are enjoyed for each ton of high cost coal from arched headings. These figures will be used in estimating crew sizes and costs.

In Table 3 the figures shown in column 1 for an average seam thickness of 20 ft. are used for cost estimating purposes. Columns 2 and 3, showing minimum and maximum thicknesses, are displayed for information only.

TABLE 3
DEVELOPMENT RATIOS AND TONNAGE
FOR VARIOUS SEAM THICKNESSES

	Average ⁽¹⁾	Minimum ⁽²⁾	Maximum ⁽³⁾
Seam thickness in feet	20	13	25
Raw tons per ft. of room:			
In-seam, in-place	120	78	150
Recovered, 80%	96	62	120
Recovered and diluted (x 1.07)	103	66	128
Development tonnage per foot			
Arched monitor rooms	3.5	3.5	3.5
Ventilation rooms	3.5	3.5	3.5
Undercut rooms, set of 3	<u>10</u>	<u>10</u>	<u>10</u>
Total	17	17	17
Recovered by monitor/ft. of room	86	49	111
Percent of each type of development mining to total:			
arched rooms	3.4	5.3	2.7
ventilation rooms	3.4	5.3	2.7
undercut rooms	<u>9.7</u>	<u>15.2</u>	<u>7.8</u>
all rooms	16.5	25.8	13.2
Percent mined by monitor	83.5	74.2	86.8

(1) Average thickness of reserves, used for cost purposes.

(2) Minimum thickness for any sustained period of several months, shown for reference only.

(3) Maximum thickness shown for reference only.

Retreat Mining

Perhaps the greatest advantage of hydraulic mining is the use of a high pressure jet of water to reach out 75 to 100 ft. or more to extract top coal and pillars without the necessity of men or machines entering the area from which pillars are being extracted. This is particularly advantageous in thick seams where adequate roof support is impractical. To visualize the difficulties and dangers of sending men and machines into such high places as was practiced in former times see the picture taken at Castlegate years ago, Figure 12.

By contrast, extraction by hydraulic jet uses only a stream of water to bring out the coal, see Figure 23. Thus, safety, recovery and productivity are greatly enhanced.

Abutment Pressure

A major key to this system is the use of abutment pressure in a controlled way to crush the coal so that it is easily flushed out of the mining area where the pillar and top coal are to be recovered.

All mines experience abutment pressures. They have been studied, measured, tested, analyzed, depicted, and written about voluminously by many prominent authorities on rock mechanics. The degree to which abutment pressures increase over normal strata pressures are depicted for various locations by various authors in Figures 13 through 22.

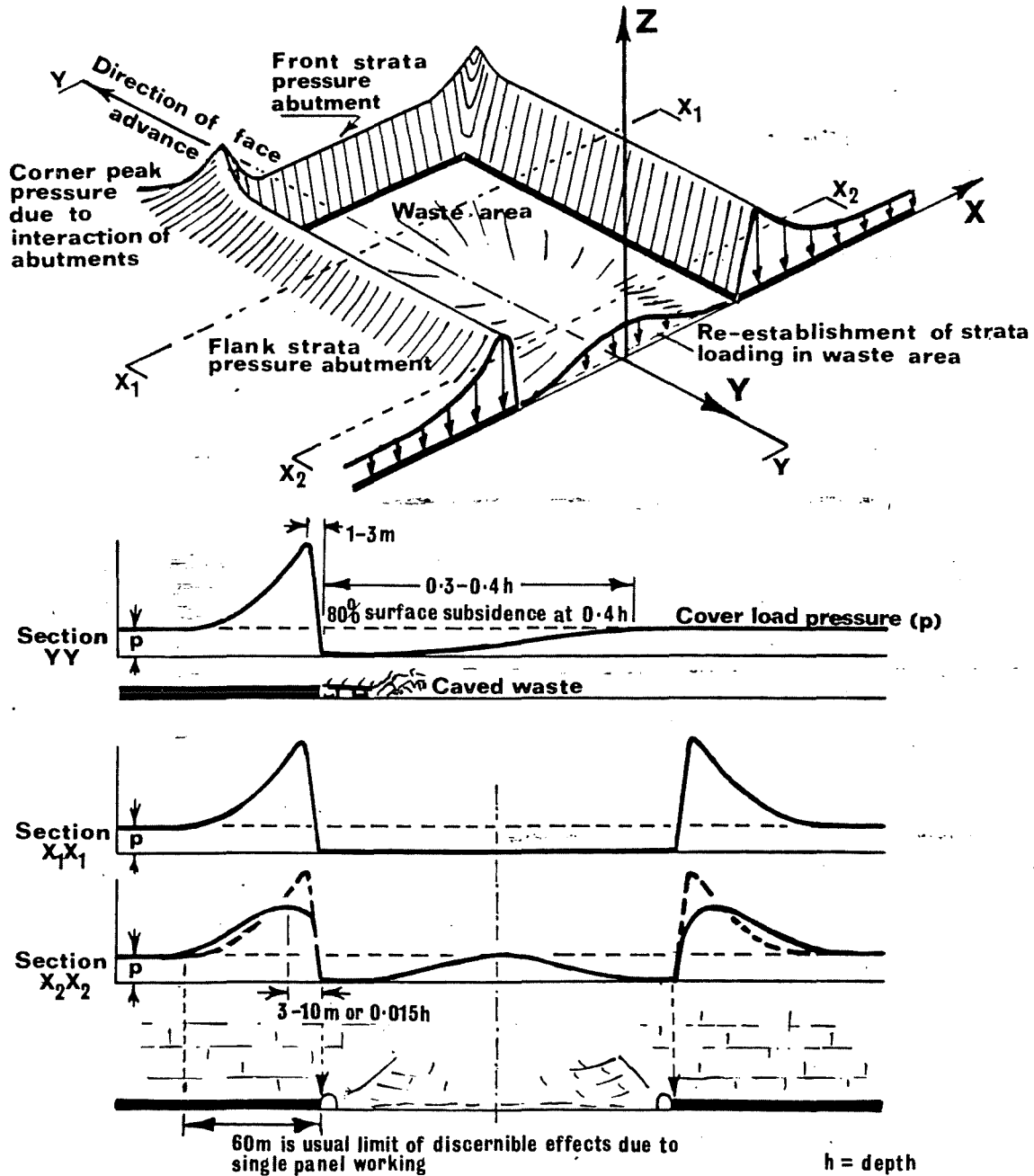
FIGURE 12

PHOTOGRAPHS TAKEN AT CASTLEGATE
OF OLD METHODS FOR TAKING
TOP AND PILLAR COAL IN THICK SEAMS



Figure 13

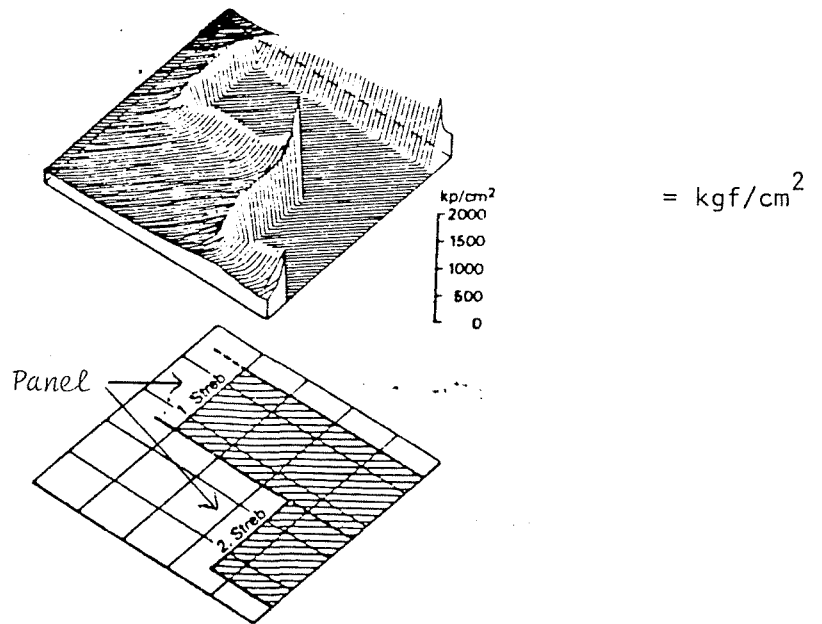
Abutment Pressure Around a Longwall Panel



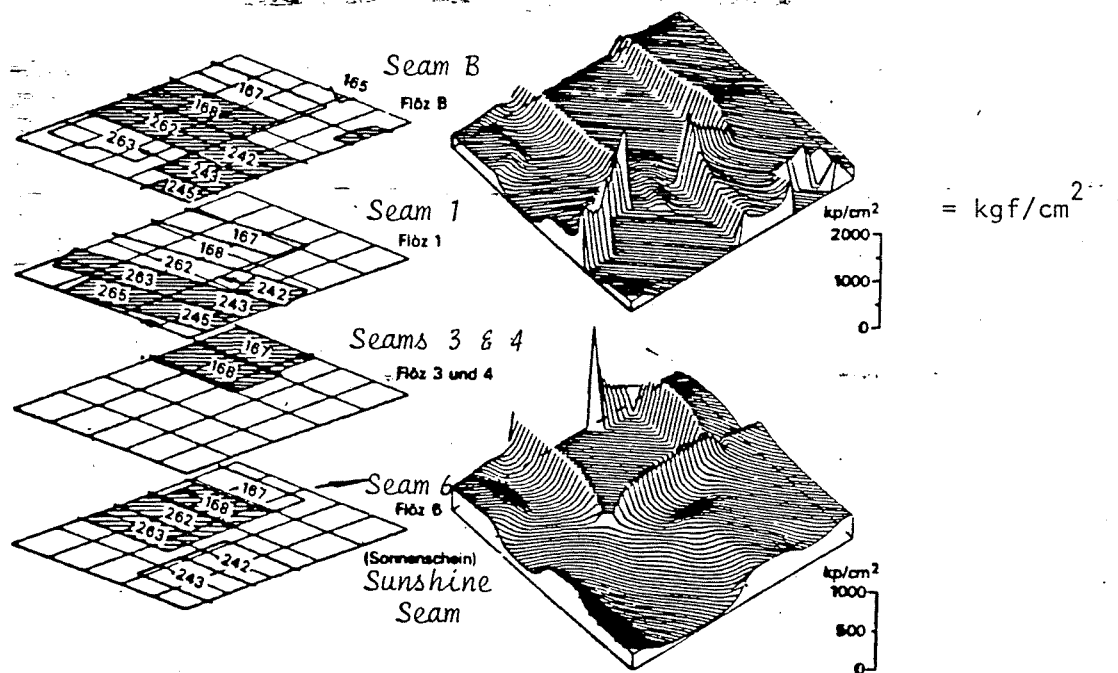
Strata pressure redistribution in the plane of the seam around a longwall face (Information based on British research findings)

Source: B.N. Whittaker, The Mining Engineer, October, 1974.

Figure 14



Typical case of the pressure distribution illustrated by use of a plotter.



Plotted picture of pressure distribution near the face (measuring point No. 5) on January 1, 1976 in the Westfalen Coal Mine.

SOURCE: Oskar Jacobi, "Practical Strata Control;" and Everling, G., Determination of Rock Pressures Surrounding a Longwall Panel, GLUCKAUF 109 (1973), pp. 1131 - 1133.

Figure 15
Impact Stresses in Coal Beds

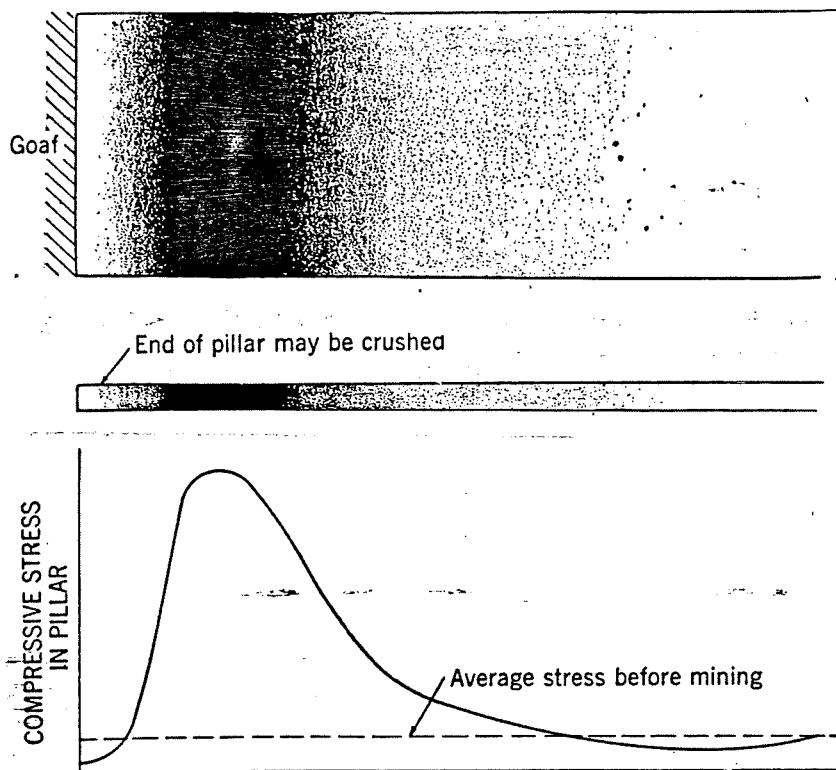


FIGURE 5.—Stress Concentrations in Pillars Paralleling Extensive Mined-Out Areas Under Strong Roof and on Strong Floor.

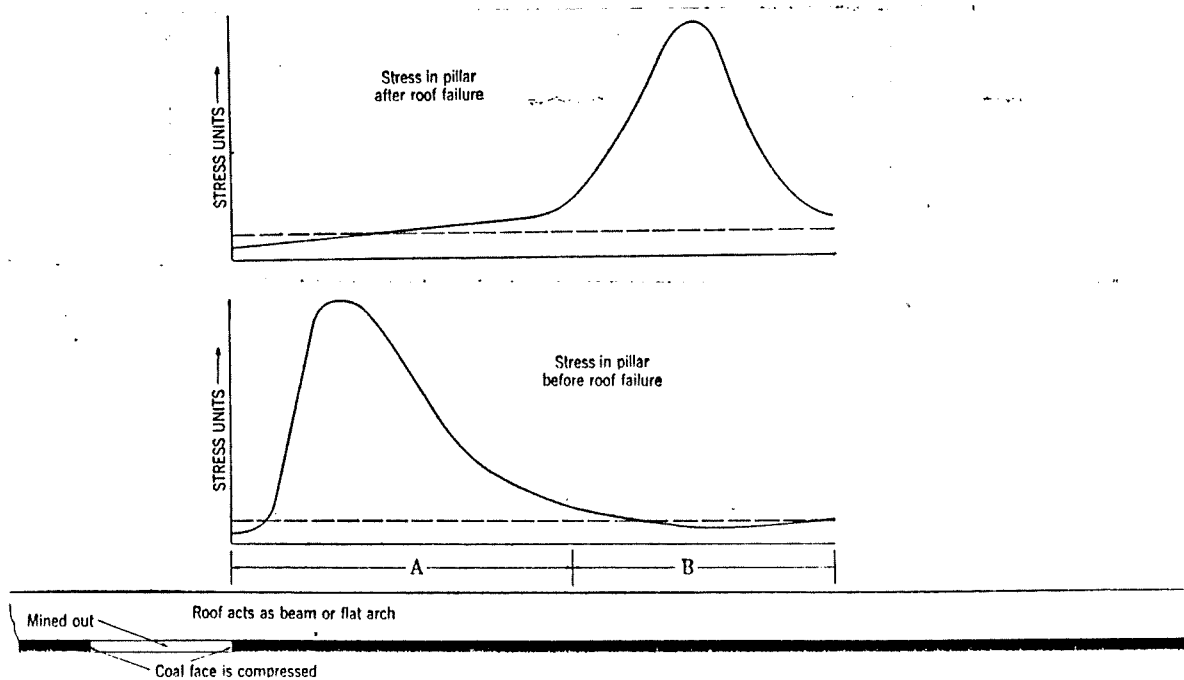
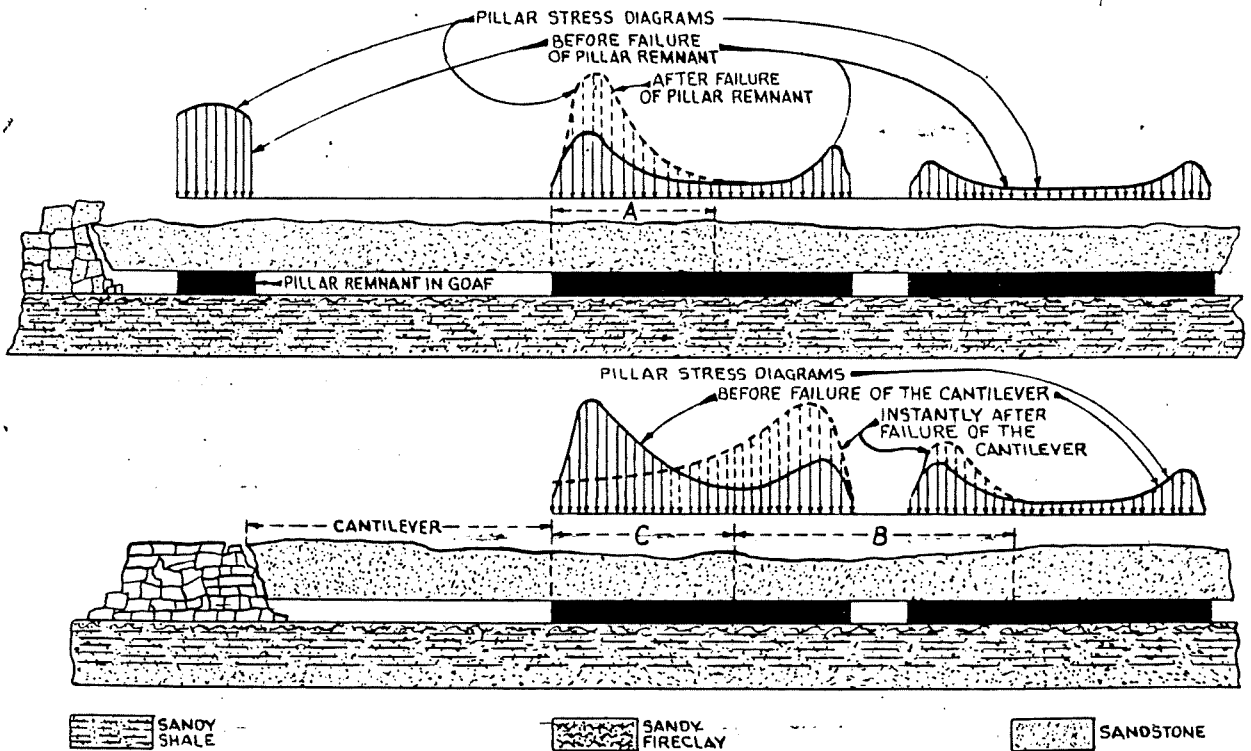


FIGURE 14.—Impact Stresses in Coal Beds Due to Energy Released in Roof and Floor Rocks by Failure of Beam or Flat Arch.

Source: C.T. Holland and E. Thomas, Bureau of Mines Bulletin 535.

Figure 16

Impact Stress in Coal Beds

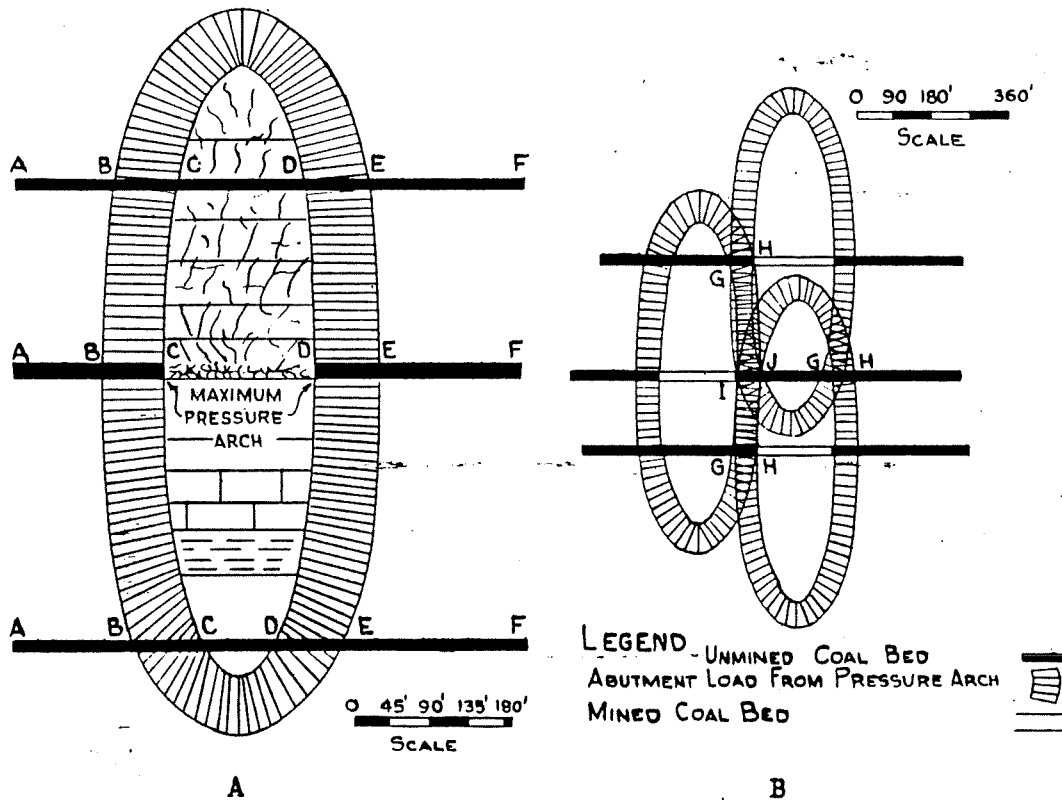


Impact stress in coal beds. If the pillar remnant fails suddenly, part or all the load it supported is instantly transferred to the pillar at A and may produce a high impact stress which might cause a shock bump at A. As the span of the cantilever increases due to mining the stress at C also increases. If the span becomes long, a pressure bump may occur at C. If the cantilever fails suddenly, the stress at C is decreased, but a high impact stress may be caused in the area B and could result in a shock bump in this area.

Source: C.T. Holland. The Cause and Occurrence of Coal Mine Bumps, 1958.

Figure 17

Abutment Areas Adjacent to the Maximum Pressure Arch



Abutment areas adjacent to the maximum pressure arch. Part A shows the general properties of the arch. DE is the high stress area in the abutment. Note it is acting in all beds across which the arch extends. CD is a low pressure area, and in EF the stress in the coal bed is only a little greater than that which prevailed before the middle seam was mined. In Part B is illustrated how abutment loads from areas mined in overlying and underlying beds may be superimposed. At IJ, three abutment loads act, resulting in a very high stress in the rocks. At GH two abutment loads are superimposed. Many other possible combinations exist.

Source: C. T. Holland, AIME Annual Meeting, 1958.

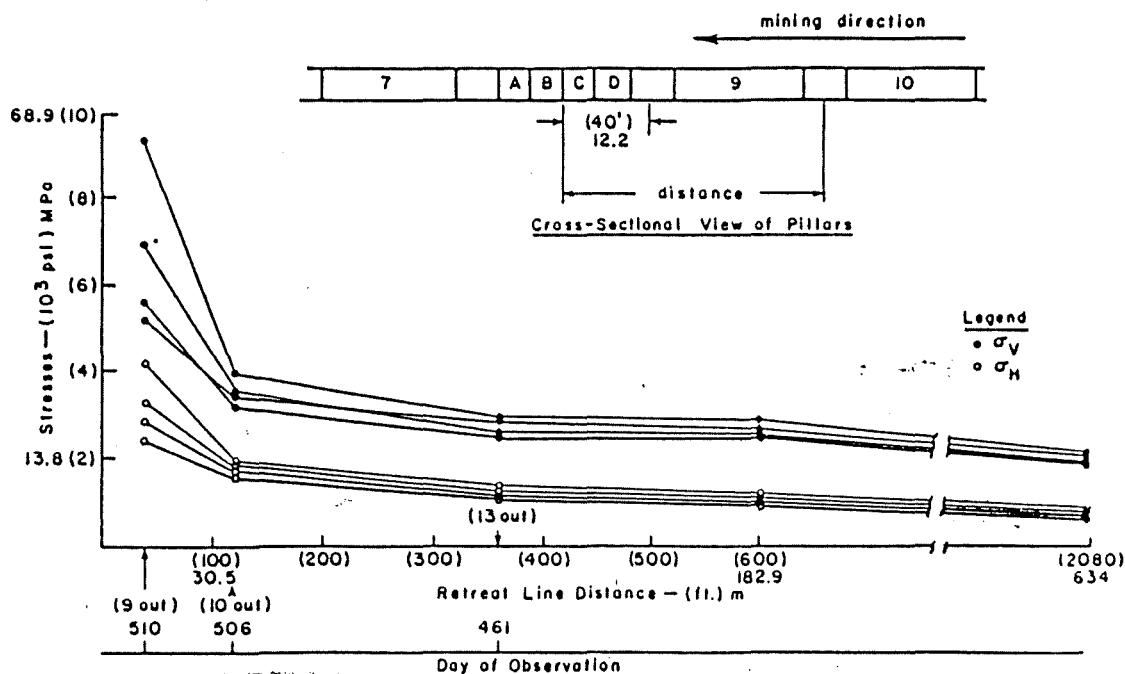
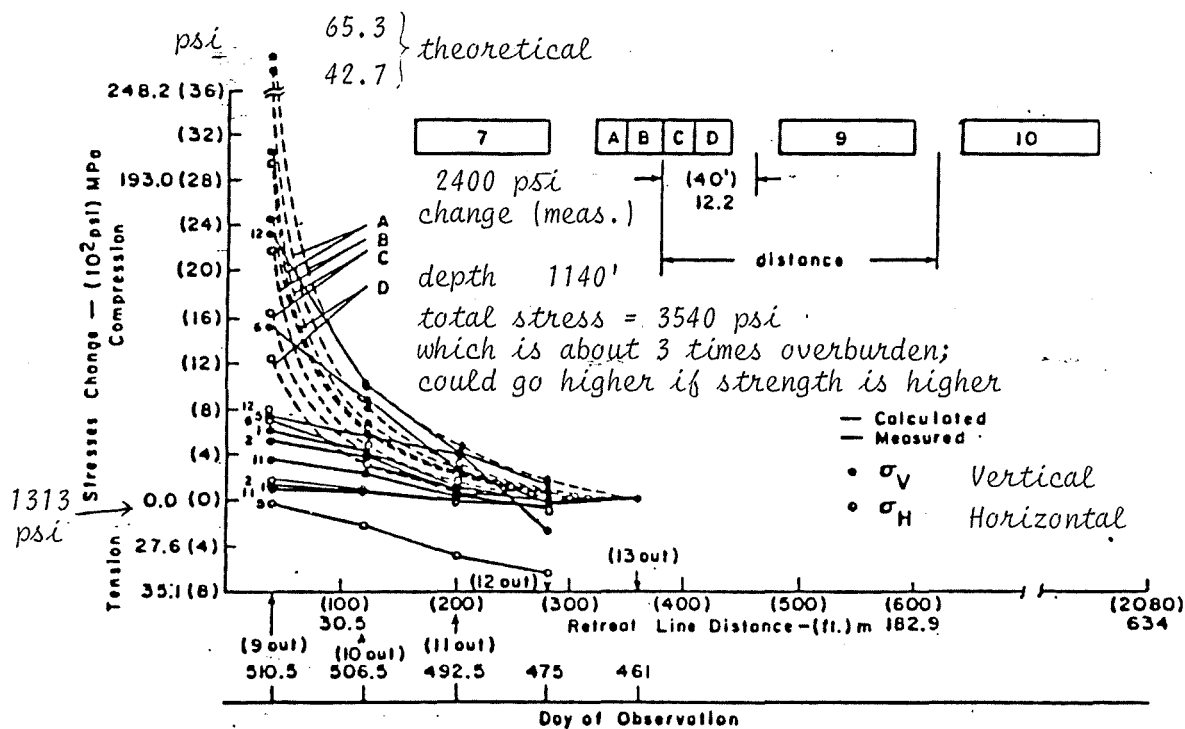


Figure 18. Calculated Stress at the End of Stage Mining.



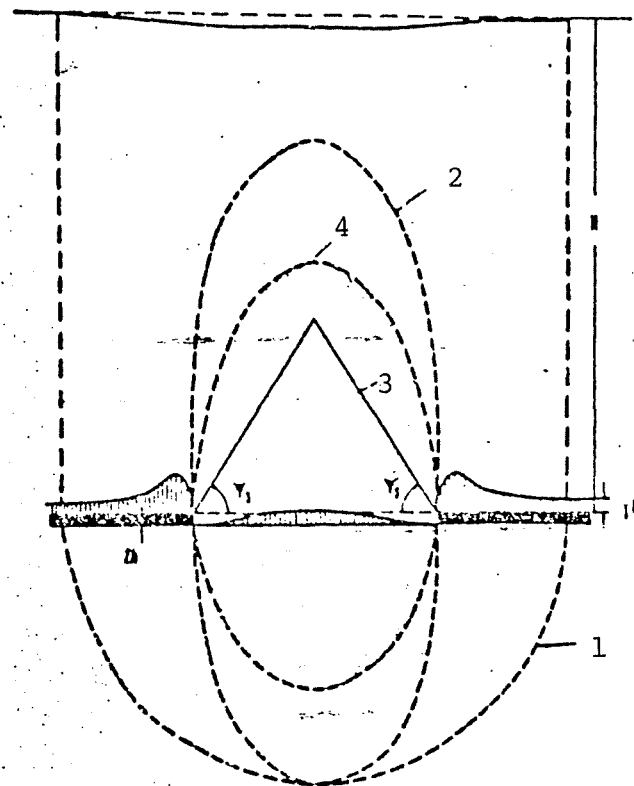
Stress gauge and calculated stress changes, as a function of distance between test pillar and pillar retreat line.

Figure 19. Stress gauge and calculated stress changes, as a function of distance between test pillar and pillar retreat line.

SOURCE: W. G. Pariseau, Inter. Journal of Rock Mech., Min. Science & Geomech. Abstracts, Vol. 18, pp. 315-316.

Figure 20

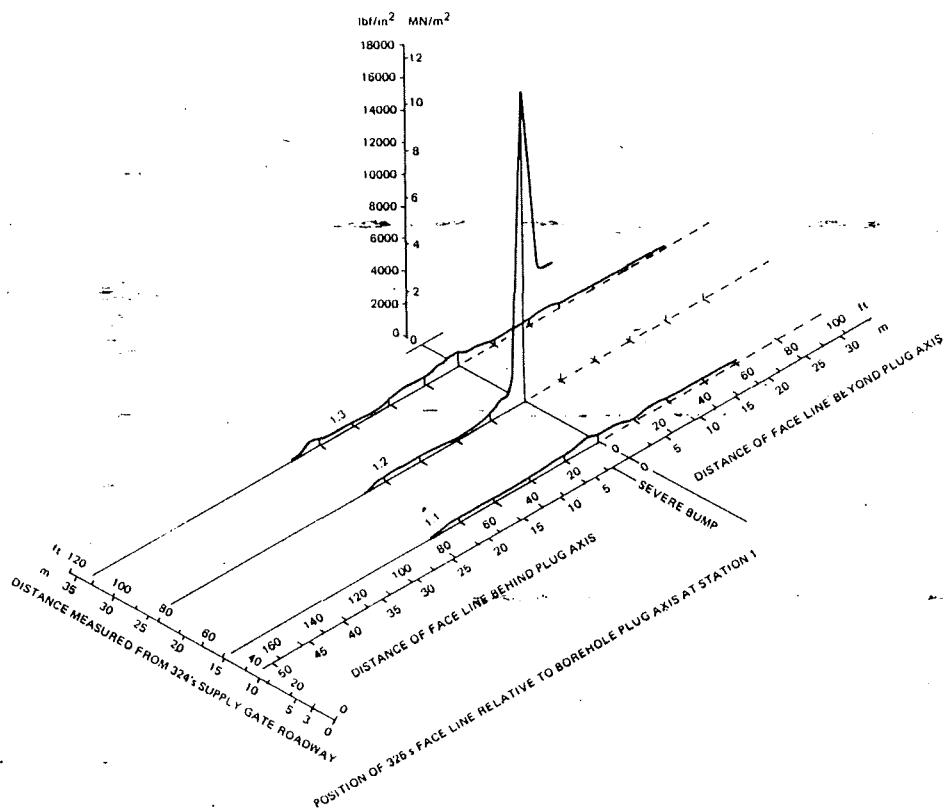
Decaying Pressure Change of Rock



- Decaying pressure change of rock formation around extraction workings, along protected layer. 1. Influence zone of workings. 2. Decompression zone. 3. Total displacement zone. 4. Safe zone.

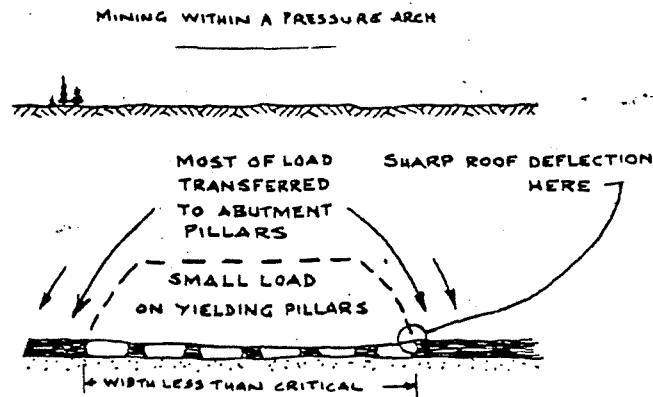
Source: J.M. Pietuchow, Measures of preventing Mine Bursts (Bumps) in the Coal Mines of U.S.S.R., translated by I. Huck, 1976.

Figure 21
Stress Distribution

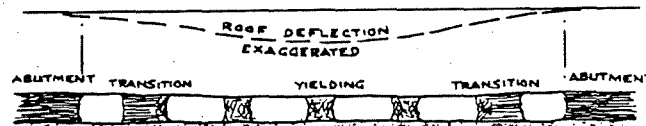


Source: R. Shepherd and W.H. Kellet, Strata Behavior,
Colliery Guardian, March, 1973

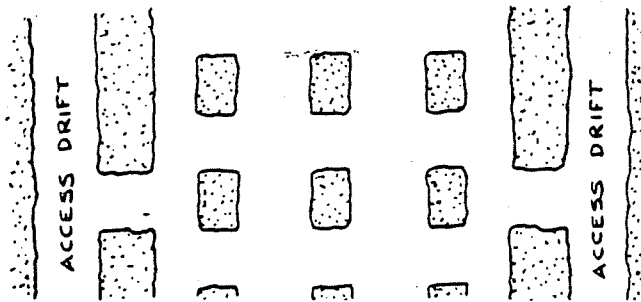
Figure 22
Mining within a Pressure Arch



Problems can be expected in the outer entries of such a panel, where there will be relatively sharp deflection of the roof, but this problem can be remedied by making the outer rows of pillars slightly stiffer than the inner pillars, redistributing the roof deflection as shown. This approach works



out nicely, as shown in the next sketch, because the stiff pillars also isolate the outer drifts, preventing local roof failures from spreading.



IF PART OF YIELDING-PILLAR AREA COLLAPSES, ACCESS DRIFTS ARE PROTECTED BY STIFFER 'CHAIN' PILLARS.

Source: Jack Parker, How to Design Better Mine Openings, E/MJ, December, 1973.

Illustrative of the views of some of these authorities is a statement in a personal communication from Louis A. Panek, Senior Technical Advisor at the USBM Denver Research Center, and a long-time student of the geotechnical factors in undercut caving mining:

"Your concept, supported by observations, that the abutment loading on coal is perhaps more important than the water jet in hydraulic mining is similar to what I have found from analysis of a large number of ground movement observations in the vicinity of a large active caving zone, namely that the large extraction zone causes the properties of the nearby mineral materials to be quite different from those of the typical laboratory test specimen. Failure to recognize such factors leads to meaningless designs."

In addition, probably all experienced mining engineers, supervisors and miners who have spent years extracting pillars or working on longwall faces are very familiar with this phenomena. In fact, where pillars are extracted, they are constantly trying to cope with this pressure by such remedies as:

- Recovering as much coal as possible from the pillars, thereby allowing caving to be as complete as possible, and minimizing the width and intensity of the abutment pressure.
- Maintaining as straight a pillar line as possible.
- Maintaining pillars as uniform in size as possible.
- Optimizing the open area outby the caving edge.
- Mining as much as possible in the distressed (crushed) zone near the caving edge.
- Destressing by water infusion, etc., etc.

Except at the tailgate corners of retreat panels, most longwalls handle the abutment pressures very well. As the abutment pressure builds up ahead of the face in a new longwall face, the coal becomes partially crushed and easier to cut, and the face becomes more productive. The coal is cut at higher rates with less consumption of power and wear on the shearing machine.

It is intended in this plan to take advantage of this same phenomenon. Peak abutment pressures generally range from 3 to 5 times the normal strata pressure. The peak usually occurs from 15 to 40 ft. from the caving edge (note previous figures). It varies for a number of reasons, some of which involve the strength and elasticity of the coal, the time the face has been standing, the length of the cave line, the straightness of the cave line, the overburden depth and the strength of the overburden, etc. The more pillars and "stumps" that are lost in the extracted area, the wider is the abutment pressure, and the farther it runs out into the adjacent pillars. Also, the more the cave line deviates from a straight line, the wider and more severe is the abutment pressure.

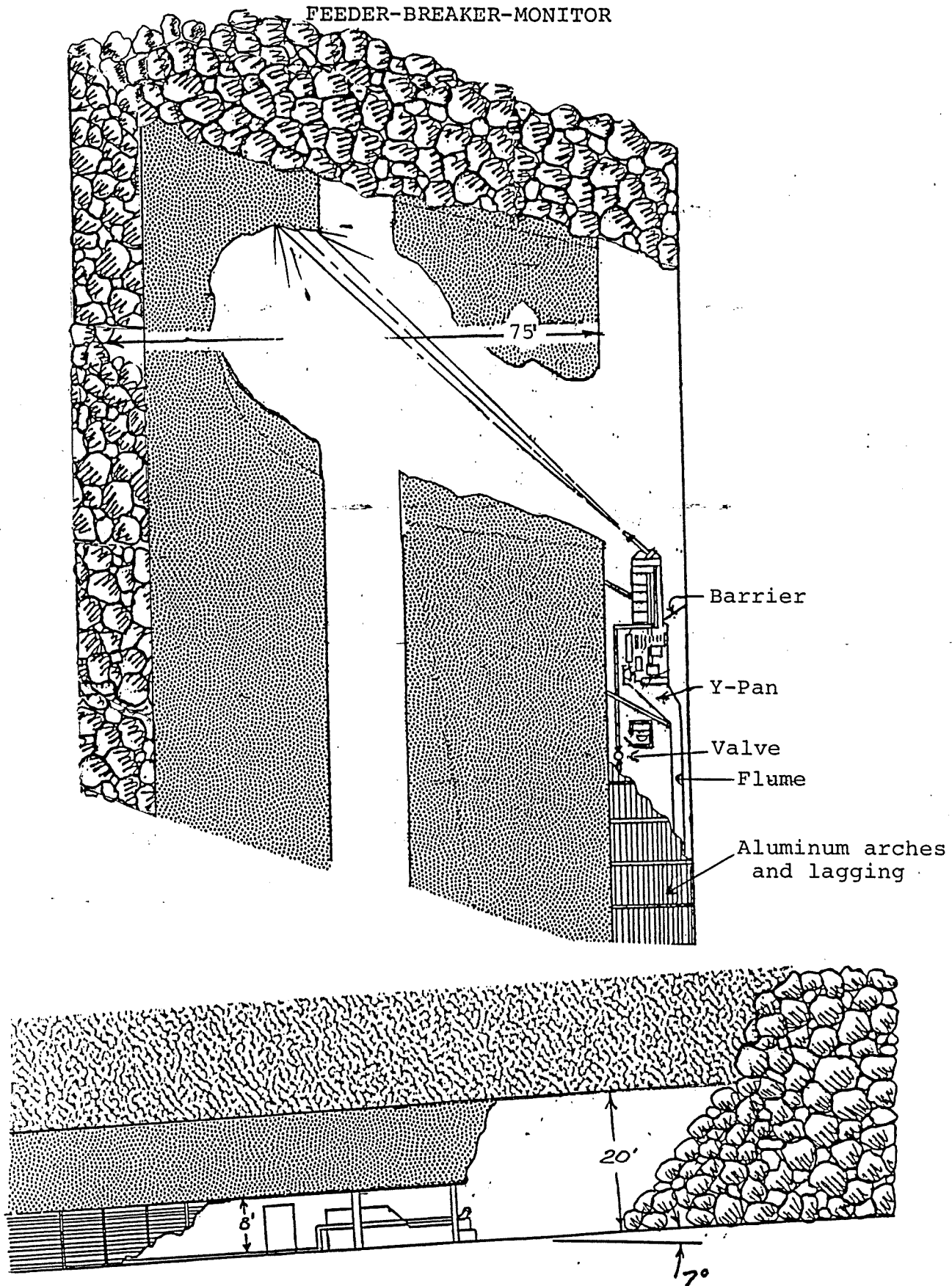
~~Retreat~~ Production

In planning hydraulic mining at this site, the broken pillar and top coal is to be recovered by a high volume, high pressure jet of water directed by a monitor, mounted on a feeder-breaker (see Figure 23). The feeder-breaker unit will serve to crush the oversize lumps to a size that will flume readily, without blocking the flumes, and with a minimum ratio of water to coal.

In normal operation the feeder-breaker is located in the fluming room near the caving edge. The inbye end of the feeder-breaker with the monitor is positioned just under the last arch, 30 to 40 ft. back from the lip of the cave. Having to withstand the abutment pressures, this place must be secure. To insure that it is adequately supported, it will be driven with a small cross-section no larger than is necessary to provide enough working room around the boom header used for driving the place, and the feeder-breaker-monitor for retreat. The boom header is about 6.2 ft. by 6.5 ft., and the feeder-breaker-monitor unit is about 5 ft. by 5 ft. The smaller dimension of the feeder-breaker, therefore, will permit a fair degree of convergence without "freezing" the unit in place. The boom header will be out of the

FIGURE 23

FEEDER-BREAKER-MONITOR



SCALE: 1" = 20' (approximately)

room before the abutment pressures occur; therefore, its larger dimensions will not be a limit to the amount of convergence that can be tolerated.

The fluming room is supported with small cross-section yieldable arches which have been proven the world over to be able to withstand severe abutment loads, and the resulting crushed ribs, bumps, convergence, etc.

Bumps of the violent type will be avoided by:

1. Sizing all pillars small enough to be gently yielding, rather than so large as to store great amounts of energy conducive to violent bumps. This avoids not only severe bumps but also leaving solid pillar cores which tend to punch holes in the roof or shear the roof at the ribs.
2. Maintaining the edge of the gob in as straight a line as possible, after allowing for the 150 ft. offsets from one retreating room to another.
3. Sizing pillars uniformly.
4. Open end extraction of pillars. This means that only extraction in the distressed (crushed) edge of the pillar next to the gob will be practiced.
5. More complete extraction of pillars. Since no machines or men have to go out into the pillars being extracted, virtually all pillars can be extracted, or at least destroyed, with the water jet. In normal pillar extraction with continuous miners and shuttle cars, pillar remnants, stumps and fenders are often left, because it is too hazardous to leave the men and machinery in the pillared area any longer. There is no risk to the jet of water; therefore, there is no need to leave such pillar remnants.
6. Drastically reducing the time for recovery of the pillars. The production rate with the jet is at least double that of the best continuous miner-shuttle car unit. Moreover, each pillar is smaller, only 26 by 40 ft., than a conventional room and pillar system would use at this depth. Time for

recovery is probably not more than one-fourth that for present pillar extraction methods. In addition, there is so little equipment used for hydraulic mining, and it is so simple and maintenance-free, that pillar extraction can be carried on continuously, 3 shifts per day.

Three monitors are provided so that production can be continuous, thus:

- 1 - producing
- 1 - being moved back and its place being prepared
- 1 - serving as a standby.

The feeder-breakers size the coal to a top dimension of 6" and smooth out the surges in coal flowing out of the pillar area. Thus, for a given volume of water, the amount of coal can be maximized and fluming made more reliable.

The feeder-breaker also serves as a base for the monitor and for a winch used for moving the unit, dragging pipe and moving arches. As the feeder-breaker-monitor retreats down the room, one-half of the coal pillar on either side will be extracted. This "double wing" room retreat system allows a high ratio of monitor coal to be mined per unit of development work and per move-back of the monitor.

The flatter pitch at this site also reduces dilution and delays to the monitor from rock falls or rock slides. When the monitor cuts through the pillar in a mine having a much greater pitch (e.g., 30 degrees), the rock in the gob slides down, not only blocking the monitor but also contributing to dilution. Likewise, when the roof caves in, the rock slides to the monitor causing dilution and delay. The 7 to 8 degree pitch at Price River Coal avoids most of these problems.

Productivity from a monitor is dependent on many things, e.g.:

- workability of the coal, including strength, elasticity, hardness, jointing and fractures, etc.;
- water pressure;
- distance from the nozzle to the coal;
- water volume;
- percentage of total time the monitor operates; and
- degree to which the coal is broken or crushed by natural forces of overburden and gravity.

In turn, the degree to which the coal is broken by natural forces is dependent on:

- the extent of undercutting, i.e., inducements to caving and crushing;
- abutment pressure;
- overburden depth; and
- strength of roof and overlying strata (see previous figures on this subject).

While the strength (or workability) of the coal is usually looked upon as a major factor in monitor productivity, the induced caving plan in this study makes coal workability relatively unimportant. The Book Cliffs coals in Utah are above average in strength and elasticity. If it were necessary to cut all the coal hydraulically at any significant distance from the nozzle, as in pillar extraction, it would not be an economical proposition. The pressure, pipe and power requirements would be too high. This plan substitutes abutment and induced caving pressures, which are relatively free, for water jet pressure, which is expensive because of the high cost of electrical energy.

Having thus broken the coal in the pillars, the key remaining variable affecting productivity is then water volume. This plan provides a minimum of 3,000 gpm. Minimum pressure at the nozzle is 1,000 psi, giving enough energy to project the jet at least 100 ft. on either side of the monitor, sufficient force at that distance to attack the crushed pillar remnants.

For any given set of mining conditions, there is a trade-off between:

- extraction or cutting distance;
- percent of recovery; and
- extraction rate.

To optimize these variables, actual experience will be needed. However, room and pillar mining has been practiced in this immediate area of Utah for more than 100 years. Based in part on the experience of: room and pillar mining in Utah, hydraulic mining in the Balmer South Mine of B. C. Coal, and induced caving methods in general, the following is estimated for this site, given an average volume of 3400 gpm and a minimum nozzle pressure of 1,000 psi:

- extraction distance, maximum: 120 ft.
average: 60 ft.
- recovery: 80%
- extraction rate: 4.7 tons per minute.

It is preferred to maximize the ratio of low cost monitor coal to high cost development coal by increasing the extraction distance, and in turn the spacing between rooms, rather than maximize the tons per shift from the monitor. Allowing for 75 ft. of extraction on either side, and 40 ft. move-backs, an area 150 ft. by 40 ft. would be extracted per move-back.

If the roof turns out to be too weak to permit economic recovery with the monitor rooms spaced 150 ft. apart, the spacing will have to be reduced. If the roof is stronger than estimated, the spacing might be increased.

Water to Coal Ratio and Monitor Productivity

In hydraulic mining, the ratio of tons of water divided by tons of coal extracted by the monitor is important. Two ratios are frequently reported and, unless properly defined, can be confusing. They are:

1. The ratio of overall water to total tons mined. This includes low pressure fluming water, and, if for convenience, pumps are not shut off when coal is not being transported, the ratio may include water that is pumped and wasted.
2. The ratio of high pressure water in tons to the tons mined by the monitor.

The latter is the more meaningful figure, inasmuch as high pressure water costs more than low pressure fluming water.

The volume of water delivered to the monitor serves as the fluming medium to transport the broken coal. The pressure or force with which the jet of water is delivered against the face affects the breaking power and rate. The combination of force per unit of area of the jet plus the volume of water represents the total breaking, penetrating power of the jet.

The proportion of coal which must be broken by the jet vs. that which is broken by strata and abutment pressures varies widely from property to property. As mentioned before, the conditions at this property and the induced caving system planned herein leave little breakage required for the monitor jet. Therefore, the main concern is to provide sufficient water volume to flush the unsized, rom, broken coal, out of the area being pillared along the 4 to 7 degree gradients to the feeder-breaker. Once the coal gets into the feeder-breaker, the crusher roll will reduce the lumps to 6" maximum, small enough for easy and reliable fluming.

To better insure that the larger lumps will be flushed to the feeder-breaker on the flatter gradients at this property, a generous assessment of water volume required to move the coal has been used. Some of the provisions planned to insure adequate production from the monitor are:

- An average water volume of 3400 gpm.
- 5,000 hp supplying high pressure to the monitor (using two 2500 hp pumps).
- Twin 6-inch pipes from the pumps to the monitor.
- A water/coal ratio of "3/1," more than double the average at B. C. Coal's Balmer South Mine.

The water to coal ratio at other hydraulic mines, and that projected for this site, is shown below:

	<u>Water:Coal Ratio (Typical or Average)</u>							
	<u>Monitor</u>				<u>Development</u>			
	<u>Water Volume</u>		<u>Raw</u>		<u>Water Vol.</u>		<u>Raw</u>	
	<u>in</u>	<u>tons</u>	<u>in</u>	<u>Water</u>	<u>in</u>	<u>tons</u>	<u>in</u>	<u>Water</u>
		<u>per</u>	<u>tons</u>	<u>to</u>		<u>per</u>	<u>tons</u>	<u>to</u>
	<u>gpm</u>	<u>min.</u>	<u>/min.</u>	<u>Coal</u>	<u>gpm</u>	<u>min.</u>	<u>/min.</u>	<u>Coal</u>
				<u>Ratio</u>				<u>Ratio</u>
Balmer So. with								
1 pump	1500	6.25	4.5	1.4	800	3.3	3.0	1.1
2 pumps	2300	9.58	7.0	1.4	800	3.3	3.0	1.1
Sunagawa	900	3.74	NA	NA	264	1.1	0.9	1.2
Lujiatuo	1320	5.5	NA	NA	NA	NA	1.5	NA
Price River Coal								
Minimum	3000	12.5	4.2	3.0	1300	5.4	3 ⁽¹⁾	1.8
Average	3400	14.2	4.7	3.0	1600	6.7	4 ⁽¹⁾	1.7

(1) Probably limited by cutting capacity of mining machines.

The above table then derives, from the volume of water and the estimated ratio of water/coal, the typical monitor production rate of 4.7 tons per minute. This rate would, of course, vary with many factors, e.g., static and friction head as it affects volume

and nozzle pressure, seam thickness, etc. The average condition is used in this report for estimating production and costs.

If demand and prices justified more tonnage, production could probably be increased by reducing the spacing between monitor rooms and, in turn, increasing the development cost.

In order to handle the typical 3400 gpm water volume, and at the same time keep the high pressure pipes at a size which can be handled and coupled more easily, twin 6-inch diameter pipes will be used to within a few hundred feet of the monitor. This also permits the use of a more economical model HP-70ES Victaulic coupling.

Monitor production will utilize 2 workers in the room. They will operate the monitor alternately and will stagger their lunch time. In order to maximize mine production and economize on capital and peak load power costs, the operators of both the monitor and the dewatering station will change shifts at their working places, stagger their lunch period, and work 9-1/4 hour shifts portal-to-portal. All other employees will work 8 hours portal-to-portal.

The typical time available for monitoring coal is then estimated as follows:

	<u>Minutes</u>
Available time per shift, portal-to-portal	555
Less non-productive time:	
Mantrip time and miscellaneous	<u>75</u>
Time available at the face	480
 Production time: 69%	 331
Monitor production per shift:	
331 x 4.7 tons/minute =	1556

Move-backs and arch recovery will require 4 men each shift. A move-back of the feeder-breaker-monitor also entails removing the "Y-pan," diversions, flume, and high pressure pipes. The feeder-breaker-monitor is winched back to the new position, the "Y-pan" and barricades installed, and the flume and high pressure pipe reconnected for mining the next pillar block.

After the feeder-breaker-monitor is pulled back, the arches will be loosened and recovered, utilizing a tugger hoist or winch. (The Lujiatuo hydraulic mine in China has developed excellent procedures for economically recovering 99.9% of their arches.)

The frequency or number of move-backs depends on the seam thickness and the strength of the roof. Overall mine recovery has been estimated at about 80%. Therefore, the amount of raw coal recovered per move-back of 40 ft. for various heights is:

Seam Thickness in ft.	Coal Per Move-back					No. of Move-backs Required /Monitor Shift of 1556 Tons
	Raw Coal In-Place Before Develop.	Coal Re- covered @ 80%	Coal Mined in Develop- ment	Coal Re- covered by Monitor	Pillar Coal Re- covered with 7% Dilution	
13	3120	2496	680	1816	1943	0.80
20 (avg.)	4800	3840	680	3160	3381	0.46
25	6000	4800	680	4120	4408	0.35

For average conditions with a seam height of about 20 ft., the 4-man crew provided for moving back would have about 2.2 shifts to accomplish their work, including the recovery of arches.

It is estimated the 4-man move-back crew over 2 shifts will use their time as follows:

Total time available in man-minutes: $2 \times 4 \times 410 = 3,280$

<u>Task</u>	<u>Men</u>		<u>Minutes</u>	<u>Man Minutes</u>
Remove pipe and flumes:	2	x	135	270
Winch back monitor:	2	x	60	120
Recover 13 arches:	2	x	820	1640
	1	x	180	180
Replace pipe and flume:	2	x	90	180
Lunch break	4	x	30	120
Other				<u>774</u>
Total				3280

Productivity for retreat mining, in raw tons per man shift:

Monitor	$1556 \div 2 =$	778
Moving back	$1556 \div 4 =$	389
Combined	$1556 \div 6 =$	259

Productivity in Mining Sections

	<u>No. of Workers Per Shift</u>	<u>Unit Shift</u>	<u>Man Shift</u>
Retreat coal monitoring and moving back	6	1,556	259
Arched rooms	5	63	12.6
Ventilation rooms	2	63	31.5
Undercut rooms	<u>2</u>	<u>181</u>	90.5
Total of Sections	15	1,863	124.2 avg.

TABLE 4
TONNAGE AND CREW REQUIREMENTS FOR DEVELOPMENT

Coal thickness in ft.	<u>20(1)</u>	<u>13(2)</u>	<u>25(3)</u>
Monitor raw coal per shift	1556	1382	1617
Monitor shifts per year	720	720	720
Percent of total coal mined by monitor, from Table 3	83.5	74.2	86.8
Total tons per year (000s)			
raw coal from monitor	1120	995	1164
raw coal total mine	1341	1341 ⁽⁴⁾	1341 ⁽⁴⁾
Tons of development raw coal required per year in: (see percentages in Table 3)			
arched rooms (000s)	45.6	71.1	36.2
ventilation rooms (000s)	45.6	71.1	36.2
undercut rooms (000s)	<u>130.1</u>	<u>203.8</u>	<u>104.6</u>
Total development	<u>221.3</u>	<u>346.0</u>	<u>177.0</u>
Tons/year monitor raw coal (000s)	<u>1120.0</u>	<u>995.0</u>	<u>1164.0</u>
Tons/year total raw coal (000s)	1341.3	1341.0	1341.0
Tons/year total coal, clean (000s), 86% yield	1154	1154	1154
Development tonnage required to be mined per average unit shift over 720 shifts per year in:			
arched rooms	63	99	50
ventilation rooms	63	99	50
undercut rooms	181	283	145
No. of workers required per unit shift in:			
arched rooms	5	8	4
ventilation rooms	2	3	2
undercut rooms	2	3	2

- (1) Average thickness of reserves, used for cost purposes.
- (2) Minimum thickness for any sustained period of several months, shown for information only.
- (3) Maximum thickness shown for information only.
- (4) Assumes the reductions in monitor productivity resulting from thinner seams are equal to the increases in the amount of development required; likewise, the increases in monitor productivity from working in thicker seams is offset by the decrease in development tonnage required.

Ventilation

The air requirements can probably best be estimated from the experience in the No. 5 Mine. When producing 6,000 tons per day this mine liberated 1,800,000 cu. ft. of CH_4 per day, equivalent to 1250 cfm.

The daily tonnage planned from the hydraulic mine is about the same. Therefore, if methane liberation were to be the same, the air requirements for various degrees of dilution would be:

% CH_4	0.3	0.4	0.5
Air required, cfm	417,000	312,500	250,000

The splits to be ventilated are:

Main levels		2
Monitor rooms on retreat:	active	1
	moving back	1
	spare	1
Monitor room on development		1
Monitor room developed ahead		<u>1</u>
Total		7

It is worthy to note the experience of providing ventilation for a longwall face in another part of this property with similar gas content. There it was found that 100,000 cfm on the split was adequate. Of this quantity, 50,000 coursed through the face and 50,000 passed to the bleeders by way of the entries alongside the gob.

Currently being installed in Crandall Canyon, about 2,000 ft. from the edge of the thick coal to be mined in this plan, is a new intake shaft and fan (see Figure 24). The shaft is 1400 ft. deep and 26 ft. in diameter. Intake air is being provided by two 11 ft. diameter fans, each with a capability of 720,000 cfm at 8.1 inches of water gauge; at lower ventilation pressures, volume could go up to a maximum of 1,200,000 cfm. The maximum pressure capability is 14 inch water gauge.

FIGURE 24

VENTILATION SYSTEM

NEW PEERLESS ROCK SLOPES
(2) RETURNS

Nº 6 SEAM
VENTILATION ROOMS
(4) RETURNS

CRANDALL CYN.
EXHAUST SHAFT

CRANDALL CYN.
INTAKE SHAFT
(2) FORCED-AIR FANS
CAPABILITY:
1,200,000 cfm @ 14"

SUB 3 SEAM MAIN ENTRIES
(3) INTAKES
(3) RETURNS

20,000 cfm
10,000 cfm
LEAKAGE
350,000 cfm

20' DIA. INTAKE SHAFT

ROCK TUNNEL
MATERIAL SUPPLY
(1) INTAKE

ROCK TUNNEL
CONVEYOR
(1) RETURN

ROCK TUNNEL
MATERIAL SUPPLY
(1) INTAKE

Nº 6 SEAM
VENTILATION ROOMS
(6) INTAKES

80,000 cfm 200,000 cfm

320,000 cfm

320,000 cfm

40,000 cfm
LEAKAGE

Nº 6 SEAM MAIN LEVELS

DEWATERING STATION

10,000 cfm

From this source it is planned to draw 350,000 cfm, equivalent to 50,000 cfm per split. It is estimated that, after losses from leakage, 40,000 cfm will reach the last open crosscuts, the pillars being extracted. The coal mine safety law allows up to 1% methane in the return air. Assuming the main returns could average 0.6% methane, there is then capacity to remove: $350,000 \times .006 = 2100$ cfm. (See Figure 24.)

The 350,000 cfm of intake air will be forced down the 6,000 ft. Sub-3 seam aircourses. Approximately 80% will go up the 20 ft. air shaft to the A-seam and 20% up the 2500 ft. rock tunnel to the A-seam. Thence, the air passes to the main level aircourses. The main returns are up the four 4500 ft. aircourses to the New Peerless rock slopes which conduct the air to the surface.

To ventilate the development faces, brattice lines are used to conduct the air from the last open crosscut nearly to the face.

The dual room concept for each monitor room provides a return air course for each working place. The return room is positioned in the upper part of the seam where the roof rock can be secured with bolts. These rooms also serve as returns and bleeders for the ventilation of the area being pillared. Their location at the top of the seam is efficient for sweeping the gas from the excavated area. In other words, this system of ventilating the retreating rooms has these advantages over a system which passes return air over the gob:

1. It is a much more reliable passageway for the return air.
"Through the gob" is a haphazard and variable return airway. It cannot be traveled or inspected. The wider the gob area, the less air that can be passed through. On the other hand, the return airway used in this plan is travelable; it is positive and much more dependable.
2. Passing air through the gob creates zones which are not ventilated enough to cool them but which receive sufficient

air to permit combustion and the build-up of heat; thus, conditions are conducive to spontaneous combustion. The ventilation rooms planned herein eliminate this situation and minimize the risk of "spon-com." The zone of incomplete ventilation, i.e., stagnant air where heating can start, is kept moving by operating the monitor round-the-clock on a 3-shift per day schedule.

WATER SYSTEMS

The water systems for Price River Coal's property include:

- pumping of natural mine water inflow,
- low pressure water for fluming development coal,
- high pressure water for hydraulic jet pillar extraction.

Figure 25 displays the hydraulic system schematically.

Pumping of Natural Mine Water Inflow

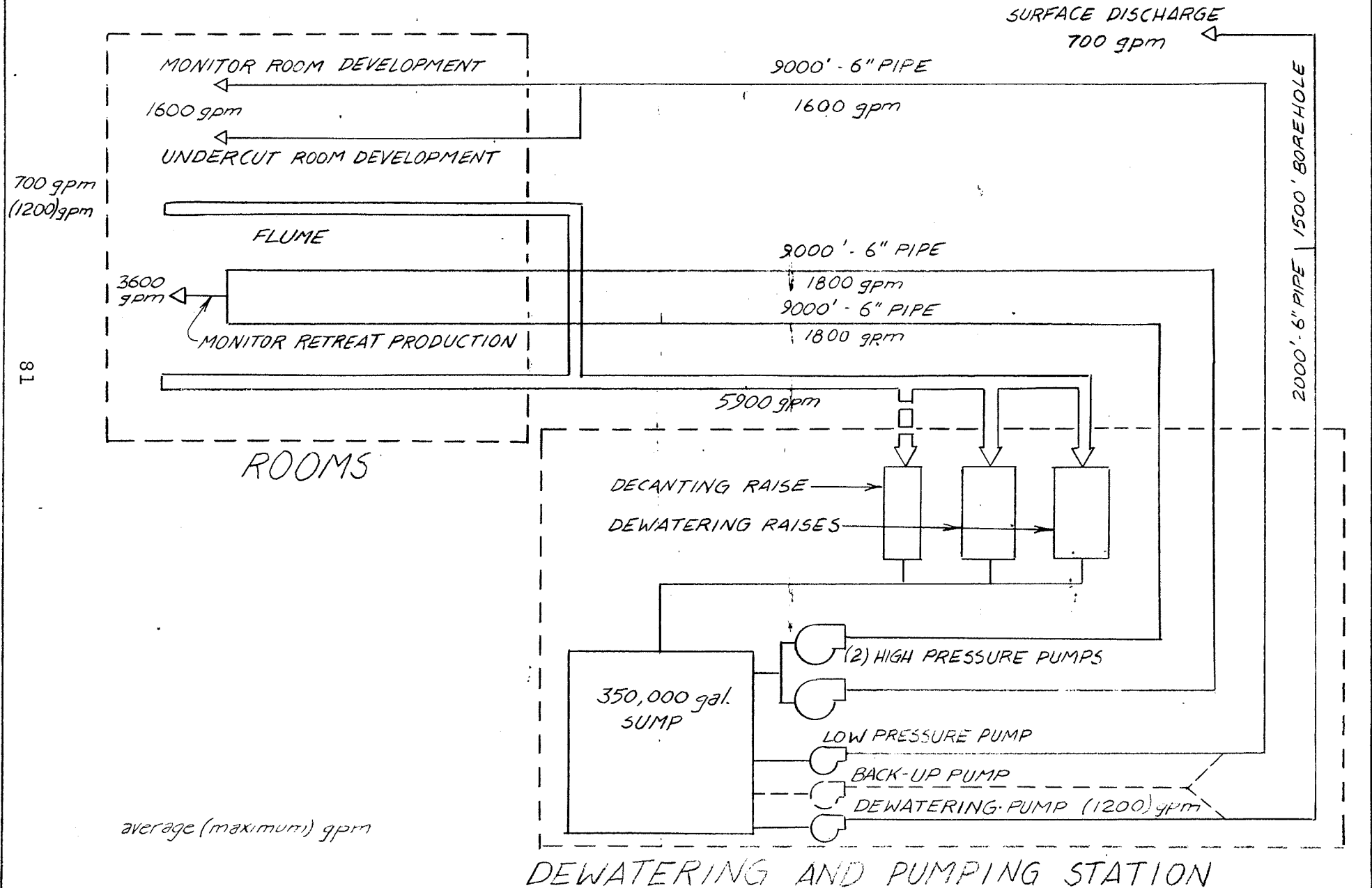
The old workings of "New Peerless" Mine contain an estimated 400 million gallons of water which must be removed before connections are made thereto from the hydraulic mine. The company has an active permit to discharge this water at the rate of 300 gpm into the Price River. The quality of the water is considerably better than the normal water in the Price River. The plans for longwall mining in the Sub-3 seam call for development headings to be driven below workings in the old Peerless Mine. Also, the plans for mine dewatering in Sub-3 provide pipe and pumping capacity to remove the water in the "New Peerless" before the hydraulic mine will be developed. Therefore, the estimates for the hydraulic mine in the A-Seam are spared the cost of dewatering this old mine.

Providing sufficient barriers are left in mining under the Price River, the inflow of natural mine water is not expected to be a major problem at this property. At the Sunnyside Mines, about 40 miles to the east in the same Book Cliffs Coal Field, full extraction of the coal seam has been practiced by room and pillar and longwall mining under Grassy Trail Creek, a perennial stream. The overburden there is only 700 to 1,000 ft.; yet the surface stream continues to run apparently undiminished. About 1,000,000 gallons per day (700 gpm) are pumped out of the mine. This water has a higher salt content and a different quality than the water flowing on the surface, which indicates that there is not free communication between the surface stream and the gob below. The shale layers between the sandstone tend to seal the cracks caused by subsidence.

In this report, a capacity of 1200 gpm will be provided for removal of natural mine water.

FIGURE 25

HYDRAULIC SYSTEM



All mine workings are on a fluming grade, so natural mine water will join the production water and flow to the dewatering station. No extensive network of dewatering pumps and pipes is needed to keep the mine dewatered.

The mine pumping system then consists of only a 1200 gpm pump at the bottom of the dewatering station and a 6-inch pipeline running 2,000 ft. southwest along the main levels and 1500 ft. up a borehole to Price River Canyon. In case of a breakdown, one of the fluming water pumps, or high pressure pumps, could be used as a spare.

		Feet
The head required is:	Elevation head: Dewatering station	100
	2,000 ft. main level @ 7%	140
	Borehole to surface	1500
Pipe friction: $3500 \div 100 \times 8.87 \text{ ft./100 ft.}$		
$\times 1.2 \text{ (see Figure 26)}$		373
		2113

FIGURE 26
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

5 Inch		6 Inch		8 Inch	
Extra strong steel—sch 80		Extra strong steel sch 80		Extra strong steel sch 80	
4.813" inside dia		5.761" inside dia		7.625" inside dia	
Flow U S gal per min	Head loss ft per 100 ft	Flow U S gal per min	Head loss ft per 100 ft	Flow U S gal per min	Head loss ft per 100 ft
460	4.50	400	1.39	450	.428
480	4.88	450	1.74	500	.522
500	5.28	500	2.13	550	.625
550	6.35	550	2.55	600	.736
600	7.51	600	3.02	650	.857
650	8.77	650	3.52	700	.986
700	10.1	700	4.06	750	1.13
750	11.6	750	4.64	800	1.27
800	13.1	800	5.25	850	1.43
850	14.8	850	5.90	900	1.59
900	16.5	900	6.60	950	1.77
950	18.4	950	7.33	1000	1.95
1000	20.3	1000	8.09	1100	2.34
1100	24.5	1100	9.74	1200	2.77
1200	29.0	1200	11.5	1300	3.23

Pump required: (see Figure 27)

Ingersoll-Rand 4x11 DA or equivalent

Impeller: 4x11 DA, 285 mm

Curve No.: P-4x11 DA-1-2

Quantity: 1200 gpm

Head per stage: 470 ft. at 1200 gpm

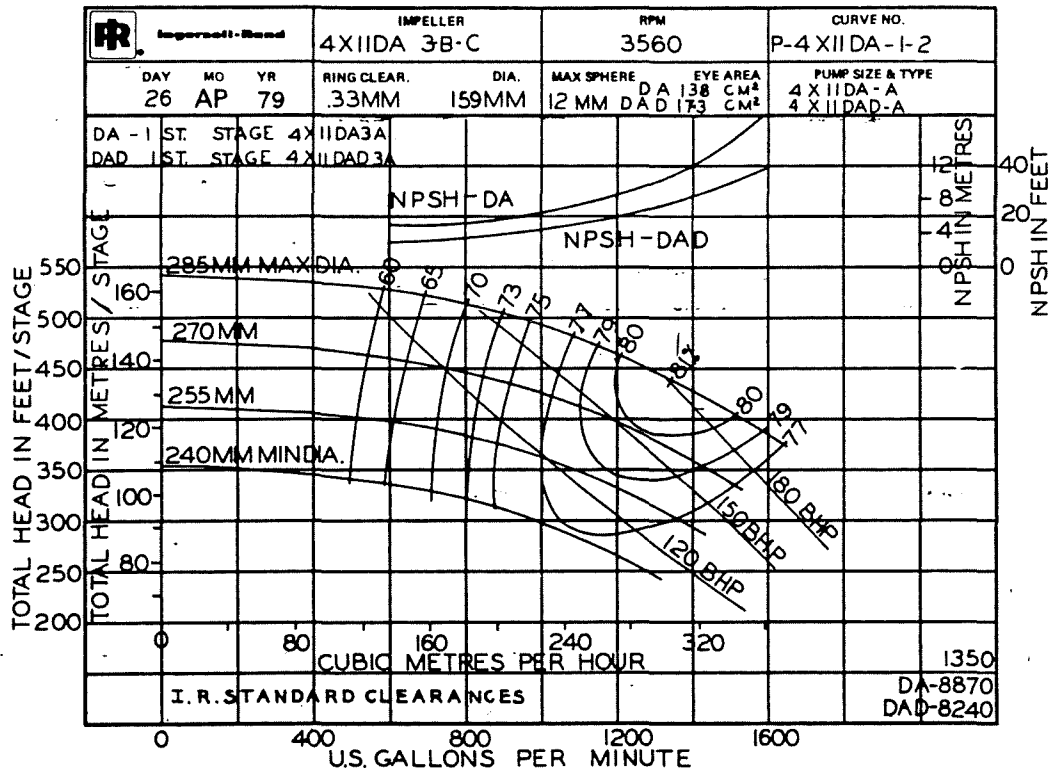
Efficiency: 80%

Hp per stage: 200

No. of stages required: 5 (see below)

Total hp for pump: 1,000

FIGURE 27



In the interest of standardization, in-line spares, and interchangeability, the pump selected will be interchangeable with that used for low pressure fluming water. The piping will be arranged so that a back-up pump can serve either for dewatering or low pressure fluming.

Low Pressure System

Low pressure pumps will supply water through as much as 9,000 ft. of Schedule 80, 6-inch pipe to the boom header and continuous miners doing development and undercut work. As the boom header advances in the monitor room, one 6-inch, Schedule 80, steel pipe will be installed. Aluminum pipe will be used in the ventilation and undercut rooms. After a monitor room is completed, an additional high pressure pipe will be installed so that two 6-inch pipes will supply high pressure water to the monitor.

The same pump as selected for mine dewatering will have the capacity to supply 1300 gpm to the development faces under the most adverse conditions of static and friction head at 81% efficiency. Under average conditions, the pump will supply 1600 gpm at 77% efficiency.

	<u>Hydraulic Conditions</u>	
	Head:	
	Gpm:	
	<u>Maximum</u>	<u>Average</u>
Elevation head	<u>1300</u>	<u>1600</u>
Friction head	740	500
Worst case, $9,000 + 100 \times 13.5 \times 1.2$	1458	NAp
Average case: $5500 + 100 \times 20.3 \times 1.2$	<u>NAp</u>	<u>1340</u>
Total	2198	1840
Per stage	440	368
Hp for 5 stages	1000	1000

High Pressure System

Under the maximum conditions of static and friction head, 3,000 gpm of high pressure water will be supplied to the monitor at a minimum nozzle pressure of 1,000 psi, through two parallel 6-inch, Schedule 80 pipes in the main level and in each of the rooms. Two 2500 hp pumps have been selected with performance capability as shown on the curves in Figure 27. For both the right and left sides of the slopes, the horsepower requirements are calculated for the worst conditions of static and friction head, and for the average conditions.

	<u>Hydraulic Conditions</u>		
	<u>Head:</u>	<u>Maximum</u>	<u>Average</u>
	<u>Gpm:</u>	<u>3000</u>	<u>3600</u>
Pressure in ft. of head			
At nozzle (1,000 psi)		2309	2309
Elevation head		740	500
Friction in pipes		<u>1920</u>	<u>1915</u>
Total		4969	4724
Head per stage in feet		497	472
Hp per stage		240	240
Hp for 10 stages per pump		2400	2400

Pumps required - Figure 28

Main pump (Ingersoll-Rand DA series or equal)

Curve No. P-6x11 DA-2-3

Impeller: 290 mm

Head per stage: 497 ft. at 1500 gpm
472 ft. at 1700 gpm

No. stages required: 10

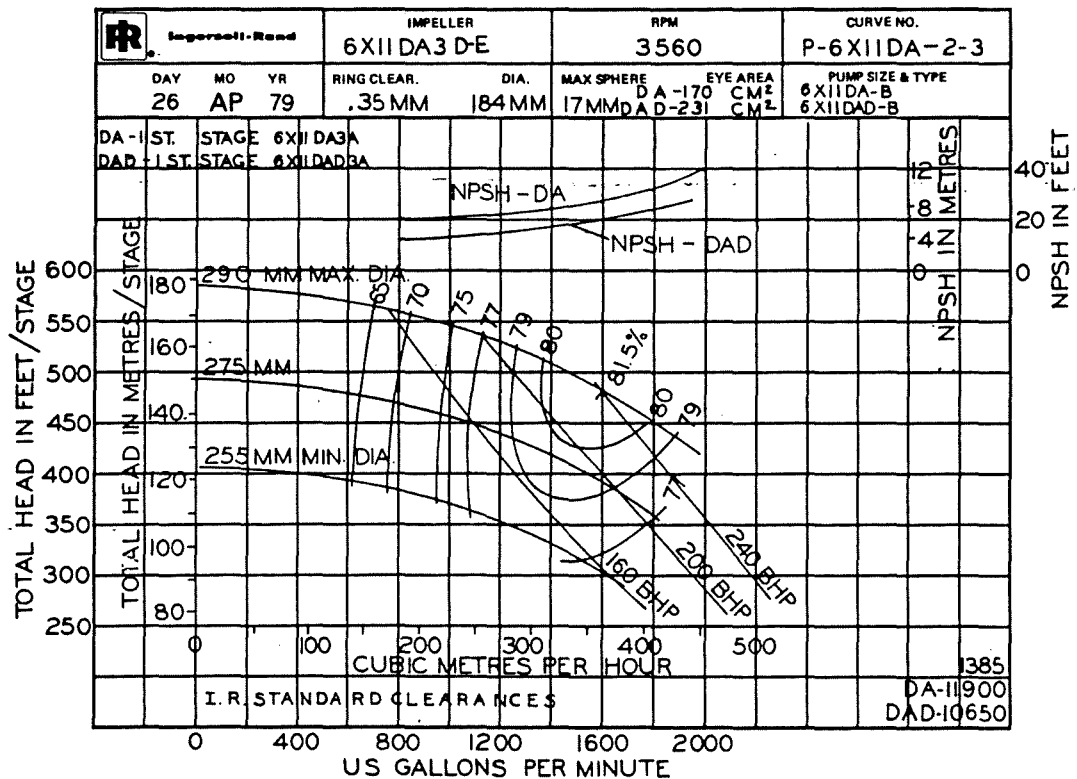
Hp per stage: 240

10 stages = 2,400 hp

Use 2 - 10-stage pumps each with 2500 hp motor having a 15% service factor.

FIGURE 28

HIGH PRESSURE PUMP CURVE



INGERSOLL-RAND COMPANY, PUMP GROUP, PHILLIPSBURG, N.J. 08865

Coal Dewatering Station

The dewatering station is located underground and is designed to accommodate production from one hydraulic monitor and the development units. The total maximum water flow is planned at:

	<u>gpm</u>
Monitor	3600
Fluming development coal	1600
Natural water	<u>700</u>
Total	5900

The monitor has an estimated working time of 331 minutes per shift, or 69% of total time. The low pressure water for development is used only as required, about 175 minutes per shift for average conditions, ~~or~~ 36% of the time. The percentage of time that both development and monitor water is being used concurrently is: $.36 \times .69 = 24\%$. The time that the dewatering station will be receiving no feed is: $.64 \times .31 = 20\%$. Therefore, the dewatering station receives a variable load, but most of the time only the monitor production, 4 to 6 tons per minute.

During the 24% of the total time when development coal is being mined concurrently with monitor coal, the average tons per minute will be about 8, i.e., 480 tph; peak production is planned to 750 tph.

To handle peak conditions, the equipment will be sized for 800 tph of raw coal and 5,900 gpm. To keep the size of the chamber required to house the dewatering equipment to safer and more economic dimensions, this plan envisages a 2-line plant.

Note Figures 4 and 5. The main elements of the station are:

- A 90 ft. high by 33 ft. diameter raise for emergency decanting of slurry in the event of dewatering station failure.
- Two raises, averaging 80 ft. in height, with diameters at the widest level of 40 ft., to house the dewatering equipment.
- A 300,000 gallon sump in two rock tunnels below the dewatering station to provide for collection of water from the station to recycle to the faces; also to provide storage for water when the mine is idle.
- A rock tunnel located below the sump to house dewatering, high pressure and low pressure pumps, as well as access to them.
- A conveyor belt entry from below the dewatering raise and the decanting raise which leads to the main slope conveyor belt.

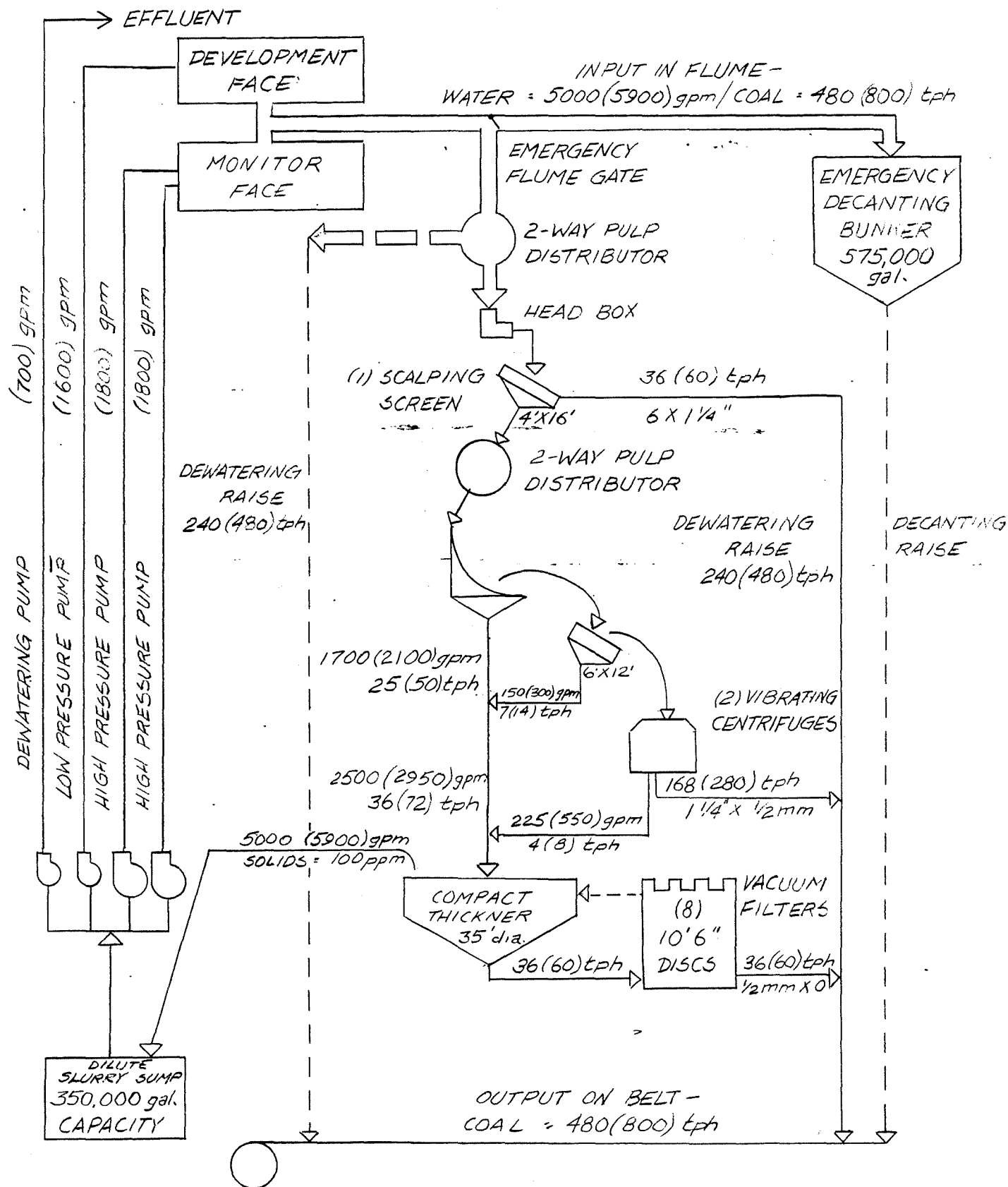
A flow sheet is shown in Figure 29.

The coal and water mixture from the mining faces will first pass over the emergency decanting bunker where the flume will be equipped with an automatically controlled gate which will open to divert the slurry to the decanting bunker if there is a failure of the dewatering equipment. This gate will provide a safeguard to protect the dewatering station from spills during power outages and other breakdowns.

Under normal operation, the slurry will be divided into two equal streams, each carrying up to 2950 gpm of water and 375 tph of 6" x 0 raw coal. Each line of the the process is described below.

The coal will pass over a 4 ft. x 16 ft. single deck vibrating screen which scalps off the plus 1-1/4 inch coal for conveying to the surface. The minus 1-1/4 inch coal with the water will be split into two process streams by a pulp distributor. Each of

FIGURE 29
DEWATERING & PUMPING STATION FLOWSHEET
 INSTALLED UNDERGROUND



the two streams will flow to a 5 ft. wide sieve bend followed by a 6 ft. x 12 ft. vibrating dewatering screen with a 1/2 mm grid.

The dewatering screens are sized primarily for water removal and not for size classification in order to retain as much as possible of the minus 1/2 mm in the oversize.

Two centrifuges will be installed in each line with a capacity of 150 tph each. The centrifugally dried 1-1/4 inch x 1/2 mm product passes to the belt conveyor. The water and minus 1/2 mm coal will be flocculated and passed to a 35 ft. diameter compact thickener, Enviroclear or equal.

(Flocculent and thickener technology has improved in recent years, allowing the use of a thickener compact enough to be housed underground, providing a competent rock measure such as the Aberdeen Sandstone is present close to the coal seam.)

The pulp which enters the thickener is estimated to contain 10 to 15% of the raw coal product, and the percent solids is estimated to range from 4.5 to 6.6%.

Size of Thickener:	<u>Peak</u>	<u>Average*</u>
Use: gpm	2950	2500
Thickener diam. (Enviroclear or equal):	35 ft.	35 ft.
Plan area:	962 ft. ²	962 ft. ²
Gpm/sq. ft.:	3.07	2.60

*With all pumps running against average static and friction heads.

The clarified overflow from the thickener will contain a maximum solid content of 100 ppm, and a maximum size of 400 mesh. The overflow is delivered to the sump and, in turn, to the low and high pressure pumps. The underflow (maximum rate, 52.5 tph) will be routed to vacuum filters in each line.

Since the material is coarse, the filters are estimated to have the capacity to dewater 100 lbs. per sq. ft. per hour. Thus, approximately 3,000 sq. ft. of filter area is required, as follows:

$$(52.5 \text{ tph/line} \times 2,000 \text{ lb/t}) \div 100 \text{ lb/sq. ft./hr.} = 1050 \text{ ft.}^2$$

Filter:

Type: Vacuum disc

Diameter: 10 ft. 6 in.

Area per disc: 155 sq. ft./disc

No. of discs: $1050 \div 155 = 6.8$ discs at peak capacity

Use 1 unit/line: 8 discs, 10 ft. 6 inches in diameter, with total filter area of 1240 sq. ft.

Actual load per sq. ft. under peak load = 85 lbs. per sq. ft. of filter area

The effluent from the filters will be recirculated to the thickener.

The advantages of using screens and centrifuges in this circuit are: relative insensitivity to pulp density, and ability to absorb surges. The higher the flow to the dewatering screens, the more inefficient the screens become, passing more fines to the centrifuges. Similarly, the thicker the bed of material in the centrifuges, the more fines will filter out. This compensating effect will tend to level out the feed to the fines circuit.

With regard to the coal which enters the emergency decanting bunker, this will be dewatered slowly by allowing the water to percolate through the coal and out the screened bottom of the bunker. The effluent from the decanting bunker will be stored and circulated to the fines circuit. When dewatered sufficiently for belt transport, the coal will be fed onto the conveyor belt to the preparation plant.

All water from the dewatering raise and the additional mine water inflow will collect in the sump below the dewatering station.

MANPOWER ESTIMATE

Since the mine workers at Price River Coal Company are UMWA, they do not produce coal on Sunday. Therefore, the mines typically operate 5 days per week, and as many shifts as needed, subject to maintenance requirements. The hydraulic mining operation is planned for 240 days per year. Where round the clock operation is needed, as for monitoring and dewatering coal, 3 shifts per day, 5 days per week, are scheduled. To permit shift change at the face, 9-1/4 hour shifts are planned. All other underground work will be done on 8-hour shifts, 5 days per week.

TABLE 5
TOTAL WORK FORCE AND PRODUCTIVITY

	<u>Total Worker Shifts/Year</u>
Underground (Table 5-A)	
Section workers	11,520
Dewatering, pumping and conveying	3,600
General underground	<u>7,920</u>
Sub-Total	23,040
Trainees	1,152
Saturday Work	<u>520</u>
Sub-Total with hourly workers	24,712
Supervision	<u>4,800</u>
Total underground, hydraulic mining only	29,512
Workers common to all mining on West side	
12 workers x 240 days per year	<u>2,880</u>
Total underground	32,392
Surface (from Price River Coal's "Technical Report Supplement, 3/4/81). Workers allocated on tonnage basis for 1989 when full production projected.	
1,154,000 ÷ 5,352,565 x 139 = 30	
30 workers x 240 days =	<u>7,200</u>
Total worker shifts per year	39,592
Active employees required:	
Hydraulic underground only, 29,512 ÷ 240 =	123
Incl. shared workers, 39,592 ÷ 240 =	165
To cover absentees (5%), total employees =	173
Productivity for:	<u>Tons clean coal per man day</u>
Hydraulic mining only	39.1
Hydraulic mining plus allocated share of other underground workers	35.6
Total hydraulic mining, incl. allocated share of workers in all common facilities, e.g. surface, preparation, etc.	29.1

TABLE 5-A
UNDERGROUND WORK FORCE
Hydraulic Mining Only

	Hours Per Shift	Shifts Per Day	Workers Per Shift	8 hr. Worker Shifts Per Year	9-1/4 hour Worker Shifts Per Year	Total Worker Shifts Per Year
Section Workers						
Development:						
Dual rooms (1 unit)	8	3	7	5,040	NAP	5,040
Undercut rooms (1 unit)	8	3	2	1,440	NAP	1,440
Pillar extraction						
by monitor	9-1/4	3	2	NAP	1,440	1,440
moving back	8	3	4	2,880	NAP	2,880
Mechanics/electricians	9-1/4	3	1	NAP	720	720
Sub-Total Sections			16	9,360	2,160	11,520
Dewatering/pumping & slope belt						
Attendant	9-1/4	3	1	NAP	720	720
Clean-up & lubrication	9-1/4	3	2	NAP	1,440	1,440
Mechanic/electrician	9-1/4	3	1	NAP	720	720
Belt attendant	9-1/4	3	1	NAP	720	720
Sub-Total Dewatering, etc.			5	NAP	3,600	3,600
General Underground						
Ventilation, stoppings, seals, overcasts, rock dust, etc.	8	1	8	1,920	NAP	1,920
Flume lines & pipe, gen. maint.	8	1	4	960	NAP	960
Material and supply crew	8	3	4	2,880	NAP	2,880
Roadway maint. for transport	8	1	4	960	NAP	960
Maint. of roof, ribs	8	1	4	960	NAP	960
Fireboss	8	1	1	240	NAP	240
Sub-Total General Underground			25	7,920	NAP	7,920
Sub-Total Hourly Workers ⁽¹⁾				17,280	5,760	23,040
Trainees (5%)				864	288	1,152
Provision for Saturday work ⁽¹⁾	8	NAP	10	520		520
Total Hourly Workers				18,664	6,048	24,712
Supervision						
Development foreman		3	1	NAP	720	720
Monitor foreman		3	1	NAP	720	720
Move-back foreman		3	1	NAP	720	720
Dewatering & pumping foreman		3	1	NAP	720	720
Shift foreman		3	1	NAP	720	720
Underground superintendent		1	1	NAP	240	240
Construction foreman		1	1	NAP	240	240
Maintenance foreman		3	1	NAP	720	720
Sub-Total Supervision			8		4,800	4,800
Total Underground ⁽¹⁾ , Hydraulic Mining Only				18,664	10,848	29,512

(1) Overtime is provided for in the cost per man day estimates.

DEVELOPMENT AND PRODUCTION SCHEDULE

The decision to develop the hydraulic mine would, among other factors, be subject to several preliminary accomplishments:

1. Finding a market for the coal.
2. Adding about 600 acres to the existing Federal leases.
3. Successful drilling of the additional acreage.
4. Obtaining necessary permits from the regulatory agencies.
5. Extraction of the D-seam reserves which overlies the area in the A-seam to be caved.

Items 2, 3 and 4 are believed to be low risk factors, and item 1 is mostly a matter of time. To the extent that finding a market is based on competitive power, this method of mining should have a favorable impact.

Assuming the above were accomplished and the starting point is the underground development, the following schedule is estimated. The principal components of the initial development work and the time required are:

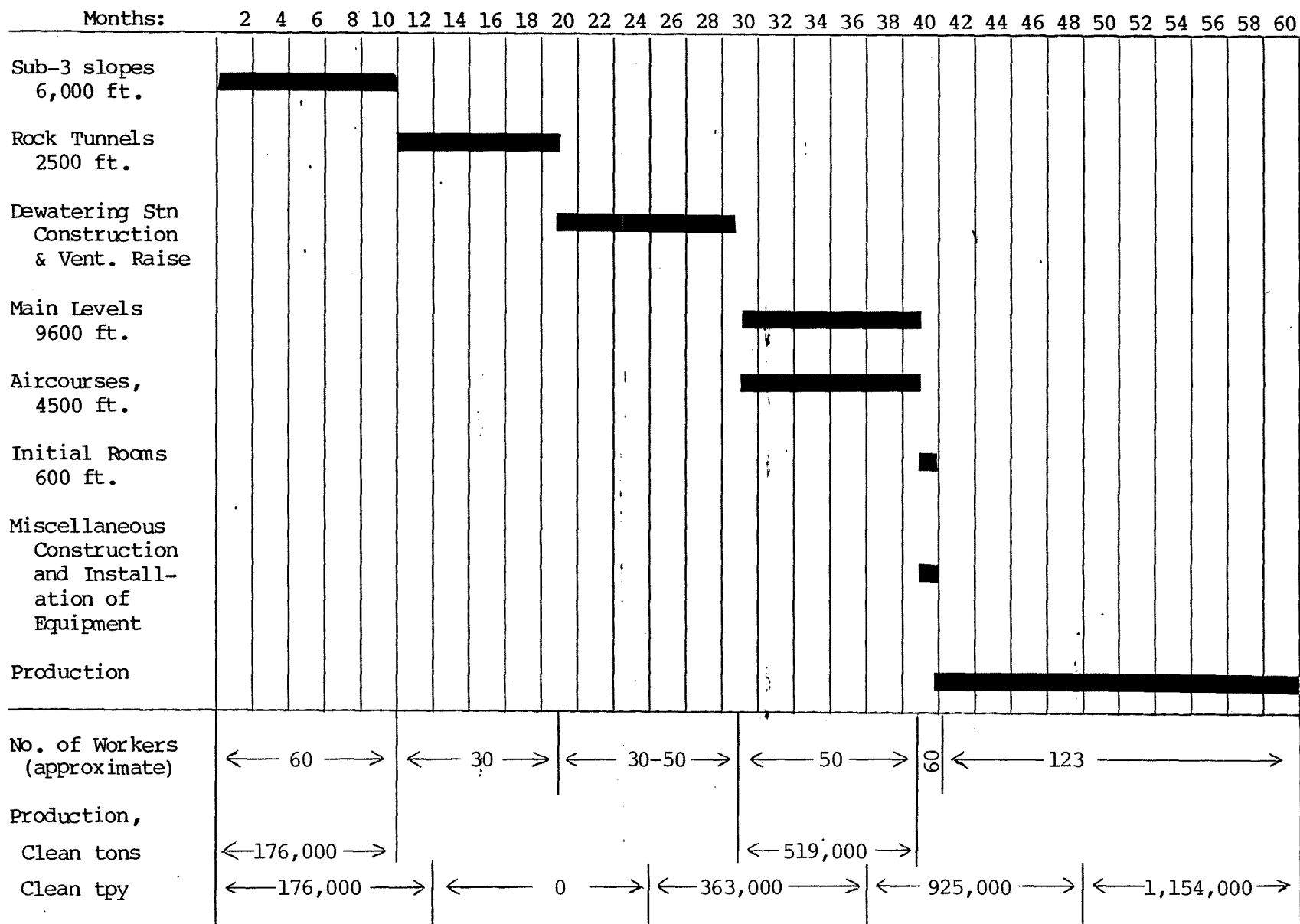
	<u>Time Required (Months)</u>
• Development of six main slopes, 6,000 ft., from the bottom of the Sub-3 slopes to the start of the rock tunnels: 10 months	10
• Two 2500 ft. rock tunnels to the dewatering station: 9 months	9
• Rock excavation, construction and start-up of the dewatering and pumping station: 10 months	10
• Development of the main levels to the southwest, approximately 9600 ft., with 2 continuous miners: 8 months	8
• Concurrent with the drivage of the main levels, development of the 4500 ft. aircourses between the dewatering station and the New Peerless rock slopes: 8 months	Nap
• Initial development of three 600 ft. long dual-rooms and undercut rooms: 1 month	<u>1</u>
Total	38

Table 6 shows the development and production schedule, including the number of workers required.

Future Development

Main level development to serve the area southeast of the dewatering station will be done after 8 to 10 years when the southwest reserves are approaching exhaustion.

TABLE 6
SCHEDULE FOR DEVELOPMENT AND CONSTRUCTION



FINANCIAL ANALYSIS

The cost estimates for the hydraulic mine at Price River Coal Co. include:

- Direct and Indirect Operating Costs
- Capital Expenditures
 - Hydraulic mine expenditures, and
 - Amortization costs of existing mine property and assets
- Cash Flow Forecast
- Internal Rate of Return

Operating Costs

Because the amount of development work per ton of coal recovered is inversely proportional to the seam thickness, the cost of coal will vary with seam thickness. For this study cost estimates are based on the average seam thickness of 20 ft. Estimated operating costs include an allocated share of the costs for common facilities such as the ventilation system, conveyor from Crandall Canyon to the surface, preparation plant, shaft hoist, shop, warehouse, bathhouse, engineering, accounting, general management, etc.

Direct Costs

Direct costs are estimated per ton of clean coal and are summarized in Table 7. Detailed costs are shown on attached Tables 7-A through 7-E as follows:

Table 7-A	Salaries and wages, including all benefits and taxes
Table 7-B	Materials and supplies
Table 7-C	Repairs and maintenance
Table 7-D	Power and other costs
Table 7-E	Common facilities: coal haulage, preparation, load out, fan, hoist, shop, warehouse, bathhouse, etc.

Indirect Costs

Indirect costs are shown as follows:

Table 7-F	General, administrative & taxes
Table 7-G	Insurance and royalties
Table 7-H	Amortization charges for existing facilities

TABLE 7

OPERATING COSTS

Production: 1,154,000 tons of clean coal per year

Number of active underground employees, excluding absentees: 123

DIRECT COSTS

Cost/ton

Hydraulic Mining

Payroll & payroll loading, Table 7-A	\$4.71
Materials & supplies, Table 7-B	2.61
Repairs and maintenance, Table 7-C	0.74
Power, Table 7-D	1.03
Other direct costs, Table 7-D	<u>0.10</u>

Sub-Total Mining

\$ 9.19

Common (or shared) services and facilities,
allocated cost, Table 7-E

Coal haulage	\$0.49
Ventilation by central fan in Crandall Canyon	0.58
Bathhouse service, lamps, self-rescuers	0.20
Shaft hoist	0.11
Warehouse and material yard	0.28
Shop	<u>0.41</u>
Coal preparation, load out, engineering, accounting, general management, etc.	<u>2.57</u>

Sub-Total Common Services & Facilities

4.64

INDIRECT COSTS

UMWA Health & Welfare levy (rounded),
Table 7-A

\$1.70

General & administrative, Table 7-F

0.50

Taxes (excl. income), Table 7-G

1.31

Insurance and other taxes, Table 7-G

0.15

Hydraulic royalty, Table 7-G

0.30

Sub-Total Indirect Costs

3.96Sub-Total of operating costs before federal
coal royalty

\$17.79

Coal price f.o.r.

\$25\$30\$35

Federal royalties, Table 7-H

Royalty through 1989

\$ 0.48

\$ 0.56

\$ 0.63

Royalty 1990 and thereafter

\$ 2.00

\$ 2.40

\$ 2.80

Total Cost Before Capital & Income Tax

Present to 1989

\$18.27

\$18.35

\$18.42

1990 and thereafter

\$19.79

\$20.19

\$20.59

TABLE 7-A
UNDERGROUND WAGES AND SALARIES
1983 Dollars; from Price River Coal Co.

	Wage Earners		Salaried Employees	Total
	8 hrs. /Shift	9-1/4 hrs. /Shift		
Payroll cost per average worker shift				
Direct wage or salary	\$99.20	\$122.45	\$144.50	
Payroll taxes, benefits, etc.				
FICA	7.78	8.99	9.84	
FUI	0.25	0.29	0.23	
SUI	0.97	1.12	0.89	
Workmen's Compensation	5.21	6.02	6.02	
Black Lung, hourly component ⁽¹⁾	3.97	4.90	5.78	
Protective clothing	0.67	0.67	NAP	
Life and medical insurance	9.99	9.99	7.46	
Accident insurance	0.70	0.70	3.28	
Dental insurance	0.97	0.97	2.35	
Holiday pay	4.89	5.65	NAP	
Vacation pay	9.77	11.30	NAP	
Sick pay	2.22	2.57	NAP	
UMWA H&W fund, hourly contribution ⁽²⁾	8.01	9.32	NAP	
Pension	NAP	NAP	7.21	
Savings plan	NAP	NAP	1.29	
Sub-Total Benefits	55.40	62.49	44.35	
Sub-Total wage and benefits	154.60	184.94	188.85	
Add 2% for overtime of hourly workers	3.09	3.70	NAP	
Total payroll cost per worker per day	157.69 ⁽³⁾	188.64 ⁽³⁾	188.85	
Worker shifts per year				
Hydraulic mining, underground only	18,644	6,048	4,800	
Allocated share of workers on common facilities	2,880	NAP	NAP	
Total Underground	21,524	6,048	4,800	
Annual payroll cost (000's)	\$3,394	\$1,141	\$ 906	\$5,441
Annual production clean coal (000's)				1,154
Payroll cost per ton, including all benefits except those levied on a "per ton" basis				\$4.71
Black Lung tax (included in Table 7-G)				1.00
UMWA Health and Welfare levy (included in Table 7)				1.695 ⁽³⁾
Total employee cost per ton				\$7.405

- (1) There are two components in total Black Lung costs: (1) hourly contribution, which is approximately 4% of direct wage; 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.
- (3) In addition to the wage cost per shift, there is a levy of \$1.695 per ton of coal to the UMWA Health and Welfare Fund.

TABLE 7-B
MATERIALS AND SUPPLIES

	<u>\$/Ton</u>
Bolts and trusses (see detail below)	0.08
Timber and jacks	0.10
Arch material (see detail next page)	0.22
Flume and pipe replacements: \$500,000 per year	0.43
Rock dust, 1.5 lbs. @ \$0.0125/lb.	0.02
Ventilation	
Stoppings and overcasts	0.15
Brattice	0.05
Bits, augers, tools	0.10
Explosives for occasional rock work	0.01
Diesel fuel and lubricants	0.10
Safety supplies, cap lamps, oxygen apparatus, etc.	0.50
Dewatering station supplies	0.20
Other general supplies and miscellaneous	0.50
Freight and drayage	<u>0.15</u>
Total Material and Supplies	\$2.61

Bolts and Trusses

Advance rate per shift in ventilation rooms: 18.ft.
 Bolts per row: 3
 Rows on 4 ft. centers
 Bolts per: shift: $18 \div 4 \times 3 = 13.5$
 year: 720×13.5 ; add 50% for maint. = 14,580

Cost of bolts:

Resin bolt:	\$1.55
Cartridge: 24¢ per ft. for 6 ft. =	1.44
Plate: 6" x 6" x 3/16" =	0.55
1/3 wood plank per bolt, 2" x 10" x 9' @ \$4 ea.	<u>1.33</u>
	\$4.87

Bolt material cost per year	\$71,005
Trusses: assume 300 per year @ \$60 =	<u>18,000</u>
Total bolt and truss material: per year:	\$89,005
per ton:	\$ 0.08

TABLE 7-B (continued)
MATERIALS AND SUPPLIES

Arches

Fluming rooms are arched on 3 ft. centers. An average 18 arches are estimated to be installed per shift behind the boom header, for a total of 4,320 arches per year.

The initial complement of arches assumed to be capitalized is sufficient for 4 rooms of 2,000 ft. long each, equivalent to 2,667 sets.

Aluminum arch, 3 piece, 8' x 11' struts and fastenings cost:
\$378 per set

Capital cost of initial complement: $2,667 \times \$378 = \$1,008,000$

Operating cost for replacement arches, blocking and lagging

Arches retreated per average monitor shift: 6

Arches assumed lost per: monitor shift; 10% = 0.6
year; $0.6 \times 720 = 432$

Replacement cost per year: $432 \times \$378 = \$163,296$

Blocking per year: $4,320 \times \$6.00 = 25,920$

Lagging per year: $4,320 \times \$16.00 = \underline{69,120}$

Total arch material cost per year $\$258,336$

Cost per year for arch material per ton =

$\$258,336 \div 1,154,000 \quad \$0.22$

TABLE 7-C

REPAIRS AND MAINTENANCE

Development of rooms for hydraulic mining does not require shuttle cars, feeder-breakers, or the complex equipment involved in longwall mining. Only three simplified continuous miners are required for driving the dual rooms and undercut rooms to support a production of over one million tons per year. Having no gathering head or conveyor, these machines are low in maintenance. Moreover, only about 17% of the total tonnage is produced by these machines, about 221,000 tons of raw coal per year, 74,000 tons per machine. Hydraulic retreat mining also uses very simple, long-life equipment, i.e., monitor, feeder-breaker, high pressure pumps, and standard dewatering equipment.

	<u>Cost per clean ton</u>
Repair and maintenance cost exclusive of preparation plant and other common facilities	0.30
Dewatering station	0.18
Machinery rebuilds; assume rebuilds cost \$200,000 each for each continuous mining machine at 5 year intervals	0.10
Rebuilds on other machinery	<u>0.16</u>
Total repair and maintenance	\$0.74

TABLE 7-D

POWER & OTHER COSTS

Excludes preparation plant and common facilities of
load-out, fan, shaft hoist, main line conveyor,
bathhouse, shop, warehouse, etc.

	<u>Cost Per Clean Ton</u>
<u>Power Costs:</u> (average cost per kwh is \$0.045 for 5 day/week operation and \$0.04 for steady 7-day operations)	
<u>Pumps:</u>	
Dewatering: 1,000 hp x .746 x 20 hrs/day x 365 x \$.04/kwh + 1,154,000 =	\$0.19
Low pressure: 1,000 hp x .746 x 8 hrs. actual utilization/day x 240 x \$.045/kwh + 1,154,000 =	0.06
High pressure: 5,000 hp x .746 x 24 hrs/day x 0.69 x 240 x \$.04/kwh + 1,154,000 =	0.51
Dewatering Station: 1,500 hp x .746 x 18 hrs/day x 240 x \$.045/kwh + 1,154,000 =	0.19
Development face equipment: (221,000 tons of raw development coal produced per average year, at 2.5 tons per minute per miner)	
3 continuous miner units x 300 hp each x .746 x 491 hrs per year x \$.045/kwh + 1,154,000 =	0.01
Main slope belt, 500 tph: 450 hp x .746 x 24 hrs/day x 60% utilization x 240 x \$.045/kwh + 1,154,000 =	0.05
Lights and miscellaneous, assume 50 kw x 24 hrs/day x 365 x \$.04/kwh + 1,154,000 =	<u>0.02</u>
Total Power Cost	\$1.03
<u>Other Direct Costs</u>	
Small vehicle use, private road maintenance, equipment rentals, utilities other than electric power, etc.	0.10

TABLE 7-E
COMMON FACILITIES

Coal Haulage

Coal produced from the hydraulic mine requires transportation by a common conveyor serving the entire west side to the raw coal surface stockpile at the preparation plant. This 4-mile conveyor runs at 600 fpm, has a capacity of 1500 tph, and is made up of 11 flights in series. Four men per shift are required to operate, service and clean up this system. Besides the hydraulic mine, this conveyor would serve the other producing seams and areas on the west side of the property. It is considered a shared facility, handling a total 1,921,000 tons per year of clean coal when full production is attained.

Full conveyor capacity is 1500 tph x 8 x 720 =	8,640,000 tpy
Total raw coal planned per year from the west side: 1,921,000 ÷ 0.86 =	2,234,000 tpy
Percent of full capacity utilization	
2,234,000 ÷ 8,640,000 = 26	
For utilization of full load power, use 35%	

The operating cost of this conveyor haul is estimated as follows:

	<u>Cost per year</u>
Labor, 4 men/shift x 720 shifts/year x \$157.69/man shift =	\$454,147
Power: 3900 x 0.35 x .746 x 8 x 720 x .05 =	<u>293,268</u>
Sub-Total	\$747,415
Add on 25% for repair materials, supplies, lubricants, etc.	<u>186,854</u>
Total	\$934,269
 Cost per ton clean coal: ÷ 1,921,000	 \$0.49

TABLE 7-E (continued)
COMMON FACILITIES

Ventilation

Currently being installed at Crandall Canyon to serve the entire west side reserves are two 1500 hp fans, each capable of 720,000 cfm at 8.1 inch water gauge. One fan will operate and one serve as a standby until more air is required; then both fans can be operated. The operating cost is estimated as follows:

	<u>Cost/Year</u>
Power: $1500 \times .746 \times 0.04 \times 24 \times 365 =$	\$ 392,098
Gas for heating winter air, at 1,000,000 cfm, 1,000,000 gals. of L.P. @ \$0.50 per gallon	<u>500,000</u>
Sub-Total	\$ 892,098
Add on 25% for service, maintenance, etc.	<u>223,025</u>
Total	\$1,115,123
Cost per ton: + 1,921,000	\$0.58

Bathhouse, Lamp and Self-Rescuer Service

This is a common service, assumed to be provided by the company's new facilities at Crandall Canyon. The charge against hydraulic mining is estimated to be equivalent to 1 man per shift, 5 days per week, with a 100% add-on for cap lamps, self-rescuers, power, supplies, building maintenance, etc.

	<u>Cost/Year</u>
3 men @ \$157.69 per day for 240 days per year	\$ 113,537
100% add-on for lamps, self-rescuers, power, etc.	<u>113,537</u>
Total	\$ 227,074
Cost per ton clean coal: + 1,154,000 =	\$0.20

TABLE 7-E (continued)
COMMON FACILITIES

Shaft and Hoist Service

This is another service which will be common to all production from the west side. The hoist is a counter balanced Koepe type, powered by a 600 hp motor; speed is up to 1,000 fpm; and load capability is 30 tons. Costs are estimated as follows:

	<u>Cost/Year</u>
1 hoistman per shift, 3 shifts per day, 240 days per year @ \$157.69	\$ 113,537
1 hoistman per shift for weekends and holidays, etc., at premium time of \$230 per man day; 125 days at \$230	<u>28,750</u>
Sub-Total	\$ 142,287
Assume 50% add-on for power, maintenance, supplies, etc.	<u>71,144</u>
Total	\$ 213,431
Cost per clean ton for west side coal: $\div 1,921,000 =$	\$0.11

Warehouse and Material Yard

These facilities are also to be provided at Crandall Canyon to serve all the west side coal.

Workers allocated to hydraulic mine:

Warehouse	4
Material yard	<u>2</u>
Total	6

	<u>Cost/year</u>
Labor and supervision, $6 \times 157.69 \times 240$	\$ 227,074
Add-on for materials, power, maintenance, etc., 40%	<u>90,829</u>
Total	\$ 317,903

Cost per clean ton: $\div 1,154,000$ \$0.28

TABLE 7-E (continued)
COMMON FACILITIES

Shop

Although the hydraulic mine's need for shop service is minimal, a central shop to serve all mining on the West side is planned for Crandall Canyon. It is assumed that the equivalent of 5 workers from the shop crew are charged to the hydraulic mine.

	<u>Cost/Year</u>
Labor and supervision, 5 x \$157.69 x 240 =	\$ 189,228
Add-on, 150% for parts	<u>283,842</u>
Total	\$ 473,070
Cost per clean ton: $\frac{473,070}{1,154,000} =$	\$0.41

Coal Preparation Incremental Costs, Excluding Labor

Technical Report Supplement, amended 3/4/81, estimates costs, excluding service of capital, for coal preparation, load out, accounting, engineering, general management and other common services to be \$2.12 per ton clean coal when full production of 5,352,000 tpy is reached. Escalating at 10% per year to update this cost to early 1983 levels, this estimate becomes: \$2.57

TABLE 7-F
GENERAL AND ADMINISTRATIVE

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

	<u>Cost/ton</u>
Auto expense	
Contributions	
District Office	
Ecology	
Consultants	
Legal and Central Accounting	
Marketing	
General, Administrative and Marketing Costs =	\$0.50

TABLE 7-G
TAXES, INSURANCE AND HYDRAULIC ROYALTY

Taxes and Insurance

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

Black Lung benefit	\$1.00
OSM tax	0.15
Real estate	0.10
Miscellaneous tax and fees	<u>0.06</u>
Sub-Total	... \$1.31

Insurance and Other Taxes (not included in above)

Technical Report Supplement as amended 3/4/81 in Exhibit 7-13 shows these costs affected by volume and ranging from \$0.19 to \$0.12 per ton during the years 1983 to 1989. For the purpose of this report, an average of \$0.15 is assumed to be representative of the period of operation of the hydraulic mine. \$0.15

Hydraulic Mining Royalty

A royalty to use B. C. Coal's proprietary hydraulic mining know-how is assumed at \$0.30

TABLE 7-H
FEDERAL COAL ROYALTIES

The mineral royalty on the coal land currently held by lease from the Federal Government is \$0.15 per ton. These leases come up for renewal in 1990, at which time the royalty will probably escalate to 8% of the f.o.r. sales price. The proposed new federal lease, assumed to be added to make this plan more feasible, embraces 18% of the coal to be mined hydraulically and would be leased at 8% of f.o.r. mine price. With due consideration to the current depressed market and the uncertainties of future prices, three different assumptions have been made for the selling price. It is believed the f.o.r. price for coal of this quality could vary from as low as \$25 to as much as \$35. In turn, the royalty would vary as follows:

		Sales Price		
		<u>\$25</u>	<u>\$30</u>	<u>\$35</u>
Present to 1989				
Cost per ton				
Existing leases		\$0.15	\$0.15	\$0.15
Additional lease (8%)		2.00	2.40	2.80
Tons per year (000s)				
Existing leases		946	946	946
Additional leases		208	208	208
Cost per year (\$000s)				
Existing leases		142	142	142
Additional leases		<u>416</u>	<u>499</u>	<u>582</u>
Total		558	641	724
Cost per ton		\$0.48	\$0.56	\$0.63
After existing leases renewed and royalty increased to 8%, assumed to be 1990 and thereafter				
Cost per ton		\$2.00	\$2.40	\$2.80

Capital Expenditures

The hydraulic mining operation is treated in this study as a separate cost center.

The feasibility has been examined in two ways with respect to capital costs:

1. Charging only the incremental capital necessary to establish the hydraulic mine, and determining its contribution to earnings.
2. Charging both the incremental capital and the appropriate share of capital for common facilities, both existing and now in progress of engineering and construction. These are termed "common facilities," see Table 8-E.

The capital expenditures for the hydraulic mine proper include:

- Initial capital expenditures to bring the hydraulic mine to full productive capacity, including the initial complement of aluminum arches, pipes and flumes.
- An operating loss for driving the access entries in the Sub-3 seam, i.e., six entries 6,000 ft. down slope at 9%, approximately along the 5 ft. isopach line. This drivage will produce 176,000 tons of clean coal on which it is estimated a loss will be incurred of \$10 per ton if the selling price is only \$25 per ton. If the selling price is \$30, the loss would be \$5 per ton; with a \$35 selling price, the loss would be zero. (With each \$1.00 difference in selling price, the cost will vary by 8¢ because of the Federal coal royalty.)

If, during the mine life, any equipment is replaced because of obsolescence, it is assumed that the resultant savings from improved performance would be sufficient to offset the purchase cost. Therefore, no accounting is made for such expenditures and resultant savings.

Table 8 summarizes all the capital expenditures and the following tables detail the capital costs:

- Table 8-A - Underground Pumping and Dewatering Station
- Table 8-B - Water System
- Table 8-C - Mining Equipment
- Table 8-D - Power, Ventilation and Compressed Air
- Table 8-E - Common Facilities

The capital expenditure schedule, Table 9, combines the initial capital expenditures from Table 8 and the schedule of development from Table 6, for use in the cash flow schedules, Table 10.

TABLE 8
CAPITAL EXPENDITURES
Hydraulic Mine Only

	<u>Total (\$000)</u>
Rock tunnels, 2 - 2500 ft. @ \$400/ft.	2,000
Brushing roadway and hard surfacing, 8500 ft. @ \$20/ft.	170
Underground pumping and dewatering station (Table 8-A)	
Machinery	1,146
Construction	1,750
Rock excavation	<u>670</u>
	3,566
Water System (Table 8-B)	
Pumps	1,311
Piping	882
Flumes	850
Mine dewatering borehole in Price Canyon	<u>33</u>
	3,076
Mining Equipment (Table 8-C)	
Development and retreat equipment	3,966
Material supply and transportation	830
Conveyors	<u>850</u>
General	<u>810</u>
	6,456
Ventilation raise: 230 ft.; Sub-3 to A-seam, 16'x16'	116
Power, ventilation & compressed air system (Table 8-D)	<u>1,170</u>
Sub-Total Equipment & Construction	16,554
Management and engineering (10%)	1,655
Other	500
Contingency (15%)	2,483
Operating capital (incremental)	2,000
Exploration	<u>150</u>
Total Initial Capital Expenditures for the Hydraulic Mine	23,342
Interest during construction	<u>1,500</u>
Total Underground Capital Before Loss Carried Forward	24,842

Capital Including Loss Carry Forward

Coal price f.o.r.	<u>\$25</u>	<u>\$30</u>	<u>\$35</u>
Total underground capital before loss carried forward	24,842	24,842	24,842
Loss carry forward for main slope development in Sub-3 seam	1,760	880	0
Total capital for hydraulic mine underground, excluding "common facilities"	\$26,602	\$25,722	\$24,842

NOTE: Expenditures include freight and sales taxes as appropriate.

TABLE 8-A
CAPITAL EXPENDITURES
Pumping and Dewatering Station

		(\$000)		
	<u>Units</u>	<u>Unit Cost</u>	<u>Total</u>	<u>Group Total</u>
Dewatering Machinery				
Flume diverter for emergency sump			10	
Slurry 2-way splitter			5	
S.D. screens: 3/4" opening, 4 x 16'	2	20	40	
2-way pulp distributor	2	10	20	
Sieve bends: 5 ft. wide, 45 degree, 3/4 mm openings	4	4	16	
Dewatering screens 6' x 12'	4	20	80	
Centrifuges: 150 tph each for dewatering 1-1/4" x 28 mesh; Wemco 1100 or equal	4	35	140	
Flocculant systems	2	10	20	
Underflow pumps	2	10	20	
Compact clarifier/thickeners: 35', Enviro-clear or equal	2	160	320	
Vacuum filters: 8 discs each, 10'6" diam., 1240 sq. ft., based on 85 lbs/sq. ft., peak load complete with vacuum pump, receiver, etc.	2	165	330	
Overhead cranes: 10 ton	2	40	80	
Decant gate		15	15	
Feeder for unloading decanting emergency sump		25	25	
Conveyor for feeding decanted material to main belt		25	25	
				1,146
Construction				
Concrete			150	
Steel: material and fabrication			300	
Erection of steel and equipment			400	
Piping: material and erection			300	
Electrical: transformers, breakers, centralized and automatic controls, wiring material and labor			600	
				1,750
Rock Excavation				
Dewatering raises:				
Dewatering, 80' x 40' diam., 2 @ \$67/yd.			499	
Emergency sump, 90' x 33' diam., 1 @ \$67/yd.			191	
				670
Total Dewatering Station				3,566

TABLE 8-B
CAPITAL EXPENDITURES
Water System - Pumps, Pipes, Flumes and Borehole

	(\$000)	
	<u>Total</u>	<u>Group Total</u>
Pumps		
Units for low pressure fluming water and dewatering: Ingersoll-Rand 4x11 DA-1-2, 5 stage, 1,000 hp, or equal; 3 @ \$147,000	441	
High pressure pumps, Ingersoll-Rand, 6x11 DA-2-3, 10 stage, 2500 hp; 2 @ \$235,000	470	
Pump installation (5 pumps), 1 lot	<u>400</u>	
		1,311
Pipes and Flumes		
Pipe		
Mine dewatering pipeline, 6", Schedule 80 (2500 ft. along main levels, plus 1700 ft. up the borehole to Price Canyon); total 4200 ft., @ \$10/ft.	42	
High pressure lines for monitor, 6", Schedule 80, A-53, 33,000 ft.*, @ \$10/ft.	330	
Couplings: Victaulic HP70ES; 1,775 @ \$26 ea.	46	
Controls: valves, fittings, 1 lot	400	
Compressed air pipe, 3", 17,000 ft. @ \$3.75/ft.	<u>64</u>	
		882
Flumes		
High molecular weight polyethylene lined, 17,000 ft. @ \$50/ft.	850	
		850
Borehole for dewatering mine, 10" diameter, 1700 ft., price includes installing 6", Schedule 80 pipe, welded joints, turnkey	33	<u>33</u>
Total Water System		3,076

*Includes a twin line, 9600 ft. along the main levels to the west, 3 - 2,000 ft. rooms, plus 1800 ft. for operating inventory.

TABLE 8-C
CAPITAL EXPENDITURES
Mining Equipment

	(\$000)	
	<u>Total</u>	<u>Group Total</u>
Development and Retreat Equipment*		
Continuous miner, drum head type, modified for hydraulic mining, with bolters	750	
Continuous Miner, drum head type, modified for hydraulic mining, without bolters	650	
Boom type roadheader	800	
Single boom roof bolter	100	
Aluminum air pipe, 3" diam., 10,000 ft. @ \$3.75/ft.	38	
Air tools, jacks, tools, etc.	60	
Aluminum yieldable arches, 2,667 @ \$378	1008	
Feeder-breaker-monitor units, air powered, 100 hp, 3 @ \$220,000	<u>660</u>	
		3,966
Equipment for Material Supply & Transportation		
Diesel mantrips, 4 @ \$50,000	200	
Diesel material supply trucks, 2 @ \$90,000	180	
Diesel maintenance truck	120	
Load-haul-dump (Eimco 913)	170	
Load-haul-dump (Eimco 911)	120	
"Kubota type" tractors, 2 @ \$20,000	<u>40</u>	
		830
Main Slope Conveyor		
Terminal groups, 42-inch; 2 @ \$175,000	350	
Structure and belting (8500 ft.)	<u>500</u>	
		850
General Equipment		
Tools	60	
Safety Equipment	150	
Spare Parts	<u>600</u>	
		<u>810</u>
Total Mining Equipment		6,456

*Equipment for the initial slope development in the Sub-3 seam will be furnished from the company's existing fleet of equipment, which is more than adequate.

TABLE 8-D
CAPITAL EXPENDITURES
Power, Ventilation and Compressed Air Systems

	(\$000)	
	<u>Total</u>	<u>Group Total</u>
Surface sub-station, 69/7.2 kv, 12,000 kva		350
Underground power distribution, 7200 v with couplers every 1,000 ft.		
Main line: 4/0, 15 kv insulated, 10,000' @ \$9.50/ft.	95	
Rooms: #2, 15 kv, insulated, 8,000 ft. @ \$6.00/ft.	48	
Telephone and communication system	<u>100</u>	243
Power centers with incoming and outgoing switch gear and circuit breakers		
2500 kva 7200/4160 v for high pressure pumps, (1 outlet), 2 @ \$50,000 each	100	
2500 kva 7200/440 v for dewatering and low pressure pumps (10 outlets) 2 @ \$80,000	160	
400 kva, 7200/440 v for development headings and conveyor, 4 @ \$17,000	68	
125 kva, 7200/440 v, 4 @ \$11,000	44	
H.V. switches	<u>60</u>	
		<u>432</u>
Total power system		1,025
Ventilation (fan is included in "Common Facilities")		
Overcasts for initial development		100
Compressed Air		
Rotary screw compressor, 150 hp, 600 cfm, for 150 psi, installed at dewatering station		<u>45</u>
Total Power, Ventilation and Compressed Air Systems		1,170

TABLE 8-E
CAPITAL FOR COMMON FACILITIES
Including Reserves, Crandall Canyon Complex, 4-mile Conveyor
and Preparation Plant

(\$000)

Reserves

The reserves engineered for the West side, excluding all coal over 12 ft. thick, are 68.2 million tons. Contained in this area is a 903 acre block of A-seam coal where the thickness of the seam is 12 ft. or more. The reserve of recoverable clean coal in this block under former mining plans is:

$$903 \times 43,560 \times 12 \div 25 \times 0.80 \times 0.86 = 13.0 \text{ million tons}$$

Of the total West side reserve this constitutes:

$$13.0 \div 68.2 = 19\%$$

For the purpose of this report then, 19% of the cost of West side reserves will be charged against the hydraulic mine.

The cost for the western reserve reported in Technical Report Supplement of 3/4/81 is:

Settlement and reorganization	22,298
Land	955
Mineral rights	16,282
Water rights	481
Total as of 3/4/81	40,016

Assuming escalation at 10% per year since 3/4/81, these costs become:

48,419

Cost of reserve charged to hydraulic mine

$$0.19 \times \$48,419,000 = \text{9,200}$$

Crandall Canyon Facilities

Fan, hoist, shafts, shop, warehouse and bathhouse

These facilities are near completion and are estimated to cost \$37,000,000. They are planned to service a total production from the West side of 1,921,000 tons per year of clean coal. If the hydraulic mine is implemented, it would contribute 1,154,000 tons or 60% of this total and, therefore, could be expected to bear this share of the capital cost.

$$60\% \text{ of } \$37,000,000 = \text{22,200}$$

West Side 4-mile Underground Main Line Belt Conveyor

This facility is estimated to have cost \$1,000,000 per mile in 1983 cost levels, for a total of \$4,000,000. The 60% share allocated to the hydraulic mine would be:

2,400

TABLE 8-E (continued)

(\$000)

Preparation Plant Load-Out

The Technical Report Supplement, dated 3/4/81, shows a capital cost for the preparation plant and load-out at \$56,675,000 to service an annual production of 5,353,000 tons per year by the year 1989. The 1,154,000 tons planned for hydraulic mining would constitute 21.6% of the output. Charging this proportion to the hydraulic mine would come to:

12,242

SUMMARY TABLE 8-E

Reserves, including "settlement and reorganization," land, mineral rights, and water rights 19% of \$48,419,000	9,200
Crandall Canyon facilities 60% of \$37,000,000	22,000
West side "4 mile conveyor to surface" 60% of \$4,000,000	2,400
Preparation plant, load-out, etc. 21.6% of \$56,675,000	<u>12,242</u>
Total Share of Common Facilities Chargeable to Hydraulic Mine	\$45,842

TABLE 9
CAPITAL EXPENDITURE SCHEDULE
(\$000)

YEAR:	1	2	3	4	TOTAL
Rock Tunnels	-	2,000	-	-	2,000
Roadway Brushing & Surfacing	170	-	-	-	170
Underground Dewatering and Pumping Station	-	-	-	-	-
Machinery	-	-	1,146	-	1,146
Construction	-	850	900	-	1,750
Rock excavation	-	670	-	-	670
Water System	-	-	-	-	-
Pumps	-	-	1,311	-	1,311
Pipes	-	-	882	-	822
Flumes	-	-	850	-	850
Dewatering Borehole	-	-	33	-	33
Mining Equipment	-	-	-	-	-
Development & Retreat	-	3,966	-	-	3,966
Material supply, transpor- tation and clean-up	-	830	-	-	830
Conveyors	850	-	-	-	850
General	-	405	405	-	810
Ventilation Raise	116	-	-	-	116
Power, Ventilation	-	-	-	-	-
Compressed Air	-	-	1,120	50	1,170
Sub-Total Capital	1,136	8,721	6,647	50	16,554
Management & Engineering, 10%	828	414	413	-	1,655
Sales Tax & Freight (not incl.)	100	300	100	-	500
Contingency (15%)	170	1,308	997	8	2,483
Operating Capital	500	500	500	500	2,000
Exploration	150	-	-	-	150
Total Initial Capital					
Expenditures	2,884	11,243	8,657	558	23,342
Shared Capital (Common Fac.)	45,842*	-	-	-	45,842
Total Capital Expenditures with Common Facilities	48,726	11,243	8,657	558	69,184

*Capital for Common Facilities, Table 8-E.

Cash Flow Forecast and Rate of Return

Cash Flow

The cash flow forecast is based on sales revenue, operating costs, capital expenditures and the development schedule. Three cases for sales prices are used: \$25, \$30 and \$35 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, the 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 for the hydraulic mine only is shown for a sales price of \$30 per ton (Table 9-B-1). Only summaries are shown for the other two sales prices.

- Production

The cash flow analysis presumes that the saleable tonnage produced will remain constant after retreat mining is begun in the fourth year.

- Operating Costs

For each of the three sales price cases, operating costs, as shown in Table 7, vary due to USGS royalties being based on 8% of the sales price after the present leases are renewed. Development costs for the access entries in the Sub-3 seams are estimated to be \$35 per ton, including Federal coal royalty, when the selling price is \$30 per ton. The fourth

year, when retreat mining begins, operating costs are estimated at:

Sales price	<u>\$25</u>	<u>\$30</u>	<u>\$35</u>
Operating cost	\$19.96	\$20.36	\$20.76

- 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:

5 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. Shared facilities, including existing capital, are depreciated over the life of the reserve. 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, being 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. The State of Utah allows depletion of 33-1/3% of net income..

- Income Taxes

The tax rates used in this cash flow are the Federal Tax Rate and the Utah State Tax Rate for 1982. The tax rates used are:

	<u>Federal</u>	<u>Utah</u>
1st \$25,000	17%	4%
2nd \$25,000	18%	4%
3rd \$25,000	30%	4%
4th \$25,000	40%	4%
Over \$100,000	46%	4%

Federal tax is applied to the income after deducting state tax for previous year.

- 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. The ITC is a credit against taxes. 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 8. 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: \$25, \$30 and \$35, and on two scenarios for each price:

1. Cash flows 10-A, 10-B, and 10-C do not reflect shared facilities.
2. Cash flows 10-D, 10-E and 10-F take into account, during the first year, the shared capital facilities (Table 8-E).

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

TABLE 10
SUMMARY OF CUMULATIVE CASH FLOWS
FOR A RANGE OF COAL SALES PRICES
(Without Allowing for Effects of Interest)
(\$000)

Year	Only Incremental Capital Charged Against Project			Including Allocation of Shared Facilities		
	Table 10-A \$25/ton	Table 10-B \$30/ton	Table 10-C \$35/ton	Table 10-D \$25/ton	Table 10-E \$30/ton	Table 10-F \$35/ton
1	< 4,574>	< 3,764>	< 2,954>	<50,416>	<49,606>	<48,796>
2	<15,817>	<15,007>	<14,197>	<61,659>	<60,849>	<60,039>
3	<26,143>	<23,664>	<21,185>	<71,985>	<69,506>	<67,027>
4	<23,408>	<16,674>	<10,072>	<69,250>	<62,516>	<55,782>
5	<17,396>	< 5,451>	4,337	<63,238>	<51,195>	<39,442>
6	<11,384>	5,555	16,467	<57,226>	<39,884>	<23,406>
7	< 5,371>	14,877	27,857	<51,213>	<28,807>	< 9,424>
8	560	22,549	38,585	<45,201>	<17,880>	1,999
9	6,418	30,233	49,323	<39,201>	< 7,606>	13,433
10	12,277	37,916	60,062	<33,303>	773	24,867
11	18,098	45,599	70,801	<27,405>	9,151	36,302
12	23,889	53,283	81,540	<21,681>	17,530	47,736
13	29,342	60,966	92,279	<16,157>	25,909	59,170
14	34,045	68,650	103,018	<10,734>	34,288	70,604
15	38,711	76,333	113,757	< 5,361>	42,667	82,039
16	43,358	84,017	124,496	< 12>	51,045	93,473
17	47,996	91,700	135,235	5,323	59,424	104,907
18	52,629	99,384	145,974	10,653	67,803	116,341
19	57,259	107,067	156,713	15,979	76,182	127,776
20	61,888	114,751	167,452	21,304	84,561	139,210
21	66,517	122,434	178,191	26,628	92,939	150,644
22	71,145	130,118	188,930	31,952	101,318	162,079
23	75,773	137,801	199,669	37,276	109,697	173,513
24	80,401	145,485	210,408	42,599	118,076	184,947
25	87,029	155,168	223,147	49,922	128,455	198,381

Internal Rate of Return

17%	30%	41%	4%	10%	14%
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Payback Period After 2 Years Construction

4.6 yrs	2.2 yrs	1.4 yrs	12.7 yrs	6.6 yrs	4.5 yrs
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TABLE 10-A

CASH FLOW FORECAST

(Only Incremental Capital Charged Against Project)

Sales Price: \$25/ton f.o.r. Mine
(\$000)

Year	Ship- ments, Tons (000)	Revenues (\$25.00 per ton)	Oper- ating Costs (\$19.79 ton) (1)	Oper- ating Profit <Loss>	Total Taxes	Oper- ating Cash Flow	Capital Expendi- tures	Net Cash Flow(2)	Cumu- lative Cash Flow
1	176	4,400	6,090	6,090	-0-	<1,690>	<2,884>	<4,574>	<4,574>
2	-0-	-0-	-0-	-0-	-0-	-0-	<11,243>	<11,243>	<15,817>
3	363	9,075	10,745	10,745	-0-	<1,670>	<8,657>	<10,327>	<26,143>
4	925	23,125	19,832	3,293	-0-	3,293	<558>	2,735	<23,408>
5	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<17,396>
6	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<11,384>
7	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<5,371>
8	1,154	28,850	22,838	6,012	<81>	5,931	-0-	5,931	560
9	1,154	28,850	22,838	6,012	<154>	5,859	-0-	5,859	6,418
10	1,154	28,850	22,838	6,012	<154>	5,859	-0-	5,859	12,277
11	1,154	28,850	22,838	6,012	<191>	5,821	-0-	5,821	18,098
12	1,154	28,850	22,838	6,012	<221>	5,791	-0-	5,791	23,889
13	1,154	28,850	22,838	6,012	<560>	5,452	-0-	5,452	29,342
14	1,154	28,850	22,838	6,012	<1,308>	4,704	-0-	4,704	34,045
15	1,154	28,850	22,838	6,012	<1,346>	4,666	-0-	4,666	38,711
16	1,154	28,850	22,838	6,012	<1,365>	4,647	-0-	4,647	43,358
17	1,154	28,850	22,838	6,012	<1,375>	4,637	-0-	4,637	47,996
18	1,154	28,850	22,838	6,012	<1,380>	4,633	-0-	4,633	52,629
19	1,154	28,850	22,838	6,012	<1,382>	4,630	-0-	4,630	57,259
20	1,154	28,850	22,838	6,012	<1,383>	4,629	-0-	4,629	61,888
21	1,154	28,850	22,838	6,012	<1,384>	4,629	-0-	4,629	66,517
22	1,154	28,850	22,838	6,012	<1,384>	4,628	-0-	4,628	71,145
23	1,154	28,850	22,838	6,012	<1,384>	4,628	-0-	4,628	75,773
24	1,154	28,850	22,838	6,012	<1,384>	4,628	-0-	4,628	80,401
25	1,154	28,850	22,838	6,012	<1,384>	4,628	2,000 ⁽³⁾	6,628	87,029

(1) Year 1, \$34.60/ton; year 3, \$29.60/ton; year 4, \$21.44/ton.

(2) Without allowing for effects of interest.

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 17%

Payback Period After 3.3 Years of Construction = 4.6 Years

TABLE 10-B
CASH FLOW FORECAST

(Only Incremental Capital Charged Against Project)
Sales Price: \$30/ton f.o.r. Mine
(\$000)

Year	Ship- ments, Tons (000)	Revenues (\$30.00 per ton)	Oper- ating Costs (\$20.19 ton) (1)	Oper- ating Profit <Loss>	Total Taxes	Oper- ating Cash Flow	Capital Expendi- tures	Net Cash Flow(2)	Cumu- lative Cash Flow
1	176	5,280	6,160	<880>	-0-	<880>	<2,884>	<3,764>	<3,764>
2	-0-	-0-	-0-	-0-	-0-	-0-	<11,243>	<11,243>	<15,007>
3	363	10,890	10,890	-0-	-0-	-0-	<8,657>	<8,657>	<23,664>
4	925	27,750	27,750	7,548	-0-	7,548	<558>	6,990	<16,674>
5	1,154	34,620	23,299	11,321	<98>	11,223	-0-	11,223	<5,451>
6	1,154	34,620	23,299	11,321	<315>	11,006	-0-	11,006	5,555
7	1,154	34,620	23,299	11,321	<1,999>	9,322	-0-	9,322	14,877
8	1,154	34,620	23,299	11,321	<3,649>	7,672	-0-	7,672	22,549
9	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	30,233
10	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	37,916
11	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	45,599
12	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	53,283
13	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	60,966
14	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	68,650
15	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	76,333
16	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	84,017
17	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	91,700
18	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	99,384
19	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	107,067
20	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	114,751
21	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	122,434
22	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	130,118
23	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	137,801
24	1,154	34,620	23,299	11,321	<3,637>	7,683	-0-	7,683	145,485
25	1,154	34,620	23,299	11,321	<3,637>	7,683	2,000 (3)	9,683	155,168

(1) Year 1, \$35.00/ton; year 3, \$30.00/ton; year 4, \$21.84/ton.

(2) Without allowing for effects of interest.

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 30%

Payback Period After 3.3 Years of Construction = 2.2 Years

TABLE 10-B-1 (Part I)
CASH FLOW FORECAST
(Only Incremental Capital Charged Against Project)
Sales Price: \$30/ton
(\$000)

Year:	1	2	3	4	5	6	7	8	9	10	11	12	13
Shipments, Clean Tons (000)	176	-0-	363	925	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154
Revenues @ \$30/ton	5,280	-0-	10,890	27,750	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620
Operating Cost @\$20.19/ton(1)	6,160	-0-	10,890	27,750	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299
Operating Profit <Loss>	<880>	-0-	-0-	7,548	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321
Taxes													
Depreciation	235	1,559	3,119	3,522	3,460	3,192	1,667	266	254	254	254	254	254
Federal Taxes													
Tax Loss Forward	-0-	1,643	3,202	7,410	6,159	2,611	746	-0-	-0-	-0-	-0-	-0-	-0-
Percentage Depletion	528	-0-	1,089	2,775	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462
50% Limit	-0-	-0-	-0-	-0-	851	2,716	4,345	5,399	5,386	5,386	5,386	5,386	5,386
Taxable Income	-0-	-0-	-0-	-0-	851	2,716	5,229	7,335	7,310	7,309	7,309	7,309	7,309
Income Tax	-0-	-0-	-0-	-0-	<371>	<1,229>	<2,385>	<3,354>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>
Investment Tax Credit	238	1,074	816	6	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
ITC Forward	-0-	238	1,313	2,128	2,134	1,775	644	-0-	-0-	-0-	-0-	-0-	-0-
ITC Limit	-0-	-0-	-0-	-0-	359	1,131	644	-0-	-0-	-0-	-0-	-0-	-0-
Total Federal Taxes	-0-	-0-	-0-	-0-	<12>	<98>	<1,741>	<3,354>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>
State Taxes (Utah)													
Tax Loss Forward	-0-	1,115	2,674	5,793	3,108	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Depletion Allowance	-0-	-0-	-0-	1,341	2,618	2,707	3,215	3,681	3,685	3,685	3,685	3,685	3,685
Taxable Income	-0-	-0-	-0-	-0-	2,135	5,422	6,439	7,373	7,381	7,381	7,381	7,381	7,381
Total State Taxes	-0-	-0-	-0-	-0-	<85>	<217>	<258>	<295>	<295>	<295>	<295>	<295>	<295>
Total Taxes	-0-	-0-	-0-	-0-	<98>	<315>	<1,999>	<3,649>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>
Operating Cash Flow	<880>	-0-	-0-	7,548	11,223	11,006	9,322	7,672	7,683	7,683	7,683	7,683	7,683
Capital Expenditures	<2,884>	<11,243>	<8,657>	<558>	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Net Cash Flow (2)	<3,764>	<11,243>	<8,657>	6,990	11,223	11,006	9,322	7,672	7,683	7,683	7,683	7,683	7,683
Cumulative Cash Flow	<3,764>	<15,007>	<23,664>	<16,674>	<5,451>	5,555	14,877	22,549	30,233	37,916	45,599	53,283	60,966

(1) Year 1, \$35.00/ton; year 3, \$30.00/ton; year 4, \$21.84/ton.

(2) Without allowing for effects of interest.

Internal Rate of Return = 30%

Payback Period After 3.3 Years of Construction = 2.2 Years

TABLE 10-B-1 (Part II)
CASH FLOW FORECAST
(Only Incremental Capital Charged Against Project)
Sales Price: \$30/ton
(\$000)

Year:	14	15	16	17	18	19	20	21	22	23	24	25
Shipments, Clean Tons (000)	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154	1,154
Revenues @ \$30/ton	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620	34,620
Operating Cost @\$20.19/ton(1)	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299	23,299
Operating Profit <Loss>	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321	11,321
Taxes												
Depreciation	254	254	254	254	254	254	254	254	254	254	254	254
Federal Taxes												
Tax Loss Forward	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Percentage Depletion	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462	3,462
50% Limit	5,386	5,386	5,386	5,386	5,386	5,386	5,386	5,386	5,386	5,386	5,386	5,386
Taxable Income	7,309	7,309	7,309	7,309	7,309	7,309	7,309	7,309	7,309	7,309	7,309	7,309
Income Tax	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>
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 Federal Taxes	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>	<3,342>
State Taxes (Utah)												
Tax Loss Forward	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-
Depletion Allowance	3,685	3,685	3,685	3,685	3,685	3,685	3,685	3,685	3,685	3,685	3,685	3,685
Taxable Income	7,381	7,381	7,381	7,381	7,381	7,381	7,381	7,381	7,381	7,381	7,381	7,381
Total State Taxes	<295>	<295>	<295>	<295>	<295>	<295>	<295>	<295>	<295>	<295>	<295>	<295>
Total Taxes	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>	<3,637>
Operating Cash Flow	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683
Capital Expenditures	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	-0-	2,000 (3)
Net Cash Flow (2)	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	7,683	9,683
Cumulative Cash Flow	68,650	76,333	84,017	91,700	99,384	107,067	114,751	122,434	130,118	137,801	145,485	155,168

(1) Year 1, \$35.00/ton; year 3, \$30.00/ton; year 4, \$21.84/ton.

(2) Without allowing for effects of interest.

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 30%

Payback Period After 3.3 Years of Construction = 2.2 Years

TABLE 10-C
CASH FLOW FORECAST

(Only Incremental Capital Charged Against Project)
Sales Price: \$35/ton f.o.r. Mine
(\$000)

Year	Ship- ments, Tons (000)	Revenues (\$35.00 per ton)	Oper- ating Costs (\$20.59 ton) (1)	Oper- ating Profit <Loss>	Total Taxes	Oper- ating Cash Flow	Capital Expendi- tures	Net Cash Flow(2)	Cumu- lative Cash Flow
1	176	6,160	6,230	<70>	-0-	<70>	<2,884>	<2,954>	<2,954>
2	-0-	-0-	-0-	-0-	-0-	-0-	<11,243>	<11,243>	<14,197>
3	363	12,705	11,035	1,670	-0-	1,670	<8,657>	<6,987>	<21,185>
4	925	32,375	20,572	20,572	<132>	11,671	<558>	11,113	<10,072>
5	1,154	40,390	23,761	16,629	<2,220>	14,409	-0-	14,409	4,337
6	1,154	40,390	23,761	16,629	<4,500>	12,129	-0-	12,129	16,467
7	1,154	40,390	23,761	16,629	<5,239>	11,391	-0-	11,391	27,857
8	1,154	40,390	23,761	16,629	<5,902>	10,728	-0-	10,728	38,585
9	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	49,323
10	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	60,062
11	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	70,801
12	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	81,540
13	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	92,279
14	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	103,018
15	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	113,757
16	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	124,496
17	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	135,235
18	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	145,974
19	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	156,713
20	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	167,452
21	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	178,191
22	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	188,930
23	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	199,669
24	1,154	40,390	23,761	16,629	<5,890>	10,739	-0-	10,739	210,408
25	1,154	40,390	23,761	16,629	<5,890>	10,739	2,000 ⁽³⁾	12,739	223,147

(1) Year 1, \$35.40/ton; year 3, \$30.40/ton; year 4, \$22.24/ton.

(2) Without allowing for effects of interest.

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 41%

Payback Period After 3.3 Years of Construction = 1.4 Years

TABLE 10-D

CASH FLOW FORECAST

(Including Allocation of Shared Facilities)

Sales Price: \$25/ton f.o.r. Mine

(\$000)

Year	Ship- ments, Tons (000)	Revenues (\$25.00 per ton)	Oper- ating Costs (\$19.79 ton) (1)	Oper- ating Profit <Loss>	Total Taxes	Oper- ating Cash Flow	Capital Expendi- tures	Net Cash Flow (2)	Cumu- lative Cash Flow
1	176	4,400	6,090	<1,690>	-0-	<1,690>	<48,726>	<50,416>	<50,416>
2	-0-	-0-	-0-	-0-	-0-	-0-	<11,243>	<11,243>	<61,659>
3	363	9,075	10,745	<1,670>	-0-	<1,670>	<8,657>	<10,327>	<71,985>
4	925	23,125	19,832	3,293	-0-	3,293	<558>	2,735	<69,250>
5	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<63,238>
6	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<57,226>
7	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<51,213>
8	1,154	28,850	22,838	6,012	-0-	6,012	-0-	6,012	<45,201>
9	1,154	28,850	22,838	6,012	12	6,000	-0-	6,000	<39,201>
10	1,154	28,850	22,838	6,012	115	5,898	-0-	5,898	<33,303>
11	1,154	28,850	22,838	6,012	115	5,898	-0-	5,898	<27,405>
12	1,154	28,850	22,838	6,012	288	5,724	-0-	5,724	<21,681>
13	1,154	28,850	22,838	6,012	489	5,524	-0-	5,524	<16,157>
14	1,154	28,850	22,838	6,012	589	5,424	-0-	5,424	<10,734>
15	1,154	28,850	22,838	6,012	639	5,373	-0-	5,373	<5,361>
16	1,154	28,850	22,838	6,012	664	5,348	-0-	5,348	<12>
17	1,154	28,850	22,838	6,012	676	5,336	-0-	5,336	5,323
18	1,154	28,850	22,838	6,012	683	5,330	-0-	5,330	10,653
19	1,154	28,850	22,838	6,012	686	5,326	-0-	5,326	15,979
20	1,154	28,850	22,838	6,012	687	5,325	-0-	5,325	21,304
21	1,154	28,850	22,838	6,012	688	5,324	-0-	5,324	26,628
22	1,154	28,850	22,838	6,012	689	5,324	-0-	5,324	31,952
23	1,154	28,850	22,838	6,012	689	5,324	-0-	5,324	37,276
24	1,154	28,850	22,838	6,012	689	5,323	-0-	5,323	42,599
25	1,154	28,850	22,838	6,012	689	5,323	<2,000> (3)	7,323	49,922

(1) Year 1, \$34.60/ton; year 3, \$29.60/ton; year 4, \$21.44/ton.

(2) Without allowing for effects of interest.

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 4%

Payback Period After 3.3 Years of Construction = 12.7 Years

TABLE 10-E

CASH FLOW FORECAST

(Including Allocation of Shared Facilities)

Sales Price: \$30/ton f.o.r. Mine
(\$000)

Year	Ship- ments, Tons (000)	Revenues (\$30.00 per ton)	Oper- ating Costs (\$20.19 ton) (1)	Oper- ating Profit <Loss>	Total Taxes	Oper- ating Cash Flow	Capital Expendi- tures	Net Cash Flow(2)	Cumu- lative Cash Flow
1	176	5,280	6,160	<880>	-0-	<880>	<48,726>	<49,606>	<49,606>
2	-0-	-0-	-0-	-0-	-0-	-0-	<11,243>	<11,243>	<60,849>
3	363	10,890	10,890	-0-	-0-	-0-	<8,657>	<8,657>	<69,506>
4	925	27,750	20,202	7,548	-0-	7,548	<558>	6,990	<62,516>
5	1,154	34,620	23,299	11,321	-0-	11,321	-0-	11,321	<51,195>
6	1,154	34,620	23,299	11,321	<9>	11,312	-0-	11,312	<39,884>
7	1,154	34,620	23,299	11,321	<244>	11,077	-0-	11,077	<28,807>
8	1,154	34,620	23,299	11,321	<394>	10,927	-0-	10,927	<17,880>
9	1,154	34,620	23,299	11,321	<1,047>	10,274	-0-	10,274	<7,606>
10	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	773
11	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	9,151
12	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	17,530
13	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	25,909
14	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	34,288
15	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	42,667
16	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	51,045
17	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	59,424
18	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	67,803
19	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	76,182
20	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	84,561
21	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	92,939
22	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	101,318
23	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	109,697
24	1,154	34,620	23,299	11,321	<2,942>	8,379	-0-	8,379	118,076
25	1,154	34,620	23,299	11,321	<2,942>	8,379	<2,000> (3)	10,379	128,455

(1) Year 1, \$35.00/ton; year 3, \$30.00/ton; year 4, \$21.84/ton.

(2) Without allowing for effects of interest..

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 10%

Payback Period After 3.3 Years of Construction = 6.6 Years

TABLE 10-F

CASH FLOW FORECAST

(Including Allocation of Shared Facilities)

Sales Price: \$35/ton f.o.r. Mine

(\$000)

Year	Ship- ments, Tons (000)	Revenues (\$35.00 per ton)	Oper- ating Costs (\$20.19 ton) (1)	Oper- ating Profit <Loss>	Total Taxes	Oper- ating Cash Flow	Capital Expendi- tures	Net Cash Flow(2)	Cumu- lative Cash Flow
1	176	6,160	6,230	<70>	-0-	<70>	<48,726>	<48,796>	<48,796>
2	-0-	-0-	-0-	-0-	-0-	-0-	<11,243>	<11,243>	<60,039>
3	363	12,705	11,035	1,670	-0-	1,670	<8,657>	<6,987>	<67,027>
4	925	32,375	20,572	11,803	-0-	11,803	<558>	<11,245>	<55,782>
5	1,154	40,390	23,761	16,629	<289>	16,340	-0-	16,340	<39,442>
6	1,154	40,390	23,761	16,629	<594>	16,035	-0-	16,035	<23,406>
7	1,154	40,390	23,761	16,629	<2,647>	2,647	-0-	13,982	<9,424>
8	1,154	40,390	23,761	16,629	<5,206>	11,423	-0-	11,423	1,999
9	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	13,433
10	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	24,867
11	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	36,302
12	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	47,736
13	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	59,170
14	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	70,604
15	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	82,039
16	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	93,473
17	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	104,907
18	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	116,341
19	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	127,776
20	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	139,210
21	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	150,644
22	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	162,079
23	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	173,513
24	1,154	40,390	23,761	16,629	<5,195>	11,434	-0-	11,434	184,947
25	1,154	40,390	23,761	16,629	<5,195>	11,434	<2,000> (3)	13,434	198,381

(1) Year 1, \$35.40/ton; year 3, \$30.40/ton; year 4, \$22.24/ton.

(2) Without allowing for effects of interest.

(3) Year 25, \$2,000,000 operating capital recovered.

Internal Rate of Return = 14%

Payback Period After 3.3 Years of Construction = 4.5 Years

MARKET

The production of Utah coal has grown during the last 10 years, from 4.8 million tons in 1972 to 16.9 million tons in 1982.

Notwithstanding this healthy growth, it is not achieving the rates of increase anticipated in the years following the first oil shock. For example, a popular forecast made in 1976 by the COAL AGE publication for Utah coal production was 29.5 million tons by 1985. The market for western U.S. coal is currently depressed because of a number of factors:

- (1) The depressed world economic conditions.
- (2) Overanticipation of the market by forecasters, which caused the mining companies to expand vigorously, creating an over-supply of coal.
- (3) The drop in growth of electric power demand over the last several years as compared to the previous growth rate of the 60's and 70's.
- (4) The enactment of the Clean Air Act, which requires new power plants to provide the Best Air Control Technology (BACT) on new power plants regardless of whether high or low sulfur coal is used, thus reducing to some extent the premium value of low sulfur coals such as are typical in Utah.
- (5) The reduction in strikes in Australia and, in turn, the increasing supplies of Australian coal; Pacific Rim customers feel less compelled to seek new sources, such as western U.S. coal.
- (6) The growing weakness against the U.S. dollar of the currencies in South Africa, Australia and Canada which of late have virtually shut western U.S. coal out of the Pacific Rim markets. As recently as 1981 this market took over 5 million tons of western U.S. coal.
- (7) High rail freight costs to the Pacific ports.
- (8) Inadequate port facilities and high port rates.
- (9) The closure of Kaiser Steel's primary steelmaking facilities at Fontana, including the coke ovens.

- (10) The unusually high availability of hydropower in western states caused by the heavy precipitation in the 1982-83 season.
- (11) Obstruction by the environmentalists to siting new coal burning power plants in Utah and California.

For the above reasons and perhaps others, the demand for Utah coal has dropped from 16.9 million tons clean coal in 1982 to an estimated 12 to 13 million tons in 1983. As a result, six coal mines in Utah have recently closed down and currently there are nearly 2,300 miners out of work, not counting about 500 idled because of the blockage of the railroad by the Thistle mudslide.

However, the long range picture is much different. The world recession is not likely to last forever. Already economic indices are pointing upward in the U.S. It is reasonable to assume there will not be a surge in availability of hydropower every year; the U.S. dollar may not forever stay so strong against the currencies of countries competing in the international coal market, etc., etc.

To look at a more fundamental trend, consider the supply and prices of gas vs. coal. The major utilities in California, i.e., Pacific Gas & Electric and Southern California Edison, are both fueling their thermal power plants primarily with gas. In the case of PG&E, 48% of the total power generated in their own system is from gas. In their last publication, PG&E's gas costs were \$5.07 per million btus. Oil accounted for a very small percentage of their fuel and cost \$6.52 per million btu. The gas consumed by just this one utility, PG&E, was the equivalent of 33,667,000 barrels of oil, or about 8 million tons of coal.

During this current depressed market, the TEX REPORT in Japan reported two Utah coal companies as quoting spot steam coal at \$47 per long ton f.o.b. ship. Backing off port costs and converting this to a price per million btu works out to \$1.70 per

million btu. These are very depressed prices and probably cannot be sustained for a long period. More representative of the long term market for Utah steam coal are the prices quoted by three companies, within a 50 mile trucking distance, to supply the Hunter plant of Utah Power & Light. These prices were obtained by Kaiser Engineers, a division of Raymond International, in a study done for Utah Associated Municipal Power Systems (UAMPS) and Deseret Generation and Transmission Cooperative. The sources and delivered starting prices were published April-12, 1983 in the Deseret News of Salt Lake City. With this data it is easy to estimate the transportation cost and derive the mine mouth prices quoted as follows:

	<u>Consol's Emery Mine</u>	<u>Getty Oil's Plateau Mine</u>	<u>Tower Resources</u>
Quoted delivered price at Hunter plant per short ton	\$33.00	\$30.38	\$29.68
Trucking distance, miles	30	40	45
Estimated trucking cost	\$ 2.25	\$ 3.20	\$ 3.60
FOB mine price	\$30.75	\$27.18	\$26.08
Delivered prices/mil. btus	\$ 1.36	\$ 1.31	\$ 1.31
Btus per ton in millions	24.26	23.19	22.65
Btus per lb.	12,130	11,595	11,325

Getty's Plateau Mine is the only mine on rail, therefore using this as a base, and adjusting for Price River Coal's higher btu gives a current market price for Price River at \$29.30 FOR.

It is estimated that, for a large volume, long term shipment to California, the rail rate would be in the range of \$17, making the total delivered price \$46.30 or \$1.85 per million btus. For conservatism use \$2.00 per million btus. Thus, Californians pay approximately three times the price per million btus for natural

Emery Counties, Utah. The Price River Coal property is a prime example of a premier coal property, now idle for lack of demand, but at the same time ready with most of the production facilities in place, to respond quickly as the market returns.

It should be mentioned here that all of the prices discussed in the foregoing are for steam coal. Metallurgical coal normally draws somewhat higher prices. The coal from this property is primarily a premium grade steam coal, but it also has weak coking properties (FSI 1-1/2 to 3). When normal economic times return, part of the production will probably be shipped to the metallurgical market.

In the future there will be a need and a healthy market for the coal proposed for production in this report. For the purpose of this study, we have used a range of prices, \$25, \$30 and \$35 per short ton, to reflect the present market conditions, the uncertainty of timing for the future market growth, and prices more realistic and sustainable.

SOCIOECONOMICS

Introduction

The hydraulic mining proposed for the the West side of Price River Coal Company's property would have minimum socioeconomic consequences since:

- The mine will only be an extension of present mine workings.
- Proposed longwall production will be replaced with hydraulic production.
- No new surface facilities will be required.

An overall general assessment of the socioeconomic conditions of the Castlegate site includes:

- labor requirements;
- the availability of trade contractors, construction materials, and supplies;
- the infrastructure conditions;
- the environmental conditions and control measures required;
- the housing and transportation of personnel;
- governmental regulations and permitting.

The Carbon and Emery County areas have historically been heavily dependent on coal mining. The region's economic health improves and declines with the state of the market for coal. In the late 19th and early 20th centuries, the Carbon-Emery economy was strong while the coal market thrived. From the 1950's through the early 1970's, however, the market for coal was largely taken over by oil and gas. With the emergence of OPEC in 1973 and the subsequent oil price increases, coal regained its competitiveness as an energy source and, until the recent recession, the high energy output, low polluting Utah coal, such as that from the Castlegate, has been in high demand. The two county area has benefited by way of jobs and income, and the population has grown.

The site is located 10 miles northwest of Price, Utah, approximately 2 miles west of the town of Helper and 110 miles southeast of Salt Lake City. The property is bisected by the main east-west line of the Denver and Rio Grande Railroad and by U.S. Highway 6 leading from Price to Salt Lake City.

The cities of Helper and Price are the population centers of the area. The population of Price was estimated at over 9,400 persons in 1980, and Helper was estimated at over 2,700 persons.

At this time, Price River Coal Company intends to continue its underground mining operations, developing the several mineable coal seams within its 27,393 acre reserves. The mining activities which were reactivated in 1971, were conducted under contract for the owner by the corporate entity, Braztah, a subsidiary of McCulloch Oil. As of December 1, 1979, American Electric Power, by way of their subsidiary, Price River Coal Company, took over the direct operating role.

Production for 1982 from the existing No. 3 and No. 5 mines, operating in the western portion of the property was 1.1678 million tons of clean coal; 1983's production is projected at 0.329 million. The planning and facilities constructed to-date are for an ultimate production of 6 million tons per year.

Labor Requirements

There were approximately 600 employees in 1982 at the Price River Coal Company complex, of which nearly 400 were UMWA (United Mine Workers of America). Without incorporating hydraulic mining into the plans, the company anticipates an additional 750 employees will be required to reach a production of 5.5 million tons per year.

The incorporation of hydraulic mining should be considered as a replacement of method rather than an addition. The productivity of hydraulic mining, for the thick coal conditions in the site selected, is estimated to be higher than for continuous miners and longwalls. Therefore, it is projected that, with hydraulic mining, the number of additional employees required would be reduced by more than 100; thus, total employees to be added would be about 650 rather than 750.

Since Price River Coal Company's increase in operations is only one of several proposed coal projects in the Central Utah Region, total Utah coal production could rise from 14.3 million tons per year in 1981 and 16.9 million tons in 1982 to between 18.8 and 22.45 million tons in 1990 according to a study made in 1981 by Abt/west. Most of this ~~production~~ would be from Carbon and Emery Counties.

Based on the coal production projections of 24 million tons per year by 1990, along with other energy related projects, Abt/West estimated population for Carbon and Emery Counties as follows:

	<u>1980</u>	<u>Abt/West's Forecast 1990* ..</u>
Population:		
Carbon County	22,179	31,400
Emery County	11,451	18,400
Price City	9,407	12,560
Helper City	2,724	4,396
Mining Employment:		9,039

These projections, made before the recession of 1982 and 1983, predicted that local population would grow at a rapid pace. Carbon County's population was projected to increase nearly 42% between 1980 and 1990 with a projected coal production increase of 40%, which is in the same proportion. It was estimated that over this ten year period, mining would account for 4 out of every 10 new jobs in the region. By 1990, it was projected that 3,514 new mining positions would be created. (The projection did

not include the 750 new employees then anticipated by Price River Coal Co.)

However, in light of the present economic environment, these expansionist forecasts of past years now appear to be far too optimistic, if not completely unrealistic. As of this writing, May 1983, there were 2,810 miners unemployed in Carbon and Emery counties. Even after the opening up of the rail line blockage at Thistle, Utah, the unemployed will still be in the range of 2300, according to the manager of the State Office for Job Service in Price, Utah. The Job Service office advises that mining employment for Carbon-Emery Counties peaked in June 1982 at 5,110. By May 1983, employment at the mines was down to 2300, which they estimate to be nearly 500 lower than would otherwise be the case due to the blockage of the D&RG Railroad by the mudslide at Thistle. Any increase in employment would be extremely beneficial to this very depressed area. By producing coal cheaper, hydraulic mining could increase the competitive power of coal and, in turn, both the market and the employment.

The hydraulic mining scenario for the Castlegate site, as presented in this report, anticipates production at 1,154,000 clean tons of coal, approximately the same as 1982's production. The hydraulic mine would add a total of 173 personnel. Therefore, approximately 22 percent of the work force would produce nearly 50 percent of the tonnage.

While a tight labor market in the future looks to be a rather remote possibility as of this writing, should it ever occur, the introduction of hydraulic mining would provide a competitive edge to Price River Coal Company. The innovative and safety aspects of hydraulic mining would attract labor.

Availability of Trade Contractors, Construction Materials, and Supplies

Locally there are 43 general contractors operating in the Carbon-Emery area. Due to the location of the project, 110 miles from Salt Lake City, 50 miles from Spanish Fork, and 60 miles from Provo, trade contractors that are not locally available can be supplemented by contractors in the Utah-Salt Lake County areas.

Construction will consist of underground rock work and will extend for three years from start-up, including driving main entries in the Sub-3 Seam. The peak number of construction employees on-site is estimated to be less than 50 employees during the construction period.

Construction materials, such as structural steel, electrical wire, mechanical equipment and specialty items, will be brought in from outside the area. Materials such as concrete, sand and gravel are available locally in the Carbon-Emery area. Several mining equipment manufacturers have local warehouses. The lead time, including permit approvals and the initial period driving in the Sub-3 Seam, should allow ample time to acquire major equipment. Hydraulic mining equipment could be sized and specified as much as two years in advance of delivery, allowing ample time for design of modifications where needed, e.g., continuous miners.

Typical mining supplies for present underground operations are already warehoused at the site; this inventory could be expanded for the proposed operation. Only the additional items specifically required by hydraulic mining would be added to inventories. These include: fluming pans, high pressure pipe, and parts for the monitor, dewatering equipment, and roadheader.

Infrastructure Conditions

The historical aspects of coal mining in the Carbon-Emery County area, as well as future population and employment trends, as discussed, concern State and County Planners. To date, the growth of the coal and energy industries in the area has been accommodated without many serious problems. However, if the population were to again grow, social services, health care, police, education, water and waste water systems, and medical services would all require similar expansions to keep pace with this growth. The following is an overview of the existing infrastructure of Price and Helper, which will be affected by development of additional coal mining operations.

SUMMARY OF PRESENT INFRASTRUCTURE CONDITIONS

	<u>Price</u>	<u>Helper</u>
Health Care:		
Hospital Beds	70	0
Physicians	23	2
Nurses	48	21
Dentists	11	1
Fire Department:	<u>Volunteer</u>	<u>Volunteer</u>
	Needs 4 volunteers and new station	Needs equipment and station
Law Enforcement:	12 full-time officers; needs new jail	5 full-time officers and uses County Sheriff assistance; needs 2 officers and 1 vehicle
Water System:	Price River Water Improvement District	
	Capacity: 18 mm gallons per day 24,000 persons	
	Upgrade: 1980-1985	
Solid Waste:	Public & Private Adequate	Public Adequate

	<u>Price</u>	<u>Helper</u>
Social Services:	Job Service Office Sr. Citizen Center Utah Department Social Services LDS Church Programs	Job Service Office Sr. Citizen Center -- LDS Church Programs
Commercial Services:	Regional Center	Minimal Services; depends on Price
Utilities:	Gas (adequate) Electric (adequate) Telephone (hook-up delays)	
Education:	<u>Public</u>	<u>Private</u>
Elementary		
No.	3	1
Enrollment	1250	285
Capacity	1216	NA
Jr. High School		
No.	1	0
Enrollment	600	250
Capacity	618	235
High School		
No.	1	0
Enrollment	900	Price
Capacity	833	NAP
College		
No.	1	0
Enrollment	970	Price
Capacity	NA	NAP
Vocational		
No.	1	0
Enrollment	25	0
Capacity	NA	0

Price River Coal Company provides water treatment facilities of 450 gpm or 648,000 gpd, which satisfies its coal mine and plant requirements for culinary water. The company controls additional water rights to supply future mining operations, and these rights would also be available to supplement additional employee housing if required. Adequate sewage facilities are provided by Price River Water Improvement District for the present mine facilities.

Environmental Conditions and Control Measures

Since the hydraulic mining plan involves no additional surface facilities, no environmental problems will be encountered, except as follows:

- Hydrologic balance - Indications to date are that the overlying bedrock formations are of low permeability and, with existing workings undermining the area, the effects of mining should be minimal on the underground hydrologic balance.
- Subsidence - Subsidence monitoring will be conducted over the area of mining. Generally, with more uniform and more complete extraction, ~~surface~~ lands may be ~~subsid~~ed as much as 12 feet without deleterious affects. If the surface is brought down uniformly in broad areas, it is often difficult for an untrained investigator to notice the effects. The overlying surface lands contain wildlife habitat but few structures.
- Air Quality - The additional particulate emissions from added coal production should be minimal as the only sources will be belt conveyors and transfer points where best available dust control measures will be employed.

Housing and Transportation of Personnel

Approximately 173 permanent workers will be employed at the facility. The company will probably meet these labor requirements by calling back employees now on layoff and/or recruiting from unemployed surplus labor in the area, and from in-migrating residents. These workers are expected to reside in Carbon County.

Housing

There is currently a surplus of housing in the area. The real estate market is depressed and prices have leveled out or even gone down in some cases. However, if rapid expansion should occur again, an area of concern in hiring in-migrating workers may become housing. Although the 1980 census indicates a total of 11,895 dwelling units in Carbon and Emery Counties, an increase of 63 percent since 1970, the Southeastern Utah Association of Governments (SEUAOG) reports the vacancy rate before the recession was very low and there were a large number of inadequate units in need of improvement or replacement. Given the current high interest rates, not only will such improvements be unlikely, but future residents could have difficulty obtaining mortgages. Before the recession it was estimated by Abt/West that housing needs are as follows for 1990.

County	1980	1990
Carbon	8,192	9,544
Emery	3,703	5,593

Should this need for additional housing actually materilize, there are numerous building contractors and construction workers anxious to supply new housing at competitive prices.

Generally, construction employees choose to locate in mobile homes and trailers or other temporary housing, e.g. motels. There are a number of suitable facilities now available in Price.

In an earlier submission to the environmental agency, Price River Coal Company proposed a plan to alleviate any possible housing shortages that might develop. In this plan the company would provide water rights and land to a developer, who would construct as many housing units as needed in a subdivision between Kenilworth and Spring Glen. This area is within two miles of the proposed mine. The road into this area would require upgrading,

possibly along the abandoned rail grade. A crossing for the main line of the D&RGW railroad would need to be constructed, and the sewage treatment facilities would need to be expanded. Such a project would provide adequate housing for employees and would assist in the recruitment of employees.

Transportation

Highway 6 provides the main access to the mine from Price and Helper. Because of the increasing volume of traffic, the problem of transportation along Highway 6 would be lessened by hydraulic mining because it is less labor intensive than other methods.

Governmental Regulations and Permitting

Permitting of the hydraulic mine could require a modification of the existing permit for the Castlegate A-Seam. This would involve the following:

1. Informal discussion with the Utah Division of Oil, Gas and Mining (DOGM). The Division has been granted primacy for enforcement of the Surface Mining Control and Reclamation Act (SMCRA) through a cooperative agreement with the Office of Surface Mining (OSM).
2. Predesign Conference with the Division of Environmental Health (DOH). Potential development of the mine must be accomplished in accordance with this Division's concurrence. The development is presented in the conference by displaying plans for air and water, and for any other concerns. Other State Divisions, such as the Division of Water Rights, are represented at the conference. The permit modifications and possible mitigation of potential environmental hazards are discussed prior to final submission of plans by the applicant.

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