BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

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23 March 1981

Dr. R. L. Evans
Director, Conservation & Technology Division
Ministry of Energy, Mines & Petroleum Resources
Suite 2006, 1177 West Hastings Street
Vancouver, B. C.
V6E 2L7

Dear Dr. Evans:

Hat Creek Coal Liquefaction Project
Prefeasibility Study Mining

We are pleased to submit this report on the mining aspects of the Prefeasibility Study of a Coal Liquefaction Plant and Thermal Powerplant utilizing Hat Creek coal.

We consider that the technical and economic results presented in this report are sufficiently encouraging to justify the necessary additional drilling and technical studies that would be required for large scale utilization of Hat Creek coal.

It must be emphasized that early planning and coordination is essential if the economic efficiency of integrating a Coal Liquefaction Plant with B. C. Hydro's proposed 2000 MW Powerplant is to be achieved.

The following members of the Mining Department contributed to the completion of this assignment:

W. C. Fothergill, P.Eng.	B. Dutt, P.Geol.
W. E. Meeks, P.Eng.	H. Kim, P.Geol.
N. P. Crocker, E.I.T.	A. W. Penner
S. S. K. Wong	R. A. Kawasaki

The execution of this study has proved to be an interesting and challenging assignment. The project presents an unusual collection of technical and practical problems that will require the application of a broad range of engineering and operating skills integrated by thorough planning to ensure a successful development.

Respectfully submitted,

J. J. Fitzpatrick, P.Eng.
Manager
Mining Department

BRITISH COLUMBIA

ENERGY DEVELOPMENT AGENCY

HAT CREEK

COAL LIQUEFACTION PROJECT

PREFEASIBILITY STUDY

MINING

BRITISH COLUMBIA HYDRO & POWER AUTHORITY

MINING DEPARTMENT

MARCH 1981

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HAT CREEK COAL LIQUEFACTION PROJECT

PREFEASIBILITY STUDY

MINING STUDY

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SECTION 1

INTRODUCTION

1.1 BACKGROUND

This report presents the results of a prefeasibility study of the mining of the Hat Creek No. 1 and No. 2 Coal Deposits to supply coal to British Columbia Hydro and Power Authority's proposed 2000 MW (net) Thermal Powerplant for 35 years and for 30 years to a Coal Liquefaction Plant to produce approximately 8000 m^3 per day of liquid products. The study was performed for the Energy Development Agency, at the request of the Ministry of Energy, Mines and Petroleum Resources of British Columbia, by the Mining Department of British Columbia Hydro and Power Authority.

The study was executed in accordance with a proposal submitted to the Ministry in September 1980, in response to terms of reference issued by the Director, Conservation and Technology Division.

The purpose of the study was to provide mining input to the overall technical, environmental and economic evaluation of the proposed Coal Liquefaction Plant. The study scope included the provision of information on mining systems, equipment, facilities and manpower as well as preliminary capital and operating costs. These results will be integrated into the studies being conducted on other aspects of the project. A program and schedule for the development of the project has been prepared.

Parallel studies were conducted as follows:

(1) The Coal Liquefaction Plant by Fluor Canada Ltd.;

(2) Project Support Facilities by B. C. Hydro's Thermal Generation Projects Division;

(3) Product Transportation by Ministry of Transport and Highways of British Columbia.

(1) Environmental and Social Analysis by B. C. Research;

(2) Economic Evaluation by the Ministry of Industry and Small Business Development of British Columbia.

The cooperation with these groups has been mutually beneficial and is acknowledged.

This study draws heavily upon the extensive studies previously undertaken by the Mining Department and their consultants in evaluating and planning the development of the Hat Creek Coal Deposits. Specific aspects of this study have been carried out through assignments to:

(1) Golder Associates: geotechnics and hydrology;

(2) B. C. Hydro Hydroelectric Generation Projects Division: creek diversion;

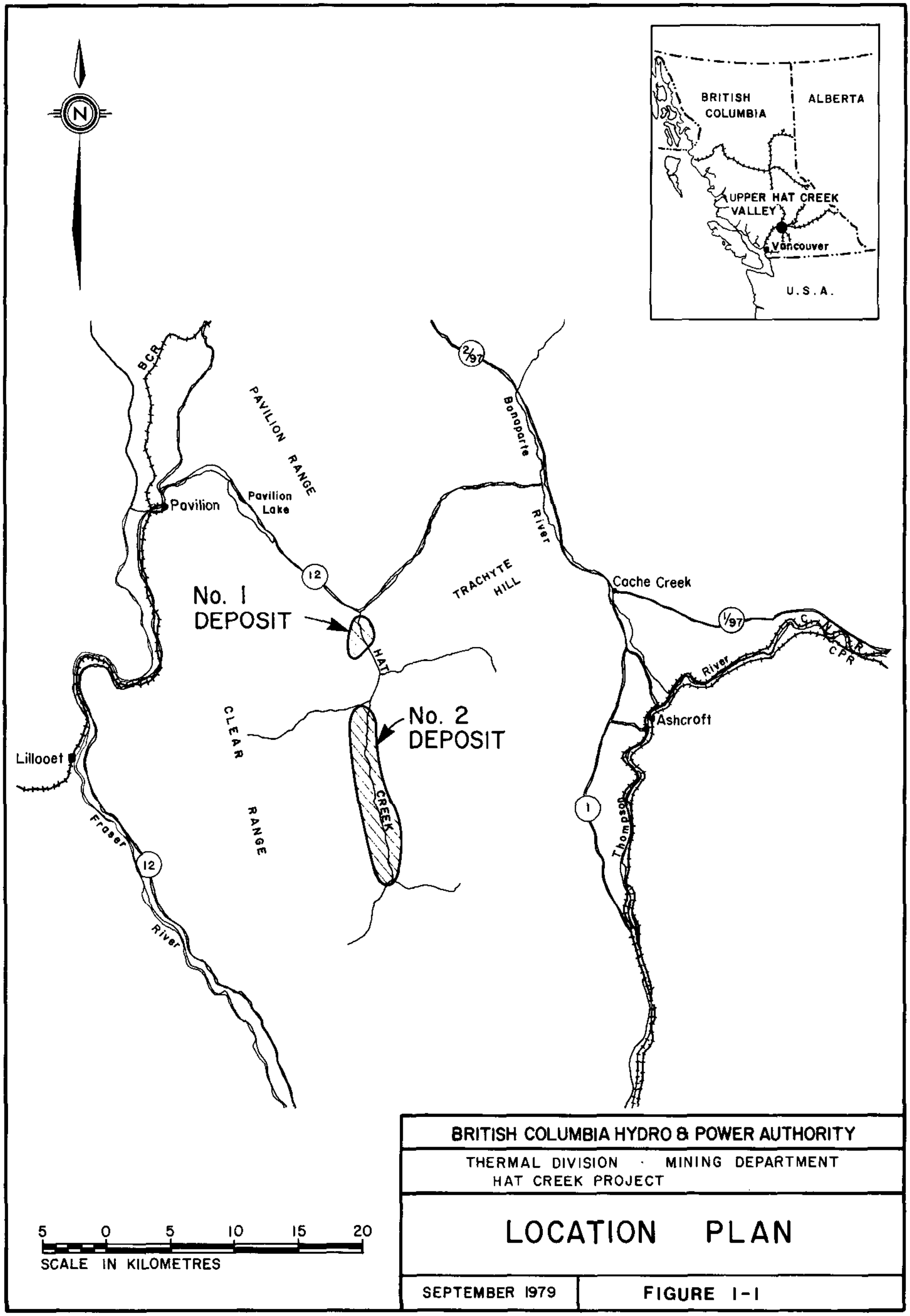
(3) Krupp Canada Inc.: application of bucketwheel technology.

Each of these groups have submitted reports on their work, (see Section 11). The results of this work has been incorporated in the appropriate sections of this report. The valuable contribution of these groups to this study is acknowledged.

1.2 PROJECT LOCATION

The project is based on the development of the extensive coal deposits located in the Hat Creek Valley in the Southern Interior of British Columbia. The valley is situated approximately 200 km North-East of Vancouver, midway between the towns of Lillooet and Ashcroft. Railroad access is available at Pavilion (B. C. Railway) 15 miles to the North-West and at Ashcroft (Canadian National and Canadian Pacific). Road access is available from Highway 12 at the North end of the valley. Highway 12 joins the Trans Canada Highway (via Highway 97) at Cache Creek approximately 30 km away by road. Regular airport service is available at Kamloops approximately 90 km to the East. Figure 1-1 shows the project location.

The Hat Creek Valley is underlain by coal deposits of unique thickness: approximately 500 m. The total resource has been projected to contain 10-15 billion tonnes of coal. Although first identified in 1877 the extent and scale of the deposits has only been recognized in the last few years. Recent exploration has identified two deposits, and a possible third, that are amenable to open pit mining.



BRITISH COLUMBIA HYDRO & POWER AUTHORITY

THERMAL DIVISION · MINING DEPARTMENT
HAT CREEK PROJECT

LOCATION PLAN

SEPTEMBER 1979

FIGURE 1-1

SECTION 2

SUMMARY

2.1 RESULTS

The principal results that have been developed in this prefeasibility study are:

(1) There are adequate coal reserves in the Hat Creek No. 1 and No. 2 Deposits to supply fuel to the Coal Liquefaction Plant and 2000 MW Powerplant for 30 and 35 years respectively. At the termination of the project a substantial quantity of recoverable coal will remain in the No. 2 Deposit.

(2) Mining plans have been developed to produce 907 million tonnes of coal over the project life, which requires the removal and disposal of 2033 million m^3 (4066 million tonnes) of waste. This represents approximately 4½ tonnes of waste for each tonne of coal: a stripping ratio of 2.24 m^3/tonne. The split between the two deposits is as follows:

(a) No. 1 Deposit

567 Mt coal; 1012 Mm^3 waste; 1.78 m^3/t stripping ratio.

(b) No. 2 Deposit

340 Mt coal; 1021 Mm^3 waste; 3.0 m^3/t stripping ratio.

(3) Summary of estimated costs in fourth quarter 1980 Canadian $:

(a)	Capital costs to full production capability, end of Year 3 (1989)	$ 981 million
	Pre-production operating costs to start of substantial production, end of Year 2 (1988)	$ 369 million
	Total Initial Capital Expenditures (excluding interest during construction)	$1350 million

(b) Additional capital expenditures including the development of No. 2 Deposit and equipment replacements over the project life. $ 841 million

(c) Total operating cost over remaining project life. $4155 million

Total Project Expenditure $6346 million

(4) The cost of coal delivered to the Coal Liquefaction Plant, and to the 2000 MW Powerplant, in fourth quarter 1980 Canadian dollars is $11.26 per tonne.

This price was produced as the result of a financial analysis carried out on the mining costs by the Ministry of Industry and Small Business Development.

(5) Operating manpower requirements are in the 1250-1300 range from Years 3-15. The anticipated peak requirement is between Years 15 and 20 when an estimated 1564 people are needed. This period coincides with operations being conducted in both deposits simultaneously.

(6) Coal quality cannot be defined reliably on a short term basis in a study conducted at this level of detail. The mean coal quality on a dry coal basis for the 567 million tonnes to be mined from No. 1 Deposit is estimated at a heating value of 17.7 MJ/kg, 34.5% Ash and 0.51% Sulphur; and for the 340 million tonnes to be mined from No. 2 Deposit, 17.3 MJ/kg, 35.5% Ash and 0.66% Sulphur. Moisture content is estimated at 23.5%. With detailed analysis and planning, and with the application of selective mining techniques it is expected that the quality of coal produced would be improved - particularly in No. 2 Deposit.

Note: all years referred to in this report are related to the start of commercial operation of the first Powerplant unit which is calendar Year 1 (1987). Year -1 is 1986; Year 2 is 1988.

2.2 CONCLUSIONS AND RECOMMENDATIONS

2.2.1 Conclusions

The following conclusions are drawn from this mining study:

(1) The project appears to be technically feasible applying well-proven mining technology.

The large volumes of material to be moved over long distances can only be handled by conveyors. The most effective method of digging and delivering the material to the conveyors is using a bucketwheel excavator.

(2) The proposed project schedule does not allow time for the required field investigations to be performed prior to the feasibility stage. Undertaking the mining feasibility studies without this additional data will seriously impair the level of confidence in the results.

(3) The mining and delivery of coal to both the 2000 MW Thermal Powerplant and the Coal Liquefaction Plant provides an effective, complementary operation. The flow of coal products will require careful coordination and balance between the two plants.

(4) Subject to confirmation by further testing, the scheme presented in this report provides suitable fuels to both the Powerplant and Coal Liquefaction Plant.

(5) The principal environmental concerns are: water quality, land reclamation and dust control. At this stage it appears that the negative impacts can be minimized by proper consideration in design and operation.

(6) This is a complex and large project. The work must be coordinated with strong direction and control exercised over the various consultants and contractors.

(7) Anticipated shortages of skilled manpower, especially tradesmen and professional staff will require the development of extensive programs to recruit, train and retain the necessary work force. The training function could well be integrated with other industry, regional or provincial schemes.

(8) The mine operation is capital intensive but the energy component is primarily electrical and operating costs should prove relatively stable. The preliminary indications are that the unit cost of coal is lower than that previously estimated in the 2000 MW Powerplant only base case study. However, as noted in Section 2.2.2 further studies are required to confirm this.

2.2.2 Recommendations

(1) The highest priority must be assigned to creating a viable mining organization to plan and direct the field investigations and feasibility studies. This initial group would provide the nucleus of the expanded organization required for the development and operation of the mine.

During the preliminary organization phase it is important to consider the integration and coordination of the design and operating objectives for the total project complex: Mine, Powerplant and Coal Liquefaction Plant.

(2) The proposed project schedule should be extended by at least two years to allow the following investigations to be completed prior to the feasibility study:

a) Geological drilling No. 2 Deposit.

b) Geotechnical and hydrological first phase investigations No. 2 Deposit and waste disposal areas.

c) Overburden drilling No. 1 and No. 2 Deposits.

d) Hat Creek diversion investigations.

e) Coal crushing, screening and material handling characteristics.

A preliminary budget of $8-10 million should be established for these programs.

(3) The mining feasibility study should be prepared using bucketwheel excavators and conveyors as the principal production system.

(4) Because it appears likely that B. C. Hydro's 2000 MW Powerplant development will proceed before the Coal Liquefaction Plant; it is essential that the requirements of the combined project be incorporated in the planning. This includes such items as:

a) Location of the crushing and blending facilities.
b) Size and capacity of the overland conveyor.
c) Creek diversion.

2.3 PROJECT DESCRIPTION

2.3.1 Introduction

A conceptual mining scheme has been developed to supply the coal requirements for B. C. Hydro's 2000 MW (net) coal fired Powerplant over 35 years and a Coal Liquefaction Plant to produce 8000 m^3/day of liquid products over 30 years. This scheme is based on mining the Hat Creek No. 1 Deposit to its economic limit first, and then mining the remaining coal requirements from the No. 2 Deposit. During the peak years of the project, when both plants are operating at full capacity, this will require the mining and delivery of approximately 30 million tonnes of coal per year.

The principal features of the project are shown on the project layout map, Figure 2-1. No. 1 Open Pit is located at the North end of the valley with the two plants to the East: the Powerplant 4½ km and the Liquefaction Plant 7 km away. Both plants are at a significantly higher elevation: approximately 550 m above the pit head. Houth Meadows to the North-West and Medicine Creek to the South-East are the waste disposal areas for the No. 1 Pit. The East end of the Medicine Creek area is reserved for the disposal of ash from the Powerplant and Liquefaction Plant. South of No. 1 Pit the No. 2 Pit is flanked by the Anderson and Ambusten Creek waste disposal areas.

In reviewing the project layout it is important to appreciate the scale of the project: particularly the materials transportation. Coal from No. 1 Deposit must be transported to the two plants a mean distance of at least 8 km from its current location; from No. 2 Deposit the distance is at least 10 km. The waste materials must be transported a similar distance. The total project involves the movement of some 5 billion tonnes of materials over distances ranging from 6 to 10 km; often negotiating significant elevation changes.

The plan developed provides for mining 567 million tonnes of coal and 1012 million cubic metres of waste from No. 1 Deposit and 340 million tonnes of coal and 1021 million cubic metres of waste from No. 2 Deposit. The waste quantities are controlled by the angle at which the pit slopes can be safely excavated. At Hat Creek the pit slopes are relatively flat: between 16° and 25°. These flat slopes are dictated by the weak rocks that surround the coal deposits. These rocks are mainly saturated claystones often with a high bentonite content.

The primary mining method selected is the bucketwheel excavator (BWE) - conveyor system, which is expected to handle approximately 85% of the materials moved over the life of the project. The truck-shovel-conveyor system selected to move the remaining materials, will provide flexibility to mine material that is inaccessible to the BWE or unsuitable for the excavator. All mined materials, after initial development is complete, will be transported by conveyor: coal to the plant areas and waste to the waste dumps where it will be placed in layers by crawler-mounted spreaders (See Figure 5-6).

2.3.2 Mining No. 1 Deposit

2.3.2.1 Pre-production Development

During the pre-production development period the mine must be prepared for coal production and the necessary equipment and facilities erected. This work will be initiated at the start of Year -4, immediately upon project authorization.

Hat Creek must be diverted around No. 1 Deposit before any significant mining can take place. The planned diversion works consists of a headworks dam to control the flow and channel it into a diversion canal which carries the water to a tunnel excavated in rock around the East side of the pit which, in turn, delivers the water to a buried conduit for returning the water to the creek downstream of the mine facilities. The diversion system is designed to handle the 1000-year return flood. An emergency spillway is incorporated into the headworks structure to prevent overtopping of the dam with the overflow water channelled to the mine.

In parallel with the diversion work, a mine drainage scheme must be developed. Numerous small lakes and ponds along the West side of the valley must be drained to reduce the risks of potentially unstable slide masses being activated. A network of ditches and pumping wells together with a series of treatment and holding lagoons will be established.

Site preparation and construction of the mine support facilities and surface equipment must be executed during the pre-production period. These facilities and equipment include: maintenance shops, warehouse and offices; overland conveyors; crushing and screening plant; blending stockpiling and reclaim equipment.

Mine development commences with the excavation and construction of conveyor ramps and roads, and the preparation of mine benches.

2.3.2.2 Pit Operation

The pit development sequence is designed to produce safe working slopes, release a reliable supply of coal for the plants and provide for the effective operation of the principal production equipment: the bucketwheel excavator. Essentially this means a bucketwheel should be kept operating on one bench as long as possible to minimize the loss of output and expense of relocation. The expense and time involved in relocation is primarily associated with the dismantling and reerection of the conveyor system. The plan developed for No. 1 Deposit results in each BWE mining on the same bench for an average of 8-9 years. This imposes a requirement for thorough planning with a severe penalty on poor planning and changes.

The plan provides for five BWE systems to be installed with each system producing at an average rate of 3000 m^3/h or 12 million m^3/year. The first system will commence operation at the start of Year -1 with the others following at six month intervals. As illustrated in Figure 5-3, each BWE will mine a total bench height of 45 m in three stages:

(1) High cut 25 m: machine travels 10 m above conveyor elevation.

(2) High step 10 m: machine travels on conveyor elevation.

(3) Low step 10 m: machine travels 10 m below conveyor elevation; cut is taken behind the conveyor.

The initial operations with the BWE are primarily on the West side of the valley above the present creek elevation. As these benches are moved uncovering the lower benches the pit is deepened to establish new mining benches. BWE will do most of the mining down to the 755 m elevation, approximately 140 m below the present valley elevation. The shovel truck system will mine below 755 m elevation; prepare benches for the BWE; supplement production where required; and be used to adjust the coal quality on a short term basis. Figure 5-4 shows the respective BWE and shovel truck operating areas in No. 1 Pit.

Each BWE is supported by its own 1800 mm wide conveyor system which is loaded by a conveyor bridge connected to the BWE. The conveyor system delivers the material to a conveyor distribution station at the pit head.

The truck-shovel system comprises 14.5 m^3 hydraulic shovels loading 91-tonne trucks, which haul the coal and waste to a truck dumping station at the central conveyor incline. The truck dumping station includes a grizzly, surge hopper and pan-feeders for loading either of the two conveyors installed. Over the life of the mine, truck dumping stations will be established at four different elevations to minimize truck requirements. The two in-pit incline conveyors also deliver materials to the conveyor distribution station at the pit head. During the pre-production period a fleet of 154-tonne trucks are provided for because of the longer truck haulage distances that are required.

2.3.2.3 Material Handling

All mine materials, coal and waste, are delivered by conveyor to the conveyor distribution station at the pit head. Seven conveyors: five from the BWE's and two loaded from trucks, carry material into the distribution station. Each of the seven incoming conveyors can discharge material onto any of the five outgoing conveyors. Three of the outgoing conveyors deliver waste materials to the waste disposal areas. The other two are overland coal conveyors, which deliver coal to the crushing and screening plant adjacent to the Powerplant.

Waste materials are delivered by conveyor to the waste areas at Houth Meadows and Medicine Creek. The waste dumps consist of an engineered waste retaining embankment constructed from material uncontaminated with bentonitic clay, and the zone behind the embankment for the placement of unstable waste materials. The materials are delivered to the stackers by 2400 mm conveyor belts and placed in lifts totalling 35 m in thickness. Three stackers, each capable of handling the output from two BWE are provided: two on Medicine Creek and the other in Houth Meadows. The East end of the Medicine Creek waste disposal area is reserved for the disposal of ash from the Powerplant and Coal Liquefaction Plant.

Two parallel overland conveyor lines 4750 m long, with a vertical lift of 535 m and constructed in three flights deliver the coal to the Crushing and Screening Plant. This plant has the function of separating two coal streams of different quality produced by the mine into four size/quality components that can be combined in varying proportions to produce specification coal for the three plants: Powerplant; Gasifiers; and the CLP Steamplant. The principal specification criteria for the three products are:

(1) Gasifiers: top size 100 mm, effectively 75 mm; +13 mm containing less than 2% -2mm fines.

(2) Powerplant: top size 50 mm; consistent quality of 18.1 MJ/kg, 33.5% Ash and 0.51% Sulphur all on a dry coal basis.

(3) Steamplant: top size 50 mm; consistent quality of 16.1 MJ/kg (dcb).

In order to meet the Gasifier coal size requirements storage and handling of the coal must be minimized. Thus, the system is designed to prepare and deliver the coal on a continuous basis with only minor surge capacity provided. The Powerplant and Steamplant coals must be blended. A comprehensive blending stockpile system is included in the scheme. The conceptual design of the coal handling system provides for flexibility and reliability of operation through a combination of redundant equipment and alternative paths through the system.

Coal delivery to the Coal Liquefaction Plant is accomplished using three conveyor lines: one dedicated to each of the Gasifier and Steamplant feed; the third conveyor can be used for either stream. Delivery to the Powerplant is by one of the two conveyors that are fed by coal recovered from the blending stockpile by bridge-type bucketwheel reclaimers. If necessary, the blending operation can be bypassed and coal delivered directly from the Crushing and Screening Plant to the Powerplant or the Steamplant.

2.3.3 Mining No. 2 Deposit

The development and mining of No. 2 Deposit has many similarities to No. 1. In this section only the major items or significant differences from No. 1 Deposit will be discussed.

2.3.3.1 Pre-production Development

The second stage of the Hat Creek diversion must be completed during Year 15 to permit the start of mine development in Year 16. This stage of the diversion works will require a new headworks dam, pit rim dam and 11 km of diversion canal along the West side of the valley. This diversion canal would also collect the flow from five tributary creeks before being discharged through a conduit to the Stage 1 headworks dam and the remainder of the original works.

A second mine drainage scheme would have to be developed on No. 2 Deposit to drain small lakes and ponds including Fishhook Lake, and the surficial materials would be dewatered by pumping.

Additional mine support facilities would be constructed adjacent to No. 2 Deposit. These would be on a smaller scale than those provided for No. 1 Deposit and limited to the facilities such as equipment service shops and field offices that must be located close to the operation.

2.3.3.2 Pit Operation

The five BWE's used in No. 1 Pit will be moved to No. 2 as their assignments are completed. In addition, two larger BWE, 5000 m^3/hour capacity, are required to handle the increased stripping ratio in No. 2 Deposit. Excavation commences in Year 16 with the initial coal production scheduled for Year 20. Total coal production comes from No. 2 Deposit from Year 22 onward.

Approximately 150 million m^3 of waste must be removed prior to the start of coal production. Peak annual production requirements of all materials are over 100 million m^3 compared to approximately 75 million m^3 for No. 1 Deposit. A fleet of hydraulic shovels and 154-tonne trucks are provided but are not assigned any specific production quantity. They will be used for handling materials the BWE cannot handle such as the 20 Mm^3 of volcanics in the South-East side of the pit, and for bench development and general support.

2.3.3.3 Materials Handling

A conveyor distribution station is required to handle seven BWE system conveyors and one truck dump conveyor coming in and seven outgoing conveyors: five waste and two coal conveyors that join the existing overland coal conveyor system.

Initial waste disposal is provided in the Anderson Creek area, but the bulk of the waste will be used to backfill No. 1 Pit after the economic coal has been mined out. Five waste spreaders are required to handle the volume produced by the seven BWE mining systems.

2.3.4 Environmental Protection

The conceptual plan includes provisions for protecting the environment. The major areas for attention that have been identified are:

(1) Water Quality

The mine dewatering and drainage scheme provides for handling three separate types of water. The first is the water suitable for simple diversion without any form of treatment. The second requires sedimentation to bring it to an acceptable quality for discharge. The third type is unsuitable for discharge and will be routed to holding lagoons and disposed of by evaporation in dust control application.

(2) Land Reclamation

Significant field testing has been done by B. C. Hydro to establish a sound basis for reclamation. Reclamation will be conducted on a progressive basis throughout the life of the project. At the end of the project the only significant area not reclaimed will be the lower elevations of No. 2 Pit.

(3) Dust Control

Dust control measures to meet the objectives have been incorporated into the conceptual design. These include: covering certain conveyors, construction of windbreak berms, orientation of the coal piles and installation of water sprays. Specific operating procedures will also be required to minimize the particulate emissions.

2.4 IMPACT ON THE HAT CREEK 2000 MW PROJECT

B. C. Hydro is proposed to build a 2000 MW (net coal fired Powerplant at Hat Creek based on utilizing part of the No. 1 Deposit. This section identifies areas where the integration of a Coal Liquefaction Plant with B. C. Hydro's project causes changes and the estimated direction and magnitude of the impact from the mining viewpoint.

(1) There is a significant change in the scale of the mining operation: approximately three times more coal must be mined at a higher overall stripping ratio. The mining methods and equipment are changed to cope with the problem. The impacts of this change are:

    (a) Increased complexity requiring more thorough planning and allowing a smaller margin for error.

    (b) The cost of coal is not expected to change significantly, but could be slightly lower.

(2) No significant change is expected in the quality of coal supplied to the Powerplant in the early years. The outlook in the later years, when the coal must come from No. 2 Deposit is less certain. The preliminary indications are that the heating value supplied could be slightly lower and that the sulphur content would almost certainly be higher: probably about 30% higher. The impact on the Powerplant could require the addition of flue gas scrubbing capacity.

(3) The coal preparation and blending facilities are relocated from the North end of the valley to the vicinity of the Powerplant. Twinning of the overland conveyor system improves the reliability of supply permitting a reduction in the size of the compacted, dead storage pile. The overall operation of the coal delivery system is considered to be more reliable and cost effective.

(4) The integrated project schedule produces a high degree of interdependence between the two plants: the gasifiers require coarse coal feed leaving fines that must be burned in either the Powerplant or the Steamplant. The Coal Liquefaction Plant operating at maximum capacity requires production of a quantity of fines far in excess of the Steamplant consumption capacity.

These excess fines must be burned in the Powerplant. If the Powerplant is operating at a low capacity factor this could lead to difficulty in maintaining coal quality in an acceptable range or require the storage of excess fines until balanced operation can be reached. Should the reverse situation occur with the Powerplant at high capacity factor and the CLP on low output little difficulty would be encountered.

(5) The creek diversion around the East side of the No. 1 Deposit is changed from an open canal to a tunnel. This change would produce environmental benefits downstream on Hat Creek because less temperature rise would occur in the water diverted through the tunnel rather than the canal.

(6) Possible scheduling difficulty could arise in the integration of the two projects. Delays in a decision to pursue the Coal Liquefaction Project could cause disruption in the Powerplant development schedule should integration be attempted later. Extra costs would be caused by a late decision to integrate. Key features of the joint project must be planned and provided for in any development on the site if excessive costs are to be avoided.

TABLE 2-1

SUMMARY OF ANNUAL COSTS, CANADIAN $ OCTOBER 1980

COAL LIQUEFACTION PROJECT PREFEASIBILITY STUDY 1980-MINING

Year	Annual Coal Production tonnes x 10^6	Total Cumulative tonnes x 10^6	$000's Annual Operating Cost	$ Annual Operating Cost/tonne	$000's Annual Capital Costs	$000's Total Annual Capital + Operating Cost	$000's Total Cumulative Operating + Capital Cost
-6			66		29,722	29,788	29,788
-5			5,764		11,295	17,059	46,847
-4			11,707		82,142	93,849	140,696
-3			33,200		190,316	223,516	364,212
-2			44,147		277,352	321,499	685,711
-1			72,228		184,375	256,603	942,314
1	4.8	4.8	90,013	18.75	156,403	246,416	1,188,730
2	15.7	20.5	111,507	7.10	33,298	144,805	1,333,535
3	24.8	45.3	126,446	5.10	16,554	143,000	1,476,535
4	29.3	74.6	126,700	4.32	21,771	148,471	1,625,006
5	30.0	104.6	126,132	4.20	3,325	129,457	1,754,463
6	30.0	134.6	121,424	4.05	9,185	130,609	1,885,072
7	30.0	164.6	119,070	3.97	14,418	133,488	2,018,560
8	30.0	194.6	116,237	3.87	14,661	130,898	2,149,458
9	30.0	224.6	115,493	3.85	9,511	125,004	2,274,462
10	30.0	254.6	115,262	3.84	16,729	131,991	2,406,453
11	30.0	284.6	114,840	3.83	7,936	122,776	2,529,229
12	30.0	314.6	115,624	3.85	13,665	129,289	2,658,518
13	30.0	344.6	114,889	3.83	19,352	134,241	2,792,759
14	30.0	374.6	117,182	3.91	25,596	142,778	2,935,537
15	30.0	404.6	120,665	4.02	129,337	250,002	3,185,539
16	30.0	434.6	122,250	4.08	150,130	272,380	3,457,919
17	30.0	464.6	156,521	5.22	144,317	300,838	3,758,757
18	30.0	494.6	167,033	5.57	34,486	201,519	3,960,276
19	30.0	524.6	164,592	5.49	27,490	192,082	4,152,358
20	30.0	554.6	164,706	5.49	52,412	217,118	4,369,476
21	30.0	584.6	157,767	5.26	9,306	167,073	4,536,549
22	30.0	614.6	139,788	4.66	50,651	190,439	4,726,988
23	30.0	644.6	145,676	4.86	3,227	148,903	4,875,891
24	30.0	674.6	143,676	4.79	6,149	149,825	5,025,716
25	30.0	704.6	144,810	4.83	16,349	161,159	5,186,875
26	28.0	732.6	143,705	5.13	3,325	147,030	5,333,905
27	28.0	760.6	141,324	5.05	5,194	146,518	5,480,423
28	28.0	788.6	131,569	4.70	16,709	148,278	5,628,701
29	28.0	816.6	125,657	4.49	4,477	130,134	5,758,835
30	28.0	844.6	118,548	4.23	5,281	123,829	5,882,664
31	28.0	872.6	116,246	4.15	8,171	124,417	6,007,081
32	8.6	881.2	101,490	11.80	3,617	105,107	6,112,188
33	8.6	889.8	79,285	9.22	5,759	85,044	6,197,232
34	8.6	898.4	67,728	7.88	7,307	75,035	6,272,267
35	8.6	907.0	73,510	8.55	491	74,001	6,346,268
Total	907.0	907.0	4,524,477	4.98	1,821,791	6,346,268	6,346,268

TABLE 2-2

BREAKDOWN OF TOTAL ESTIMATED CAPITAL AND OPERATING EXPENDITURES BY MAJOR COST CENTRES

($000's October 1980)

Hat Creek Coal Liquefaction Project Prefeasibility Study 1980 - Mining

Cost Centre	Amount ($000's)	($) Unit Cost/tonne of Coal Produced
Engineering and Construction	82,820	0.09
Mine Property Development	102,377	0.11
Buildings and Structures	40,200	0.04
Mining Equipment	567,369	0.63
Coal Conveying, Screening and Blending	232,623	0.26
Waste Disposal Equipment	426,306	0.47
Reclamation	3,005	0.003
Creek Diversion	47,500	0.05
Construction Camp	10,000	0.01
Housing and Infrastructure	28,000	0.03
Land Acquisition	4,500	0.005
Previous Studies	17,500	0.02
Contingency	259,591	0.29
TOTAL CAPITAL COSTS	1,821,791	2.01
Drilling	9,533	0.01
Blasting	27,218	0.03
Loading	560,555	0.62
Hauling	164,875	0.18
Coal Handling System	344,849	0.38
Waste Handling System	495,614	0.54
Auxiliary Equipment	210,158	0.23
Power	383,721	0.42
General Mine Expense	596,749	0.66
Overhead	671,928	0.74
Head Office Expense	200,000	0.22
Creek Diversion	6,450	0.01
Royalties	197,274	0.22
Provincial Capital Tax	35,239	0.04
Contingency	620,314	0.68
TOTAL OPERATING COSTS	4,524,477	4.98

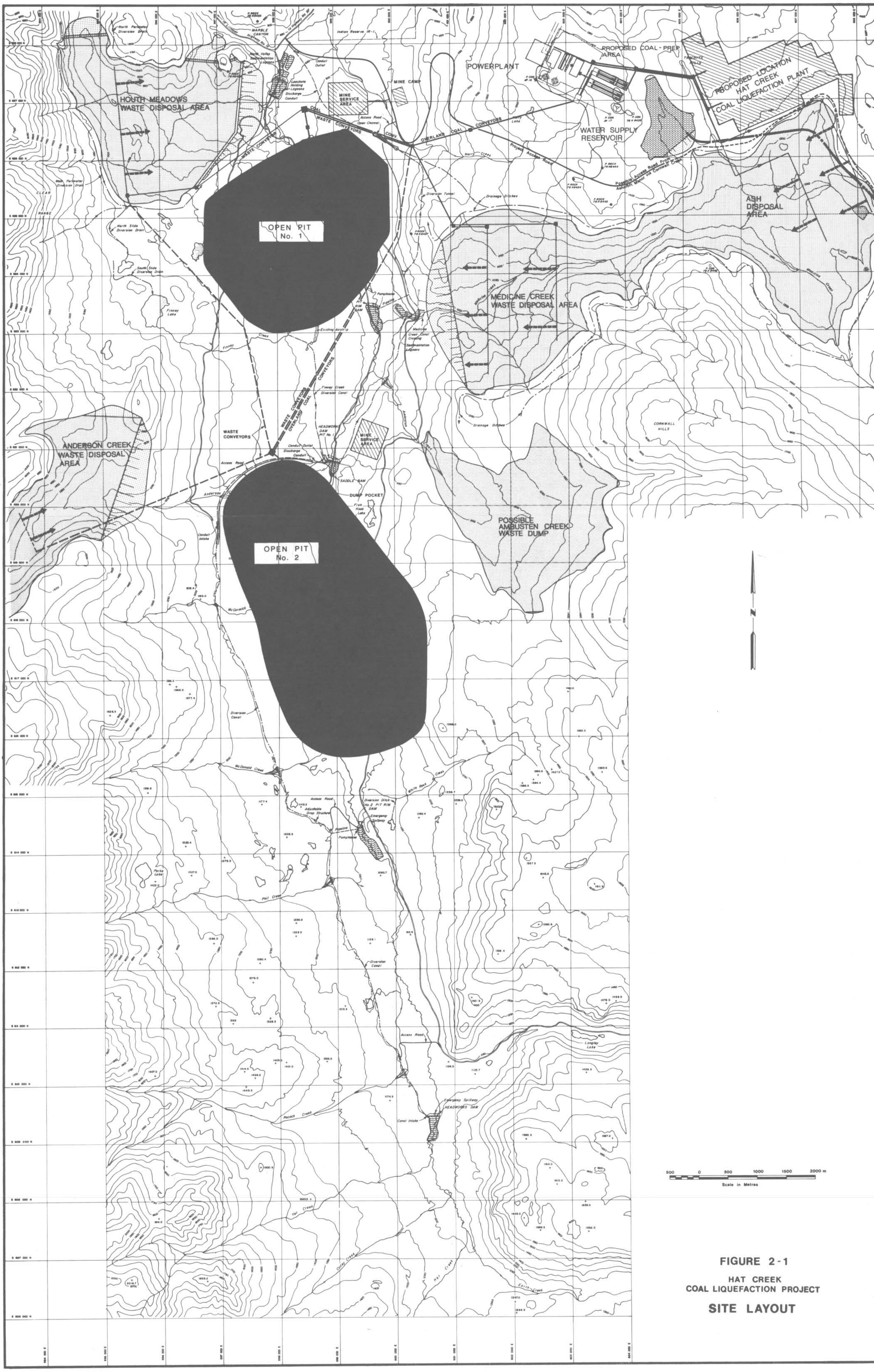


FIGURE 2-1

HAT CREEK
COAL LIQUEFACTION PROJECT

SITE LAYOUT

SECTION 3

GEOLOGY

3.1 PHYSIOGRAPHY

The Hat Creek Valley is situated in the western fringe of the Thompson plateau in the central B. C. Interior Dry Belt, about 90 km West of Kamloops and 240 km North-East of Vancouver (Figure 1-1). The low land of the Upper Hat Creek Valley is 24 km long and 3.5 km wide. The valley floor ranges in elevation from 810 m in the North to 1140 m in the South above sea level. The mountains bordering the valley are the Trachyte Hills to the East and the Clear Range to the West. The hills are elevated to 2050 m and 2300 m, respectively.

Hat Creek has its source in the South-Eastern slopes of the Clear Range and traverses the broad, North-trending grassland of the valley before turning Northeast to merge with the Bonaparte River. The Bonaparte River, in turn, joins the Thompson River just North of Ashcroft. The Thompson and Fraser Rivers, both flowing to the South, are 19 km and 22 km respectively from the Hat Creek Valley.

The uplands of the valley are covered with open forests and meadows, while the low land consists of sparsely treed open ranges of grass and sage. Extensive deposition of surficial materials during de-glaciation of the Pleistocene Period left most of the area of the Hat Creek Valley devoid of bedrock exposures. Most of the outcrops are confined to steep hillsides or along sharply incised gulleys. As a result, the geological interpretation of the Hat Creek coal basin has been based primarily on drilling and geophysical data.

3.2 REGIONAL GEOLOGY

3.2.1 Stratigraphy

The regional bedrock geology of the Hat Creek coalfield consists of various rock units ranging in age from Permian to Tertiary as illustrated in Figure 3-1 and Table 3-1. The basement of the Hat Creek coal basin is made up of the Paleozoic Cache Creek Group, which consists of limestone of the Marble Canyon Formation overlying greenstone. The Cache Creek Group in the Ashcroft area is stratigraphically overlain by the Upper Triassic Nicola (?) Group, with the contact between the two modified by the subsequent faulting (Traverse 1970 and Shannon 1980). The limestone of the Cache Creek Group to the Northwest of the Hat Creek area was intruded by the Mount Martley stock of Jurassic or Cretaceous age which is composed of granodiorite and tonalite, coeval with the Lytton Batholith. The stock is overlain unconformably by Lower Cretaceous andesite and basalt of the Spences Bridge Group. The Kamloops Group is of Eocene Age. Deposition of this unit began with volcanic eruptions of lavas and pyroclastics composed of rhyolite, dacite and basalt. A mélange unit of the undifferentiated volcaniclastics on the Western slopes of the Trachyte Hills has been assigned to the Kamloops Group on a preliminary basis. These volcanic piles are overlain by clastic rocks composed mainly of conglomerate, sandstone and siltstone of the Coldwater Formation. Lying conformably over the Coldwater Formation is the coal-bearing, Hat Creek Coal Formation, over 500 m thick. The coal-bearing strata are overlain in apparent paraconformity by the Medicine Creek Formation, which is made up of soft, semi-indurated, monotonous lacustrine sediments, with over 550 m in true thickness. The sharp paraconformable contact of the Medicine Creek Formation with the coal measures represents a rapid facies change from moor (marsh) to lacustrine depositional conditions. All the above sedimentary and igneous successions have been unconformably overlain by lahar, which is made up of angular to subrounded fragments and boulders in a sandy clay matrix derived from the lower Kamloops volcanis. Due to the unconformity between the lahar unit and the lower stratigraphic sequences, it is unknown whether the lahar is part of the Kamloops Group or if it is related to the volcanic activity of the Miocene Epoch. The typical lahar of this unit outcrops above the Hat Creek main road, about 250 m South of the Medicine Creek junction.

The youngest volcanic rocks in the area are olivine basalt, basalt, vesicular basalt, andesite (locally) and the equivalent pyroclastics, all of the Miocene Epoch. A flow or dyke of these rocks occurs in the headwall of the active slide Northwest of the No. 1 Deposit. During the Pleistocene Epoch the entire Hat Creek area, along with much of the Interior Plains, underwent extensive glaciation. This resulted in the deposition of a variety of glacial and glacio-fluvial sediments, ranging in thickness from a few metres to 200 m.

The stratigraphic correlation in the No. 1 Deposit has been established with a high degree of confidence, based on a grid spacing of 150 m which includes 228 drill holes totalling 58,400 m. The Hat Creek Coal Formation in the No. 1 Deposit attains up to 500 m in true thickness and is divided into four zones; A, B, C and D in descending stratigraphic order. Further work identified two distinct waste zones between A and B, and B and C, which have been correlated over the entire No. 1 Deposit. With a further interpretation of the geophysical logs, it was possible to subdivide it further into 16 subzones. (See Figures 3-2 and 3-3.) However, stratigraphic correlation over the entire area of No. 2 Deposit is not possible at this time due to the low drilling density and partial intersection of the coal measures in the holes. A rapid facies change from main coal facies to the silty and coarser detrital facies is evident in traversing from East to West across the Hat Creek No. 1 and No. 2 Deposits.

3.2.2 Structure

The Hat Creek coal basin lies in a North-trending topographic depression within the South-West part of the Intermontane Belt of the Canadian Cordillera. The Fraser River separates the Intermontane Belt from the Coast Plutonic Complex. During the Eocene Epoch, nonmarine, synorogenic and syntectonic clastic sediments were deposited, preceded and possibly succeeded by the accumulation of sub-aerial volcanics. Mid-Tertiary erosional activities resulted in widespread surfaces of low relief. The main physiographic features of the Fraser and Thompson river drainage systems were well established at this time. The general structure of the Tertiary Coal Basin in the Hat Creek Valley is a graben, flanked on either side by gravity faults. This graben is formed principally by downward movement on a series of North-South trending tensional faults. Transverse faults trending North-West have locally offset the graben.

The basic structure of the No. 1 Coal Deposit in the North is a syncline flanked on the East by a fault-modified anticline and another syncline. The No. 1 Deposit plunges to the South at 15° to 22°. On the basis of limited drill data, the structure of the No. 2 Deposit to the South appears to be an anticlinal horst, of which the East limb is truncated by a boundary fault, bringing the Medicine Creek Formation and the Hat Creek Coal Formation into juxtaposition with the limestone of Cache Creek Group or the undifferentiated volcaniclastic sediments and volcanic rocks of Kamloops Group. The widespread occurrences of strike-slip and normal faults with North, North-East and North-West orientations were noted within the coal measures, but only the major tectonic elements and lineaments are illustrated in Figure 3-1.

3.3 HISTORY AND EXPLORATION

Coal outcrops in the Hat Creek Valley were first reported in 1877 by G.M. Dawson of the Geological Survey of Canada. Attention centred on the creek banks where erosion had exposed coal in what is now called the No. 1 Deposit. A limited program of exploration and drilling occurred by 1925, but commercial exploitation of the deposit did not begin until 1933. Between 1933 and 1942 a small amount of coal was mined for sale to local communities. Production ceased in 1942, and no further activity occurred until 1957, when a subsidiary of B. C. Electric Company Ltd. purchased an option on the property and undertook further drilling.

Ownership of the Crown grant and two coal licenses on Deposit No. 1 were transferred to B. C. Hydro when the Crown corporation was formed in 1962. No further exploration occurred until 1974, when B. C. Hydro began preliminary drilling of the known deposit. Later in 1974 and in 1975 B. C. Hydro obtained further coal licenses for most of the upper Hat Creek Valley. Subsequent exploration confirmed two deposits, the first at the northern end of the valley and the second further south in the upper valley. It also indicated the possibility of a third deposit south of No. 2 Deposit. Detailed geological work on the No. 1 Deposit has continued into 1979, in an effort to increase and refine B. C. Hydro's knowledge of this major energy resource.

The aggregate of all the drilling carried out in the valley exceeds 100,000 meters. Of this about 78,000 meters in 474 holes were drilled for the development of No. 1 Deposit. A drilling grid pattern of 150 m was followed. The remaining 21,800 m in 64 holes were drilled in No. 2 Deposit, to the South of No. 1. The holes were located on strategic sites for establishing the limits of the coal basin and its general structure. Surface geophysical surveys were successfully used to indicate the boundaries of the coal basin (by gravity survey) and the volcanic and burned zones (by magnetometer survey).

Over 95% of the core holes drilled since 1974 have been geophysically logged: gamma ray and bulk density. All cores recovered, usually over 97%, were geologically logged and split vertically for chemical analyses. The remaining half were stored for future reference.

Coal quality evaluation was primarily based on drilled cores, however, large samples were obtained for washability and combustion tests in 1976. In 1977 the burning characteristics of the coal were studied on 7000 tons of below average quality fuel mined from Trenches A and B. These tests demonstrated that a typical Hat Creek coal can be handled, pulverized and burned in a commercial scale powerplant. Further washability tests were performed and an 80-t sample was tested in a pilot scale coal washing plant.

3.4 COAL QUALITY

Systematic analytical work has been conducted on all core samples. The length of each core sample analyzed has varied over the years but more recently the procedure has been standardized to provide a 6 m maximum sample length for proximate, heating value and sulphur analyses. Ultimate and ash mineral analysis together with ash fusion temperature and grindability tests are performed using 12 m to 18 m maximum sample lengths.

Coal quality for the No. 1 Deposit is well defined because of the large number of samples (in excess of 4000) that have been analyzed, and the stratigraphic correlation that has been established. The coal quality for No. 2 Deposit has a much lower level of confidence because fewer samples (approximately 800) for a much larger deposit have been analyzed. Significant stratigraphic correlation has not been possible because of insufficient data. However, evaluation of the data available for No. 2 Deposit indicates that the coal properties for the two deposits are sufficiently similar to support treating them as identical for the purposes of this study.

The coal quality for No. 1 Deposit is shown in Table 3-2.

3.4.1 Ash-Heating Value Relationship

Studies on ash-heating value regression curves for the four major zones, A, B, C and D in No. 1 Deposit for samples $\leq 60\%$ ash show that they are almost identical. This indicates similar depositional environment and rank of metamorphism for the entire deposit, and permits adopting a common fit curve based on 4400 samples for the whole No. 1 Deposit:

$$\text{Heating Value (db) MJ/kg} = 29.5688 - 0.34134.Y$$
$$\text{Ash \% (db)} = 86.6254 - 2.9296.X$$
$$\text{Where } x = \text{Heating Value MJ/kg (db)}$$
$$y = \text{Ash \% (db)}$$

A similar study on No. 2 Deposit coal samples produced almost identical results. Therefore, this equation is applied to both deposits.

3.5 RESERVES

The reserve estimates for No. 1 Deposit were developed using a cross sectional computer model with variable sized blocks. For No. 2 Deposit the reserves were calculated manually using cross sections.

The geological reserves established assuming a 9.3 MJ/kg cut-off grade are:

No. 1 Deposit

740 million tonnes, proven and probable, at 17.7 MJ/kg; 34.8% Ash; 0.51% Sulphur (dry coal basis).

No. 2 Deposit

4560 million tonnes possible reserves. The data available is inadequate to supply a meaningful estimate of average quality. It was assumed that 25% of the computed coal zone volume would be waste on the basis of the ratio obtained in the existing drill holes.

TABLE 3-1

HAT CREEK COAL LIQUEFACTION PROJECT - PREFEASIBILITY STUDY 1980 - MINING

REGIONAL STRATIGRAPHY - HAT CREEK COAL BASIN

Period	Epoch	Million Years	Formation or Group	Thickness (m)	Rock Types
Quaternary	Recent			Not Determined	Alluvium, Colluvium, fluvial sands and gravels, slide debris, lacustrine sediments.
	Pleistocene	1.5 - 2			Glacial till, glacio-lacustrine silt, glacio-fluvial sands and gravels, land slides.
	Unconformity				
Tertiary	Miocene	7 - 26	Plateau Basalts	Not Determined	Basalt, olivine basalt (13.2 m.y.), vesicular basalt.
	Unconformity (?)				
	Miocene or Middle Eocene ?		Kamloops Group: Finney Lake Formation	Not Determined	Lahar
			Unconformity		
	Late Eocene		Kamloops Group: Medicine Creek Formation	550	Bentonitic claystone and siltstone.
			Paraconformity		
	Late Eocene to Middle Eocene	*36 - 42	Kamloops Group: Hat Creek Coal Formation	500	Mainly coal with intercalated siltstone, claystone, carbonaceous claystone, sandstone and conglomerate.
			Kamloops Group: Coldwater Formation	Not Determined	Siltstone, claystone, sandstone, conglomerate, minor coal.
			Fault Contact or Nonconformity		
	Middle Eocene	43.6-49.9	Kamloops Group	Not Determined	Dacite (49.1 m.y.), andesite, rhyolite(49.9 m.y.) basalt and equivalent pyroclastics.
	Unconformity (McKay 1925; Duffell & McTaggart 1952)				
Cretaceous or Later	Coniacian to Aptian**	88.3±3 m.y.	Spences Bridge Group	Not Determined	Andesite, dacite, basalt (88.3 m.y.), tuff, breccia.
	Erosional Unconformity (Duffell & McTaggart 1952)				
		98	Mount Martley Stock	Not Determined	Granodiorite, tonalite.
	Intrusive Contact (Duffell & McTaggart 1952)				
Pennsylvanian to Permian or Earlier		250-330	Cache Creek Group: Marble Canyon Formation	Not Determined	Marble, limestone, argillite.
			Cache Creek Group: Greenstone	Not Determined	Greenstone, chert, argillite, minor limestone and quartzite, chlorite schist, quartz-mica, schist, phyllite.

* Based on palynology by Rouse 1977.

** Based on plant fossils by Duffell & McTaggart 1952.

TABLE 3-2

COAL QUALITY NO. 1 DEPOSIT

COAL LIQUEFACTION PROJECT-PREFEASIBILITY STUDY 1980-MINING

	As Received	Dry
Proximate Analysis %		
Moisture	23.5	-
Ash	26.6	34.8
Volatile Matter	24.8	32.4
Fixed Carbon	25.1	32.8
Ultimate Analysis %		
Carbon	35.0	45.2
Hydrogen	2.8	3.6
Nitrogen	0.7	0.9
Chlorine	0.02	0.02
Oxygen (by difference)	11.6	15.5
Sulphur Forms %		
Pyritic	0.10	0.13
Sulphate	0.01	0.02
Organic	0.28	0.36
Higher Heating Value (MJ/kg)		
	13.5	17.7
MAF Basis		27.1

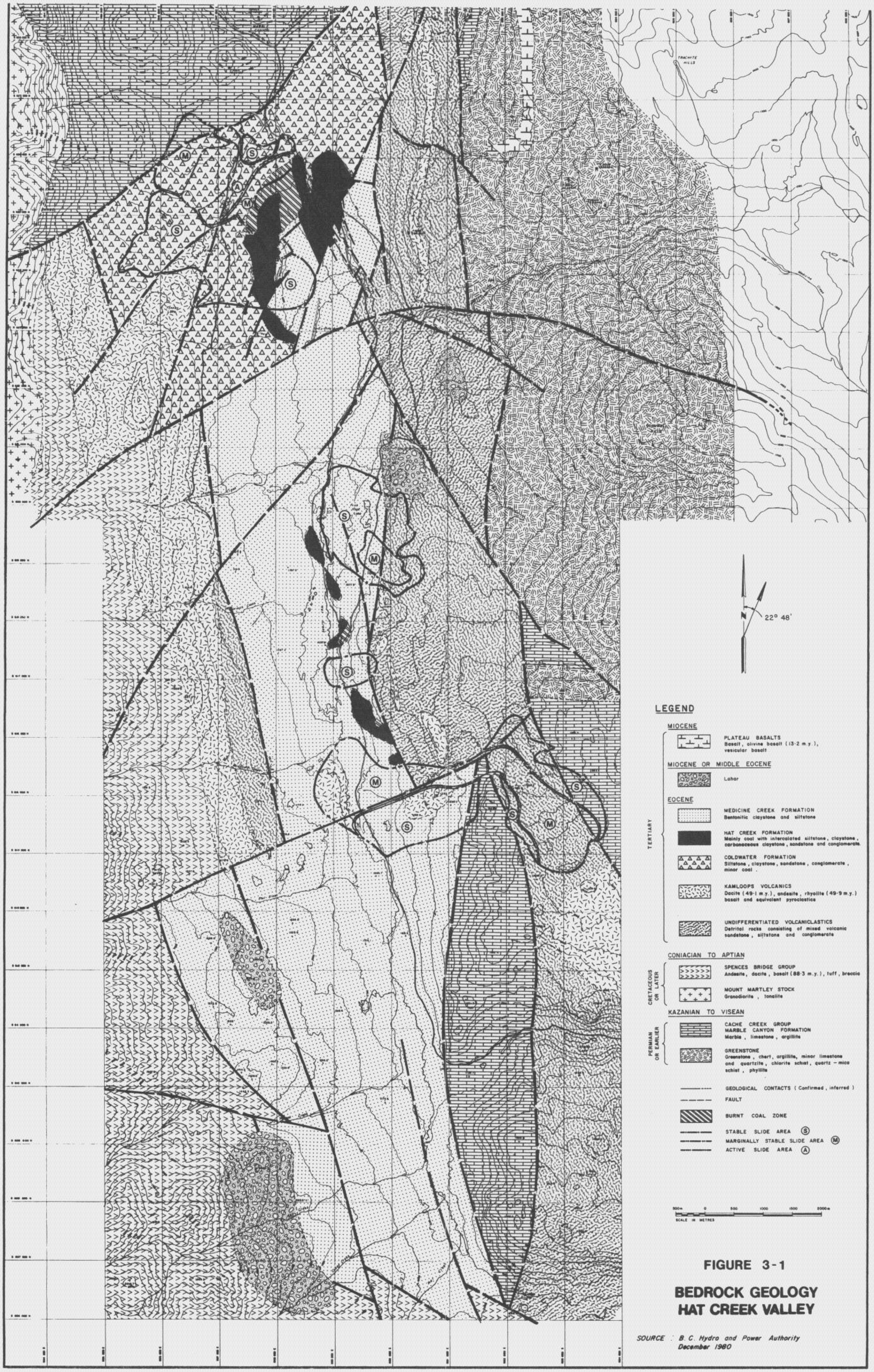


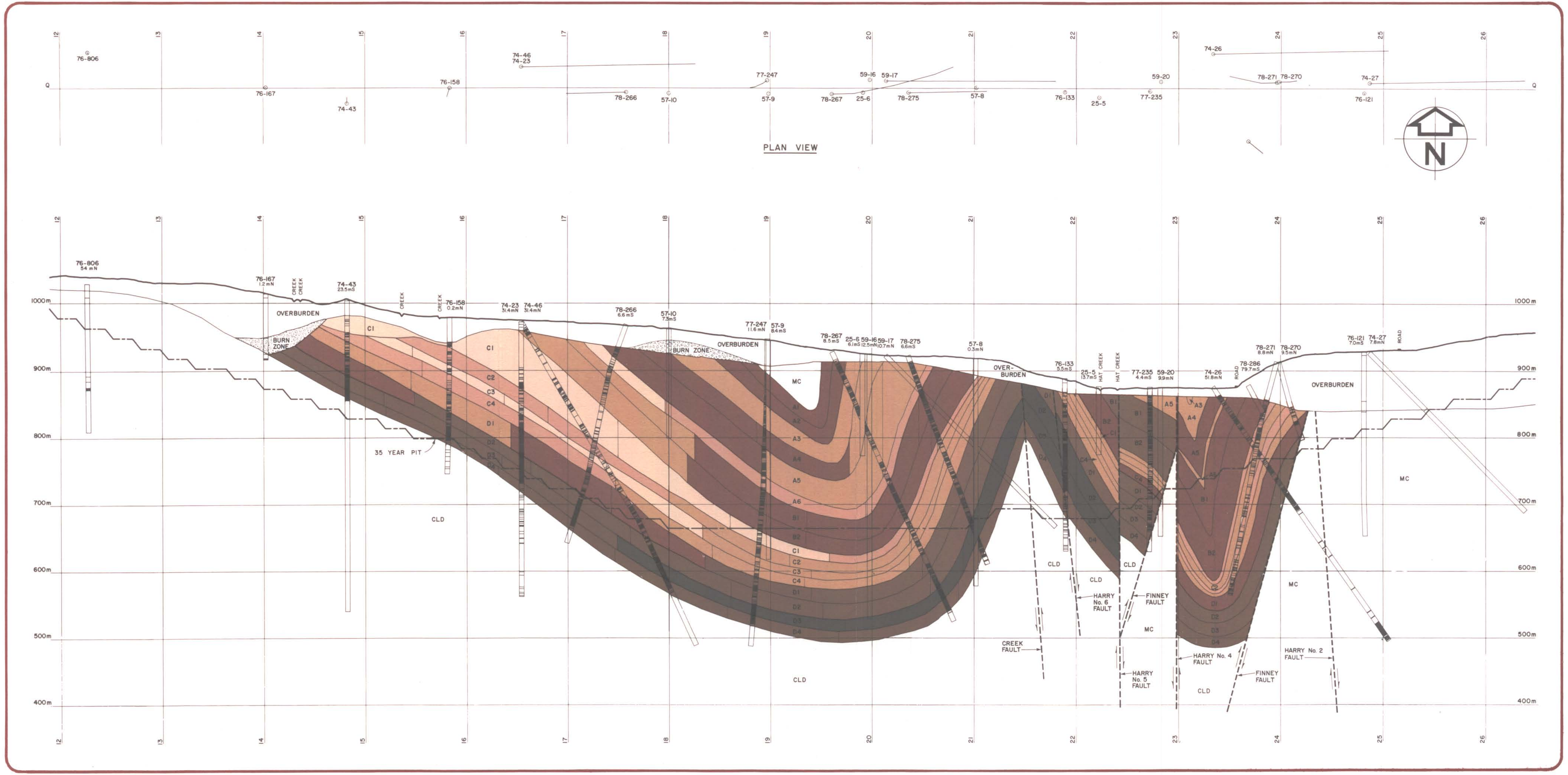
FIGURE 3-1

BEDROCK GEOLOGY HAT CREEK VALLEY

SOURCE : B. C. Hydro and Power Authority December 1980

FIGURE 3-2

Geophysical Log DH 274 showing 16 sub-zones



LEGEND

SPECIFIC ENERGY RANGES
MJ/kg (DRY BASIS)

- < 9.3
- 9.3 - 13.0
- 13.0 - 16.5
- 16.5 - 20.0
- 20.0 - 24.0
- > 24.0

- MC — MEDICINE CREEK FORMATION
- CLD — COLDWATER FORMATION
- BURN ZONE
- A6 — A6 SUB-ZONE
- C1 — C1 SUB-ZONE
- FAULT
- CONTACT
- RELATIVE MOVEMENT

SUBZONE & THICKNESS

A1	15 - 35m	C1	0 - 170m
A2	20 - 55m	C2	5 - 20m
A3	25 - 45m	C3	5 - 15m
A4	20 - 45m	C4	5 - 20m
A5	30 - 45m	D1	15 - 25m
A6	0 - 90m	D2	15 - 30m
B1	25 - 35m	D3	15 - 25m
B2	25 - 35m	D4	15 - 20m

100m 0 100 200 300m

HAT CREEK PROJECT

FIGURE 3.3

GEOLOGICAL CROSS SECTION SECTION Q

SECTION DRAWN LOOKING NORTH

SOURCE: British Columbia Hydro and Power Authority (43)

SECTION 4

MINE PLANNING

4.1 INTRODUCTION

The objective of this section is: to define the coal production requirements that are needed to supply the Powerplant and Coal Liquefaction Plant for the planned project life; to identify the assumptions made; to establish the principal design criteria and the methods of planning the pits. In performing this study, extensive use has been made of previous studies performed primarily on the No.1 Deposit. Many of the design principles and criteria used are drawn directly from B. C. Hydro's Hat Creek Project Mining Report, December 1979.

The plan developed must incorporate adequate safeguards to ensure the safety of the work force. Environmental objectives must be met and adverse impacts reduced as much as possible. Effective utilization of the resource should be maximized. These principles are recognized in this study to the extent possible, but assume more significance in detailed planning and design studies.

Conceptual design studies completed in 1976 identified No. 1 Deposit as being more economical to mine than No. 2. As a result, exploration and evaluation has been concentrated on No. 1 Deposit since that time, so that considerably more data is now available for No. 1 than No. 2 Deposit. Based on this, it was decided that for the purpose of this study No. 1 Deposit would be mined to its practical economic limit and that mining from No. 2 Deposit will be phased in as No. 1 is depleted.

4.2 ASSUMPTIONS

The following assumptions were identified in the Prefeasibility Study Proposal submitted in September 1980 and modified slightly to reflect what was actually done:

(1) The study will be conducted on the assumption that the No. 1 Deposit will be mined to its economic limit and that mining from No. 2 Deposit will be phased in as No. 1 is depleted.

(2) To the extent necessitated by the limited data available on the No. 2 Deposit, design parameters will be projected from the No.1 Deposit.

(3) The study will be based on the work reported in the Hat Creek Project Mining Report - December 1979.

(4) Creek diversion waterflows will be delivered to the streambed of Hat Creek downstream of the mining operation.

(5) B. C. Hydro's proposed 2000 MW (net) powerplant will remain on its present site and in its present form.

4.3 GEOTECHNICS AND HYDROLOGY

Extensive geotechnical and hydrological investigations were carried out between 1976 and 1978 on the No. 1 Deposit by Golder Associates. Although no specific investigations have been conducted on No. 2 Deposit, Golder has examined and interpreted the data that exists in the light of the results obtained from No. 1 Deposit. The purpose of these studies was to identify the geotechnical and hydrological factors that impact upon pit and waste dump design and to recommend design criteria for the pit slope stability, waste dump construction and pit dewatering.

This section was developed based on reports prepared by Golder Associates for B. C. Hydro in 1978 and for the Ministry of Energy, Mines & Petroleum Resources in 1980. These reports are referenced in Section 11 of this report (i.e. references 4 and 13).

4.3.1 Pit Slope Stability

The angle at which pit slopes can be excavated safely has a major impact on the economics of the mining operation. The steeper the slopes can be excavated, the less waste that must be removed. The stability of the slopes is controlled by the strength of the materials and the groundwater conditions in the area. Overall, the materials represent saturated weak rocks that were originally deposited in a lacustrine environment and are softened when wet.

4.3.1.1 Slope Design Criteria

Considering the geology and the geotechnics the present indications are that the materials in the pit slopes around No. 2 Deposit should be treated similarly to those around No. 1 Deposit. Because of the low strength of the claystones and siltstones, structure such as bedding is not expected to be a controlling factor in overall slope stability. In high slopes the materials would behave essentially as engineering soils, and the mode of failure would be generally a circular arc slip. A description of the materials encountered and their properties are presented in Tables 4-1 and 4-2. The recommended pit slope angles for mine planning purposes are as follows:

Surficial deposits (other than slide debris)	25°
Slide debris	16°
Coal	
Coldwater and Medicine Creek Formations, undifferentiated volcanics, volcaniclastic and tuffaceous rocks	20°

As more data is accumulated during future investigations and the early stages of operation, further refinement of the slope angles can be anticipated.

The recommended slope angles are based on the following assumptions:

(1) That pit slopes would only be stable if some means (natural or artificial) is available to depressurize the slopes;

(2) That pit slope depressurization by negative pore pressure generated by stress relief on excavation would be moderately successful;

(3) That slopes would be excavated to flat angles during the initial process of mining, both to minimize shearing stresses that could lead to progressive slope failures and to promote slope depressurization;

(4) That interim bench failures would be acceptable, that increased road maintenance would be necessary, and that wider benches would be needed locally;

(5) That slope height is generally not dependent on slope angle, because the design is based on the lower limiting strength of the material; and

(6) That slopes are designed to be stable only for the duration of mining.

The current pit design involves flatter interim pit slopes than final slopes with the excavation of slopes to final depth being delayed until late in the life of the pit. The BWE mining system should be favourable in that it excavates the pit to full width at a rapid rate to promote depressurization, allows sufficient time for the negative pore pressures to develop and then mines the depressurized material rapidly before the negative pore pressures dissipate.

4.3.1.2 Ultimate Slopes

The eventual dissipation or equilibrium of negative pore pressures may induce slides in the final pit slopes. The process would probably be one of progressive failure, with the back scarp of the slide retreating over many, possibly hundreds of, years until a stable situation is achieved. One way to prevent this would be to back-fill the excavation of the No. 1 Deposit with fill from waste excavated from the No. 2 Deposit when it is mined.

4.3.2 Slide Areas

Several slide areas have been identified in the Hat Creek Valley. The slides have been separated into three categories: stable, marginally stable and active. These relate to present conditions. Movement of these slide masses could be reactivated along preexisting slide planes due to excavation disturbances of their equilibrium, or by water flow or pressure. Experience has shown that movement of these slides would be of a slow, creeping nature.

Slide areas are shown in Figure 4-1.

The active slide on the northwest perimeter of No. 1 Deposit will be stabilized by surficial drainage, diverting Hat Creek and putting in a fill ramp at the toe of the slide across the valley to serve as a bridge for the conveyor and access road to Houth Meadows Waste Dump.

The slide materials are mostly bentonitic clays and volcanic debris or breccia. Over 30 million cubic metres of this material will have to be excavated in No. 1 Pit and an unknown, but substantial, quantity from No. 2 Pit. The material is known to be very sticky and difficult to handle, especially when wet in the spring. It may be impractical to maintain benches in this area for any significant length of time.

The inactive slide masses should be controllable by restricting water inflows through establishing and maintaining a drainage program. It is recommended that the slide materials be cleared back from the rim of the pit and a "creep-monitoring" system established.

4.3.3 Bench Design

Mine benches must be designed taking into account the geological structure; faults and bedding planes.

1. Faults

Where possible, faults are mined in the direction of the dip, so that the zone is traversed as quickly as possible and the fault is first met in the upper part of the face. Removal of weak, faulted ground and unloading of the lower part of the face containing the faults is therefore possible.

The weakest members of the coal sequences are normally the argillaceous interbeds along which tectonic shearing has often developed. The stability of any slope formed in the coal would therefore be dependent on the orientation of the bedding planes in relation to the bench orientation. Local joint sets and unique structures such as faults would cause local stability problems.

This situation is well exemplified in Trench A, where the northern and southern faces were excavated normally to the strike and are stable. The western face was excavated parallel to the strike and is unstable.

2. Bedding Planes

To minimize bench instability along bedding planes when the dip is out of the mining face, the benches should preferably be aligned in such a way that they are not parallel with the strike of the beds, but rather make an angle of at least 20° with that direction.

In the event of the dip of the bedding being less than 30° and out of the face, with the strike of the bedding parallel to or within 20° of the face alignment, the slope of the mining benches should be reduced to the slope of the bedding. This precaution is not necessary where the dip of the bedding is less than 20°.

4.3.4 Hydrology

4.3.4.1 Testing Programs

Extensive hydrological testing programs have been conducted in the Hat Creek Valley, primarily in the area of No. 1 Deposit in order to:

(a) Define the groundwater-pressure regime;

(b) Assess the feasibility of depressurizing the proposed mine slopes by drainage and pumping;

(c) Evaluate the permeability of the materials and their dewatering and recharge characteristics.

The testing programs were:

1. Piezometers

Piezometers have been installed in over 200 holes, many of which have multiple standpipe piezometers. Advantage was taken of holes drilled for coal exploration and geotechnical investigations to install many of these instruments. Sixteen of the holes have had more sensitive pneumatic packer-type piezometers installed in the standpipes to give pressure fluctuation readouts. Records have been accumulated on water levels in the holes; many of these readings have been taken on a monthly basis. Piezometer hydrographs have been prepared and evaluated.

2. Pump Testing

Six pump tests were carried out; designed to assess the geohydrological characteristics of the major stratigraphic units. The pump tests measure the hydraulic conductivity of the material and evaluate the possibility of depressurization (drainability or permeability), and the recharge capability.

3. Falling and Rising Head Tests

These were carried out in piezometers located within specific zones and indicate the hydraulic conductivity of the zone material.

Table 4-3 gives a summary of results of these field tests.

4.3.4.2 Regional Perspective

The topography of the Hat Creek Valley should produce recharge conditions in the higher ground on the valley walls and groundwater discharge conditions near the valley bottom. The western walls of the proposed pits would occupy relatively low ground in the valley bottom. This could result in increasing hydraulic head with depth, which is characteristic of groundwater discharge areas. Groundwater discharge in the form of springs and seeps are common below the 970 m contour in the No. 1 Deposit area, which indicates that the western pit slopes would not be well drained.

The steeper topography underlying the eastern side of the proposed No. 2 Pit coupled with the topographic divide between Ambusten Creek and Hat Creek in this region are likely to provide more complex groundwater behaviour.

4.3.4.3 Properties of Pit Materials

Groundwater behaviour in the No. 2 Deposit area is expected to be similar to that noted in No. 1 Deposit. The materials encountered can basically be divided into three hydrogeological units: the surficial deposits; the coal; and the sedimentary bedrock sequences above and below the coal.

1. Surficials

The surficials are highly variable, ranging from predominantly slide debris and till on the west to the gravels and fine sands on the east of No. 1 Deposit. The glacio-fluvial and alluvial deposits accumulated in the valley bottom have relatively high hydraulic conductivities in the No. 1 Deposit area. It is anticipated that these will provide the greatest dewatering requirement in the excavation of the pit.

Groundwater in storage within the alluvial fan in Ambusten Creek or the small terraced features in the Cashmere Creek Valley could act to enhance recharge to bedrock units beneath the eastern half of the No. 2 Deposit and contribute to stability problems in the development of the potential Ambusten Creek waste disposal area.

2. Coal

The hydraulic parameters of the coal are variable and not easily characterized. Falling head tests suggest that the B and D-zones in No. 1 Deposit are generally four orders of magnitude more permeable than the A and C-zones, probably because of their generally lower ash content and less frequent clay and carbonaceous shale bands. A pump test in the D-zone coal did not suggest good drainability, while a test in a cleaner part of A-zone coal has shown that this unit can be relatively easy to drain in some areas. It must be assumed that No. 2 Deposit parameters will fall in the same range.

3. Bedrock

Tests in the Coldwater Formation (claystone/siltstone/conglomerate) below the coal and in the Medicine Creek Formation (bentonitic claystone/siltstone) above the coal have demonstrated very low hydraulic conductivities and consolidation coefficients. Permeabilities were so low that the planned pumping tests were executed by hand-bailing.

4.3.4.4 Conclusions

The pre-mining water table surface generally parallels the topographic surface and is at or near the ground surface in the Hat Creek Valley. However, in places the piezometric surface is up to 100 m below ground on the eastern side and above ground on the western side of the valley.

Mining consideration has to be given to control of sliding, or potential sliding, by means of preventive rather than remedial action. Mine planning has to include considerable work to achieve control by two processes: drainage dewatering and unloading. The drainage should be done as early as possible before mining starts. "Unloading" should be considered part of the overall mine planning when stripping and slope angles are being assessed; the degree of negative pore pressure response will become apparent after several years of mining have taken place.

Hydraulic conductivities of all the zones in the pit area are very low except in the surficial materials (gravel and sand overburden). Permeability of the bedrock zones and the coal was so low that no pumping could be done; hand-bailing methods were used.

In general, depressurization by dewatering is not likely to be effective in these bedrock zones; pumping and drainage cannot be relied on to reduce the pore pressures in working slopes, because the ground is too impermeable.

Piezometric response data before and after the pumping test showed that there was a general downward movement of groundwater from the surficial sediments and through the overlying siltstone/claystone into the more permeable coal units.

Hydraulic conductivity values for lithologic units, while all low, have differences that might be related to formation facies variations and possibly to structural features such as faults and joints.

It is likely that for the weaker rocks the distribution of the clay fraction within the materials controls the hydraulic conductivity. Figure 4-2 shows the variations.

4.4 DESIGN CRITERIA

4.4.1 Coal Requirements

Annual coal production requirements for the project have been established that reflect:

(a) The initial commercial production dates of the various units.

(b) The anticipated on-stream factors of the plants.

(c) Coal requirements for building stockpiles, testing and commissioning operations and an allowance for sub-standard operating efficiency during early operation.

4.4.1.1 Unit In-service Schedule

(a) Powerplant

500 MW Unit 1 August 1987 - Year 1
" Unit 2 August 1988 - Year 2
" Unit 3 August 1989 - Year 3
" Unit 4 August 1990 - Year 4

These in-service (commercial operation) dates are preceded by a 6-month commissioning period.

(b) Coal Liquefaction Plant

Train No. 1 January 1988 - Year 2
Train No. 2 September 1988 - Year 2

4.4.1.2 On-stream Factors

(a) Powerplant

The Powerplant will operate at the following capacity factors expressed as a percentage of maximum capacity rating:

Years 1- 4 65%
Years 5-25 70%
Years 26-35 55%

(b) Coal Liquefaction Plant

The Coal Liquefaction Plant is expected to operate for 340 days (8160 hours) per year at 100% of rated daily output. During the first two years of plant operation this capacity has been reduced further to 70% (Year 2) and 85% (Year 3) of the normal output to allow for the usual problems of early operation.

4.4.1.3 Coal Requirements Schedule

The coal requirements schedule at 18.1 MJ/kg, dry coal basis is based upon the following:

(a) Powerplant 500 MW (net) unit consumes 450 t/h at full output.

(b) Gasifier plant consumes 41,900 t/stream day at full output.

(c) Steamplant consumes 12,000 t/stream day at full output.

(d) Commissioning allowance of 1.1 Mt for each 500 MW unit and each half of the Steamplant.

(e) For mine production scheduling purposes an additional allowance - usually 1 Mt per year is made to provide sufficient coal to compensate for miscellaneous losses and possibly lower than expected coal quality.

The coal production requirements schedule is shown in Table 4-4.

4.4.2 Material Characteristics

4.4.2.1 General Description

Open-pit mining of the Hat Creek coal deposits will generally be in relatively weak soft rocks and surficial materials. With the exception of a limited area of volcanic rock overlying the No. 2 Deposit, the coal beds will be the strongest members of the whole strata of sedimentary beds intersected by pit excavations. However, even the coal beds cannot be considered as a competent rock because it varies from hard to soft types depending on its clay content. See Tables 4-1 and 4-2.

Pit excavation will primarily involve the following four major types of materials:

Unconsolidated:	Surficial deposits	- glacio-fluvial sands and gravels;
	Slide debris	- breccia, volcanic debris, bentonitic clays;
Consolidated:	Coal beds	- in-situ coal zones;
	Coldwater rocks	- bedrock clay, waste rocks.

Moisture content of the materials to be mined will generally remain close to the in situ saturation level because of their low permeability. Bench faces may develop a skin dryness but this is only expected to penetrate to a maximum depth of one metre after a year of exposure. This point must be recognized and provided for in the design of material handling systems in particular.

4.4.2.2 Specific Gravity and Swell Factors

In the course of the exploration programs over 5000 samples of a wide range of coal and waste materials were tested for specific gravity. Extensive statistical analysis of the test results established that for coal there is a distinct relationship between ash and specific gravity. (Specific gravity (coal) = 1.21104 + 0.00738 x %Ash).

The wide variety of waste materials and the lack of precision in the quantity estimates for each make detailed analysis and correlation pointless at this stage.

The following specific gravity criteria were selected for this study:

Coal	1.50
Surficials and Waste Rock	2.00
Burn Zone	2.16
Volcanics	2.00

Based upon limited field measurements and judgement the swell factors selected for use in this study were:

Material	As Mined	Dumped in Stockpiles
Coal	35%	35%
Granular Surficials	20%	15%
Cohesive Surficials	30%	25%
Bedrock Waste	30%	25%

The most critical area for the application of these swell factors is the provision of adequate waste disposal space. For this reason excess waste dump capacity must be provided to ensure feasibility of the scheme.

4.4.2.3 Mining Implications

1. Diggability

Based upon practical experience gained during the 1977 Bulk Sample Program and the testing of materials, it has been concluded that most of the materials to be encountered can be excavated without prior blasting by either a bucketwheel excavator or a hydraulic shovel. The exceptions which could require blasting are calcite cemented conglomerate underlying the coal, volcanics and burn zone material. The surface on the west side of No. 1 Deposit has a number of glacial drift boulders which would have to be removed by other methods before the BWE could mine the area.

The climatic impact upon the material characteristics, especially the freezing of saturated clays, changes the diggability. In general this is not expected to be a problem as long as a working face is kept active; but opening up a new face could require some blasting or ripping.

2. Trafficability

Bentonitic clay is prevalent in active and inactive slide areas and in the upper-zone partings in the west and southwest of No. 1 Deposit. It is anticipated that similar occurrences will occur in No. 2 Deposit. Bentonite boils occur at several locations along the west side of the valley. This type of clay absorbs moisture, swells, and becomes extremely sticky and slippery, which presents problems in the construction and operation of haul roads.

Because of the generally weak nature of the materials, most areas of the pit will require that haul roads be constructed with a good sub-base and gravel topping. Special efforts will be required to prevent the accumulation of clay on the road surface because of the hazard to traffic when it becomes wet.

Conversely the impact of low winter temperatures will improve the trafficability and permit access to and mining of some of the more unstable materials.

4.4.3 Selective Mining

The Hat Creek coal deposits are unique, because of the immense thickness of the coal formation, which is due to the existence of a favourable depositional environment for an extended period of time. However, this period of coal deposition was frequently interrupted by episodes of flooding, which introduced non-carbonaceous sediments into the basin. These sediments produced waste partings, usually clay, in the coal sequence. The break between coal and clay is not generally sharp, but includes a transition zone which grades from good coal through a phase where the coal and clay materials combine to form a low-grade coal (silty coal), to a succeeding phase where the clay predominates (carbonaceous claystone), and finally to the clay.

These periodic inundations were particularly significant during the deposition of the A and C coal zones. The C-zone depositional environment appears to have been particularly turbulent, judging by the widespread occurrence of the lower grades of coal and the relative absence of substantial bands of good quality coal. The A-zone was deposited in an environment that alternated between relative calm and severe flooding. This has resulted in bands of good coal interbedded with clay grading to coaly shale. Within the A-zone 20 of these interbeds, ranging in thickness from 2 metres to 10 metres, have been identified. The D-zone coal was deposited during a stable period. Few waste partings were formed and the best, most consistent quality of coal, is contained in the D-zone. The B-zone was also deposited under relatively stable conditions although there were a few incursions of sediment-laden floods to produce some waste bands.

Similarly, within the predominantly waste zones, there are occasional bands of acceptable coal.

The larger waste and low-grade partings are simple to identify and easily mined as waste material. The smaller partings, up to 5 metres, are more readily mined with the coal. However, while this simplifies the mining process, it reduces the quality of the coal produced by the mine.

The separation of these smaller partings from the coal improves fuel quality. This is the selective mining process.

Preliminary studies were conducted to assess the impact on coal quality of the exclusion of waste bands varying in thickness from 0.5 metres to 5 metres. These studies indicated that significant improvements in fuel quality could be obtained with selective mining. This improvement would be particularly significant in the A-zone. In the C-zone the quality improvement would be small, but more coal would be recovered. Overall, the indications were that as much, or more, total heat content could be recovered depending on the size of parting that could be removed.

Experience gained during the Bulk Sample Program excavating the coal with a hydraulic shovel established that this type of equipment can selectively mine Hat Creek coal. During this test program, a hydraulic shovel with a 3 cubic metre bucket was able to aggregate partings one metre thick. This separation is possible primarily because of the difference in the physical characteristics between the coal which is hard, and the partings which are soft. After exposure to the atmosphere for a week or two, sufficient drying of the coal face occurs to highlight the colour differences between coal and waste. This assists in the identification of the different materials. Observation of larger hydraulic shovels with 10 cubic metre buckets at other mining operations indicates that the digging action of these machines will permit selective mining of partings

1.5 metres to 2 metres thick without reducing equipment productivity. Blasting is not compatible with selective mining because it loosens and mixes the coal and partings, destroying the physical differences that are essential to success.

Based on this evaluation of selective mining methods, it was concluded that partings 2 metres thick and greater can be segregated effectively without significantly reducing equipment productivity or increasing mining costs. In practice, it will often be possible to mine selectively bands less than 2 metres, depending on their position and attitude. With the BWE it will be possible to reduce the band size to 1 m or less.

During operation, careful control must be exercised to ensure the success of selective mining. Closely spaced sample holes must be drilled ahead of mining, to permit local correlation of coal quality for short-term mine planning. This must be supplemented by detailed geological mapping of the exposed coal faces.

These comments are primarily related to the No. 1 Deposit, where studies indicate that fuel quality can be improved by over 5% using selective mining techniques without decreasing the recovery of the resource. This improvement is reflected in the quality figures reported. For No. 2 Deposit insufficient data is available to establish the degree of improvement that can be achieved. However, it is to be expected that improvement in No. 2 Deposit coal quality will be realized by selective mining and the application of bucketwheel excavators.

The interpretation of geophysical logs indicate that there are more coaly claystone partings in the deposit than were identified in earlier sampling programs or incorporated into the evaluation. This provides scope for further improvement in run-of-mine coal quality during operation.

For this study it has been assumed that selective mining techniques will be employed to remove bands of waste and low grade coal down to 2 m in thickness.

4.4.4 Waste Dumps

4.4.4.1 Available Space and Requirements

Over the lifetime of the project in excess of 2 billion bank m^3 of waste materials will be removed from the mine. In addition the Powerplant and Coal Liquefaction Plant will produce close to 250 million tonnes of ash. Provision must be made to dispose of these waste products safely and in an environmentally sound manner.

Five locations have been identified for waste storage for this project. They are, with their approximate maximum capacities:

Houth Meadows	542 Mm^3
Medicine Creek	1030 Mm^3
Anderson Creek	760 Mm^3
Ambusten Creek	200 Mm^3
No. 1 Pit	1000+ Mm^3
Total	3500+

Estimated waste disposal space requirements are:

No. 1 Deposit (1012 M bank m^3)	1265 Mm^3
No. 2 Deposit (1021 M bank m^3)	1276 Mm^3
Powerplant	68 Mm^3
Coal Liquefaction Plant	133 Mm^3
Total	2742

Clearly, adequate space is available to dispose of the total waste to be produced. The assignment of waste to the various dumping locations is governed by economic, design and scheduling considerations.

4.4.4.2 Dump Design

Two general categories of waste materials have been established:

(1) Unstable and very weak bentonitic claystone and siltstones, and weak silty and clayey sedimentary deposits. These materials would remain in an unconsolidated condition for many years, and their shearing resistance would be that of a partially saturated material in an undrained condition. They will therefore need to be retained by well-engineered embankments;

(2) Stable and relatively stronger material consisting primarily of sand, gravel, and till. These materials are suitable for embankments as well as for construction of roads, yards, and as concrete aggregate.

Three principal issues related to dump stability have been subjected to geotechnical tests and evaluation resulting in the following conclusions:

1. The Stability of Retained Waste

As the dumps must be considered on the basis of long-term stability at maximum capacity, they must be located in relation to the walls of the ultimate pit. Field and laboratory tests were performed, including an examination of the characteristics and stability of a trial waste dump on site. From these it was concluded that the retained waste can be kept stable, whether saturated or unsaturated by keeping it within the recommended surface slope of 5%. This slope could be increased as more experience regarding slope stability is gained.

2. The Stability of Retaining Embankments and Their Foundations

The embankments must be free-draining and constructed entirely of well-graded and fairly clean sand and gravel. To remain stable, they must be uncontaminated by bentonitic clays, and be designed within a safety factor to hold the retained waste when either in a saturated or a fluid state. The recommended overall slopes for the embankments are 2.5 horizontal to 1 vertical on the outside face, and 1:1 on the inside face.

3. The Gross Interaction of Waste Dumps and Pit Slope Excavations

The Houth Meadows Dump is sufficiently close to the pit for the stability of the dump and the pit slope to be considered as a unit. A north-east to south-west-trending conglomerate ridge has been identified west of the pit. This would form a buttress and provide additional support to the dump.

The Medicine Creek Dump would be within 600 m from the pit rim of the ultimate pit. Present studies indicate that the sequence of granular rocks underlying the Medicine Creek embankment would provide adequate long-term support to the proposed dump.

The Anderson and Ambusten Creek dump sites are approximately 1 km away from the rim of the proposed No. 2 Pit. The possible embankment sites for both dumps require geotechnical investigation of the foundation conditions. Possible interaction between the Ambusten dump and the pit slopes of No. 2 Pit also needs to be investigated. Consideration of the impact of a larger and deeper excavation of No. 2 Deposit on the safety of the waste dump is also required.

Backfilling No. 1 Pit after the coal is extracted to its economic limit would improve the long term stability in the pit, particularly along the west side and the north-west corner which provides a buttress for the Houth Meadows dump. This also provides the most economic alternative for disposal of waste from No. 2 Pit.

4.5 PIT DESIGN

The approach to pit design differs for the two deposits primarily because of the significant difference in the quantity of data available and the previous work done on No. 1 Deposit that could be reused. The basic assumption followed was that No. 1 Deposit would be mined first to its practical economic limit and that the rest of the coal needed to provide the project coal requirements defined in Section 4.4.1.3 would come from No. 2 Deposit. Optimization of the pit designs must await the acquisition of adequate data on No. 2 Deposit and more detailed planning of the mining sequence in the feasibility stage.

4.5.1 No. 1 Deposit

The first step in the design was to establish the total quantity of coal that could be mined practically and economically. This step was performed using a computer model which defines the deposit in blocks that are 50 m square in plan and 15 m high. Each of the 196,000 blocks in the model is defined with its waste volume, coal tonnes and quality. A block of coal this size represents approximately 55,000 t.

A series of incremental pits were "mined" by the computer. Each increment was approximately 50 million tonnes. The "mining" process is simulated by a series of inverted, truncated cones with the cone angles matching the overall pit slope criteria. Each pit increment is evaluated based on a value assigned to the heating value of the coal less the costs of mining. If the mining costs are set at a high level, only the most profitable blocks will be mined. The cost constraint can then be progressively relaxed to permit the mining of further increments. Similar constraints can be imposed on the stripping ratio and the maximum acceptable average coal quality. In this application, the exercise in relative economics was continued through twelve pit increments. The first seven increments produced results compatible with previous design. The results presented in Table 4-5 show that the succeeding stages have rapidly increasing incremental stripping ratios. This situation obviously reaches an extreme in the twelfth stage with an incremental stripping ratio of 17.85 m^3 of waste per tonne of coal. The exercise was terminated at this point. Even in the eleventh stage with its incremental stripping ratio of 6.62:1 it is questionable whether this increment should be mined; but for the purposes of this study because the cumulative stripping ratio is still reasonable it has been accepted. This decision must be reexamined in future studies.

In summary, the No. 1 Deposit will be mined to produce 566.4 M tonnes of coal requiring the removal of 1012 Mm^3 of waste producing an overall stripping ratio of 1.79 m^3/tonne of coal. This pit establishes the boundaries for the design of the mining systems and the basis for the development of a practical production schedule.

4.5.2 No. 2 Deposit

The No. 2 Pit was designed to produce the 340 M tonnes of coal required to fill the balance of the project total requirement of 907 M tonnes. The pit design was developed using manual methods.

A series of cross sections were developed based upon the geological interpretation of the available drill hole data. The pit was laid out respecting the geotechnical pit slope criteria and adjusted by trial and error to produce a pit that yielded sufficient tonnage at the lowest stripping ratio. This pit yielded the required quantity of coal at a stripping ratio of 3:1 (m^3/tonne).

The quality of coal in the design pit was established by weighting the drill hole intersections of the pit by their length. This approach yielded the following mean quality: HHV 17.3 MJ/kg, 35.5% Ash, and 0.66% Sulphur on a dry coal basis. The expected moisture content would be 23%.

The available quality data is the result of analysis performed on rather large sample intervals which masks the potential for selective mining and quality improvement. If the experience with No. 1 Deposit holds in No. 2 the mean heating value could be expected to improve by 0.5 to 1.0 MJ/kg.

4.6 COAL PREPARATION AND PRODUCT SPECIFICATIONS

4.6.1 Product Requirements

The following size specifications are required for the products delivered to the different plants:

(1) Gasifiers

Top size 100 mm; preferably -75 mm
Nominal bottom size -6 mm
Product to contain less than 2% of -2 mm fines.

(2) Powerplant and Steamplant

Top size 50 mm.

In addition it was considered desirable to maintain the quality of the fuel supplied to the Powerplant as close as possible to the existing Boiler Fuel Specification: HHV 18.1 MJ/kg, 33.5% Ash, 0.51% Sulphur all on a dry coal basis and 23.5% moisture.

4.6.2 Coal Screening

Two streams of coal will be produced from the mine and must be handled and processed separately. The two streams are produced from different zones and have different quality and physical characteristics. The processing to be performed on the coal is a dry screening, crushing and blending operation. This process was selected over the more efficient wet screening process in order to avoid the following problems:

(a) Excessive breakdown of the coal through the leaching of thin clay bands.

(b) The cost and difficulty of settling clay from the wash water.

(c) Increased moisture content in the product which reduces heat conversion efficiency and would be expected to cause coal handling problems.

To offset these problems the wet screening process removes most of the very fine particles which are predominantly clay, leaving an improved product. The selection of the optimum screening process is an important area for testing and evaluation in the bulk sample/feasibility stage.

The dry screening process selected capitalizes on the fact that when Hat Creek coal is screened a higher percentage of ash reports to the undersize fraction. The impact is more marked when screening lower grade coal than with better quality coal. Based upon several sets of screen analysis test data a table has been developed that relates the percentage of the total feed retained on a screen to the reduction in ash content. The ash content for the undersize can be calculated. See Table 4-6.

Example: Head ash = 39.4%; 60% of feed retained.
(all figures are on a dry coal basis)
Oversize % ash = 39.4% - 5.9% = 33.5%
Undersize % ash = $39.4\% + \frac{60}{40} \times 5.9\% = 48.25\%$

Table 4-6 is considered effective in the range of 30-50% head ash. For better quality coal such as the D-zone coal limited data is available so an interpolation of the necessary points has been made.

In designing the screening process the critical factor, which must be verified by further testing, is the selection of the minimum size screen that can effect the size separation without plugging up. For this study the minimum screen size selected is 13 mm.

4.6.3 Process Design

Two streams of -400 mm run-of-mine are delivered by the overland conveyors to the Crushing and Screening House. Each stream is scalped at 100 mm with the oversize being crushed and recombined with the undersize. This material is then split at 13 mm to provide four products from the plant for blending or delivery to the three plants. Table 4-7 gives the predicted size consist of the run-of-mine coal.

The results produced by screening the two streams at 13 mm is as follows:

Stream 1: Mixture of A, B and C zone coal represents 60% of the deposit to be mined. When screened at 13 mm will produce 60% +13 mm and 40% -13 mm.

Feed Analysis (dry coal basis): 16.1 MJ/kg; 39.4% Ash; 0.65% Sulphur

Coarse Product (+13 mm) Analysis (dcb): 18.1 MJ/kg; 33.5% Ash; 0.65% Sulphur

Fine Product (-13 mm) Analysis (dcb): 13.1 MJ/kg; 48.25% Ash; 0.65% Sulphur

Stream 2: D-zone coal represents 40% of the deposit to be mined. Screened at 13 mm will produce 70% +13 mm and 30% -13 mm.

Feed Analysis (dry coal basis): 21.2 MJ/kg; 24.6% Ash; 0.30% Sulphur

Coarse Product (+13 mm) Analysis (dcb): 21.4 MJ/kg; 23.8% Ash; 0.30% Sulphur

Fine Product (-13 mm) Analysis (dcb): 20.5 MJ/kg; 26.5% Ash; 0.30% Sulphur.

4.6.4 Product Specifications

Using the results obtained in the previous section, a series of blended products were designed to meet the requirements defined in Section 4.6.1. The description of the physical plant to accomplish this is presented in Section 6.3. The product specifications were developed based on a total mine coal production of 30 million tonnes per year at an average heating value of 18.1 MJ/kg, dry coal basis.

4.6.4.1 Gasifier Coal

The gasifier coal blend was developed by assigning all of the A, B and C +13 mm fraction and sufficient of the D +13 mm fraction to meet the total requirement of the sum of volatile matter and fixed carbon established by Fluor Corporation in their study.

The result was a requirement for 14.25 million tonnes of coal at 18.92 MJ/kg (dry coal basis). A detailed product specification is presented in Table 4-8.

4.6.4.2 Powerplant Coal

The remaining D +13 mm coal plus sufficient of the D -13 mm and A, B and C -13 mm fractions to meet the Boiler Fuel Specification heating value of 18.1 MJ/kg were assigned to the Powerplant blending operation. This has resulted in a small reduction in the sulphur content of this fuel from 0.51% to 0.44% (dry coal basis).

4.6.4.3 Steamplant Coal

The remainder of the D -13 mm and A, B and C -13 mm fractions were assigned to the Steamplant blending operation. The resulting fuel has a mean heating value of 16.1 MJ/kg (dry coal basis). A more detailed product specification is presented in Table 4-8.

In order to meet these product specifications it will be essential to:

(a) Maintain a proper balance between the Coal Liquefaction Plant and the Powerplant coal consumption.

(b) Exert tight quality control on the mining operation.

(c) Design and control the operation of a complex coal preparation and blending system.

<u>TABLE 4-1</u>

DESCRIPTION OF ROCK MATERIALS

HAT CREEK COAL LIQUEFACTION PROJECT - PREFEASIBILITY STUDY 1980 - MINING

TYPE	DESCRIPTION	LOCATION	RANGE OF HYDRAULIC CONDUCTIVITY m/sec	GEOTECHNICAL COMMENTS	MOISTURE CONTENT ON DRY WEIGHT BASIS	UNIAXIAL STRENGTH	ATTERBERG LIMITS
Till	Glacial deposit composed of cobbles and gravels with occasional boulders up to 1 m dia. maximum but generally much less, in a matrix of sand, silt and clay. Locally variable, depending on matrix. Seen in base of Clay-Cut.	West and southeast sides of valley	10^{-10}-10^{-8}	Generally dense or compact, boulder size may locally inhibit digging although usually will be able to be dug by hydraulic excavator. Where gravelly, may make water.	15% - 50% Average 26%	0 - 300 kPa	LL = 86 PL = 42 (avg. from a small number of tests)
Lacustrine Deposits	Bedded silts, silty sand with coarse sand and occasional gravel may be also clayey, laminated and/or highly disturbed. Overconsolidated. Glacial origin.	Locally through-out glacial deposits. Houth Meadows embankment foundations.	10^{-7}-10^{-6}	Unusually dense. Where laminated, easy to dig but uniform heavily overconsolidated silts of Houth Meadows could give difficulties. Surface materials in Dry Lake and Houth Meadows are soft.	18% - 32% Average 25%	200 - 500 kPa	LL = 48 PL = 26 (avg. from a small number of tests)
Glacio-fluvial Deposits	Interbedded rounded-subrounded sands and sandy gravels with cobbles and boulders up to 0.7 m dia. (approx.). Much variation in grading. Some interbedded tills. Glacial meltwater deposit.	East side of valley, locally on west also.	10^{-7}-10^{-5}	Dense, possibly slightly cemented, free draining. Will not generally present digging problems. Boulder size smaller than till. Rounded materials. Some ironpans present.	Depends on drainage	non-cohesive	Non-plastic
Colluvium	Coarse, angular, roughly bedded perhaps with variable proportion of fines formed on slopes by erosion. May comprise volcanics, limestone or granodiorite.	Widespread at base of steeper slopes.	10^{-7}-10^{-4}	Variable depending on local rock type. Angular, abrasive, maximum rock size large although generally gravel to cobble sizes. Free draining, locally unstable during digging.	11% - 60% Highly dependent on composition average 30%	100 - 500 kPa, depending on composition	Varies over full range because of composition variability.
Slide Debris (Stable)	Composed of variable assortment of glacial and glacio-fluvial materials Coldwater sediments and granodioritic material often in a bentonite matrix. Seen in upper part of Trench A and Clay-Cut. Mostly post glacial.	West side of valley especially NW.	not known	Variable. Generally moderately dense. Handling characteristics similar to Clay-Cut material.	11% - 60% Highly dependent on composition average 30%	100 - 500 kPa, depending on composition.	Varies over full range because of composition variability.
Slide Debris (Active)	As above, but some softer zones. Currently unstable.	Active slide in NW and minor slides elsewhere in W.	not known	Broken locally softened and weak rock probably sticky. Some seepages. Contains some proportion of gravel. Could give some handling and trafficking problems. Occasional boils.	11% - 60% Highly dependent on composition average 30%	100 - 500 kPa, depending on composition.	Varies over full range because of composition variability.
Alluvium	Rounded sands and gravels probably with silt interbeds as seen in Trench B. Mostly reworked glacials.	Predominantly in Hat Creek Valley bottom.	10^{-6}-10^{-4}	Generally loose and free draining. Maximum size say 0.4 m. Gravel subsidiary to sand.	Depends on drainage	Usually not cohesive	Usually non-plastic but could go up to about LL = 40, PL = 15 (no test results).
Burn Zone	Varies from an irregular mass of red-brown partly-fused claystone and siltstone with some coal to well bedded slightly baked in situ Coldwater materials.	Dry Lake area. May be obscured by glacial or slide deposits in subcrop on W. side.	highly variable	Hard abrasive generally breaking up into gravel sized fragments, easy to dig. Difficult or impossible to dig where completely fused (as in part of Trench A). Some blasting locally necessary.	Insufficient data for characterization; properties highly variable.		

Source: Golder Associates

TABLE 4-2

DESCRIPTION OF SURFICIAL MATERIALS

HAT CREEK COAL LIQUEFACTION PROJECT - PREFEASIBILITY STUDY 1980 - MINING

TYPE	DESCRIPTION	LOCATION	RANGE OF HYDRAULIC CONDUCTIVITY m/sec	GEOTECHNICAL COMMENTS	MOISTURE CONTENT ON DRY WEIGHT BASIS	UNIAXIAL STRENGTH	ATTERBERG LIMITS
Claystone/ Siltstone	Very weak to moderately weak clayrich rocks in which bedding often hard to discern. Rock breaks along joints. Where softened or reworked, material highly plastic and tenacious. Zones of shearing and brecciation. Possibly tuffaceous near margins of basin. Generally dark grey or dark brown colour. Distinct tuff bands present.	Stratigraphically above the coal (Unit Tcu). Subcrops in an arc from NE to SW in final pit slopes.	10^{-12}-10^{-10}	Should be considered as a hard clay rather than a rock for excavation purposes. Easily dug where joints are present. Very uniform beds may be troublesome to hydraulic excavator. Handling and trafficking problems will occur in wet conditions due to presence of montmorillonite. Only slakes where sheared or brecciated.	13% - 32% Average 24% May tend to decrease with depth from 29% at subcrop to 18%, 150 m deeper.	400 - 12,000 kPa Average 3,700 kPa May tend to increase from 1,000 kPa to 8,000 kPa after 150 m.	LL = 95 PL = 35 (average)
Siltstone/ Sandstone	Interbedded siltstone and sandstone with subsidiary conglomerate, claystone and coals. Generally light grey in colour, highly anisotropic but bedding planes often difficult to find. Much facies variation.	W and NW pit slopes. Stratigraphically below the coal. Also occurs as interbeds in the conglomerate.	10^{-11}-10^{-10}	Should be considered as a stiff clay rather than a rock for excavation purposes. Easily dug where joints are present. Handling and trafficability problems will occur in wet conditions due to presence of montmorillonite. Dispersive, highly erodible, will form gullies, and sub-surface cavities. Slakes readily.	23% - 70% Average 31%	600 - 3,500 kPa As interbeds in conglomerate, 3,500 - 7,000 kPa	LL = 143 PL = 34 (average)
Sandstone	Varies from weak silty sandstone through to moderately strong fine grained conglomerate. Matrix usually composed of silt/clay and granular material may be tuffaceous and weak. Locally cemented especially immediately below the coal. Generally greenish.	W and NW Pit slopes. Stratigraphically below the coal. Forms interbeds in Lower siltstone/sandstone (Unit Tcl) and in conglomerate (Unit Tco_1).	10^{-10}-10^{-9}	Generally weak rock whose excavation characteristics may differ little from the siltstones. Some trafficking problems as material breaks down. Often highly bentonitic. Characterized by west face of Trench A.	19% - 32% Average 25%	Some tendancy to increase from 1,000 kPa at surface to 10,000 kPa at 300 m depth. Interbeds in conglomerate range from 3,500 kPa to 10,000 kPa and vary similarly with depth.	LL = 80 PL = 30 (based on only a few results)
Conglomerate	Highly variable in character depending on relative proportions of granular material and matrix. Coarse gravel fragments rounded to subrounded but also angular where tuffaceous. Matrix may be bentonitic. Often clacite cemented. Not yet seen in outcrop or excavation. Contains interbeds of siltstone and sandstone.	S abutment of Houth Meadows Embankment. Forms ridge between Houth Meadows and pit (Unit Tco_1). Also occurs as interbeds in Lower siltstone/sandstone (Unit Tcl) and at base of whole sequence (Unit Tco_2).	10^{-10}	Harder and more abrasive to dig. Where weathered could be disaggregated and behave as gravel. Will break down with much rehandling except where cemented. Calcite cemented conglomerate could not be dug without blasting.	Average 15%, based on few test results. Note that interbeds will raise overall average.	Depends on cementation; up to 43,600 kPa has been measured locally. Some zones almost uncemented.	LL = 60 PL = 27 (based on very few results)
Coal	Thinly bedded moderately strong but highly fractured. Interbedded with siltstone partings and beds, often highly sheared. Some cleating. Much variation from clean to dirty coal except in D-Zone. Some zones of complete fragmentation.	Centre of pit and limited area in SW wall.	10^{-11}-10^{-6}	Easily dug due to multitude of weak joints and partings. Bench failures common especially where bedding unfavourably oriented. Seepages from face, generally no sizable water inflows.	See DCA report	1,000 - 17,000 kPa	See DCA report
Coal Interbeds	Generally thinly bedded claystone/siltstone of moderate plasticity. Some bentonitic material in A-Zone and near margins of basin. May be highly sheared or brecciated.	Centre of pit and limited area in SW wall.	10^{-12}-10^{-10}	Easily dug and similar to coal in some respect although will not break up as much. Impermeable locally softened. Thinner beds may be difficult to separate from coal.	12% - 36% Average 23%	No data	LL = 59 PL = 33 (average)
Volcanics	Includes an assortment of basalts, dacites, rhyolites, agglomerates, breccias and tuffs. Closely jointed.	E and W of pit.	10^{-11}-10^{-6}	May require blasting or ripping. Generally hard and abrasive. Permeable. Generally drained.	No data	Up to 23,000 kPa has been measured. Strength may often be much greater.	N/A
Limestone	Massive or brecciated limestone with phyllite interbeds.	Underlying Houth Meadows.	10^{-9}-10^{-4}	Will require blasting. Generally strong phyllite bands weaker. Dry.	No data	No data	N/A

Source: Golder Associates

<u>TABLE 4-3</u>

HYDRAULIC CONDUCTIVITY RESULTS

HAT CREEK COAL LIQUEFACTION PROJECT - PREFEASIBILITY STUDY 1980 - MINING

LITHOLOGIC UNIT		HYDRAULIC CONDUCTIVITY RESULTS FROM FALLING HEAD TESTS (m/s): NUMBER OF TESTS	RANGE FROM	RANGE TO	MEDIUM VALUE	PUMPING TEST RESULTS: HYDRAULIC CONDUCTIVITY (m/s)	PUMPING TEST RESULTS: COEFFICIENT OF CONSOLIDATION (c_v) (m^2/yr)
Upper Siltstone-Claystone	(Tcu)	13	$1x10^{-12}$	$3x10^{-8}$	$1x10^{-10}$	$1x10^{-10}$ (W76-1) $4x10^{-11}$ (W77-4)	< 10 (W77-4) 400* (W77-3)
A zone siltstone and coal	(Tcc)	6	$1x10^{-11}$	$3x10^{-10}$	$4x10^{-11}$	$9x10^{-9}$ (W78-2)	-
B zone coal	(Tcc)	3	$2x10^{-7}$	$5x10^{-7}$	$4x10^{-7}$	-	-
C zone siltstone and coal	(Tcc)	13	$3x10^{-11}$	$3x10^{-8}$	$1.4x10^{-10}$	-	-
D zone coal	(Tcc)	12	$6x10^{-9}$	$1x10^{-6}$	$5x10^{-7}$	$6x10^{-11}$ (W77-1)	< 45 (W77-1)
Lower Siltstone-Sandstone-Conglomerate	(Tcl)	15	$2x10^{-11}$	$5x10^{-9}$	$8x10^{-11}$	$5x10^{-12}$ (W77-2)	500* (W77-2)
Conglomerate	(Tco_1)	4	$9.5x10^{-11}$	$2.9x10^{-9}$	$1.3x10^{-10}$	-	-
Limestone		7	$1.2x10^{-9}$	$1x10^{-4}$	$3x10^{-8}$	-	-
Basalt		5	$2.3x10^{-11}$	$1.8x10^{-4}$	$7x10^{-9}$	-	-
Greenstone		5	$4x10^{-10}$	$5x10^{-7}$	$1.8x10^{-7}$	-	-

* These values were calculated using some assumptions and may be rather high.

Source: Golder Associates

TABLE 4-4

COAL PRODUCTION REQUIREMENTS SCHEDULE
(M tonnes)

Coal Liquefaction Project-Prefeasibility Study 1980-Mining

Period (Years)	Powerplant	Coal Liquefaction Plant			Total Coal Consumption	Stockpile Inventory	Loss Allowance	Total Coal Produced
		Gasifiers	Steamplant	Total				
1	2.2*	-	1.1*	1.1	3.3	0.5	1.0	4.8
2	4.7*	6.6	3.2*	9.8	14.5	0.2	1.0	15.7
3	7.5*	12.1	4.1	16.2	23.7	0.1	1.0	24.8
4	10.0*	14.2	4.1	18.3	28.3	-	1.0	29.3
5-25	11.0	14.2	4.1	18.3	29.3	-	0.7	30.0
26-31	8.7	14.2	4.1	18.3	27.0	-	1.0	28.0
32-35	8.7	-	-	-	8.7	(0.1)	-	8.6
Total	342.4	416.3	123.2	539.5	881.9	0.4	24.7	907.0

*Includes commissioning allowances.

TABLE 4-5

NO. 1 DEPOSIT INCREMENTAL PITS

Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

Pit Number	CUMULATIVE Coal tonnes x 10^3	H.H.V.	Waste bm^3 x 10^3	Strip Ratio	Incremental Strip Ratio
1	50,130	18.94	42,288	0.84	
2	100,344	18.57	82,263	0.82	0.80
3	160,472	18.42	143,268	0.89	1.01
4	210,950	18.11	202,549	0.96	1.17
5	261,024	18.06	268,233	1.03	1.31
6	313,023	18.10	322,560	1.03	1.04
7	359,486	18.06	374,919	1.04	1.13
8	410,533	17.97	454,938	1.11	1.57
9	465,873	17.77	548,038	1.18	1.68
10	516,359	17.54	680,777	1.32	2.63
11	566,397	17.66	1,011,900	1.79	6.62
12	617,793	17.60	1,929,487	3.12	17.85

TABLE 4-6

CORRELATION OF DRY SCREENING DATA
Coal Liquefaction Project - Prefeasibility Study 1980 - Mining

Cumulative Weight	A, B & C Zones Cumulative Ash %
15%	(head ash - 10.87)
20%	(head ash - 10.17)
30%	(head ash - 8.83)
40%	(head ash - 7.74)
50%	(head ash - 6.84)
60%	(head ash - 5.90)
70%	(head ash - 4.76)
80%	(head ash - 3.36)
90%	(head ash - 1.73)
100%	(head ash)

Source: Simon-Carves of Canada Ltd. 1979

TABLE 4-7

DRY SCREEN ANALYSIS

Predicted Size Consist Run-of-mine Coal

Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

Top Size	A B & C Zone 200 mm % Weight	D Zone 200 mm % Weight
+100 mm	5	8
100 x 50 mm	10	19
50 x 25 mm	18	23
25 x 13 mm	26	19
13 x 6 mm	15	15
6 x 3 mm	10	6
3 x 1.5 mm	7	4
1.5 x 0.6 mm	4	3
0.6 x 0	5	3

Source: Simon-Carves of Canada Ltd.

TABLE 4-8

COAL QUALITY SPECIFICATIONS

Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

	STEAMPLANT COAL		GASIFIER COAL	
	As Received	Dry Basis	As Received	Dry Basis
Proximate Analysis				
Total Moisture %	22.5	-	24.0	-
Volatile Matter %	23.4	30.2	25.8	34.0
Fixed Carbon	23.5	30.3	26.5	34.8
Ash	30.6	39.5	23.7	31.2
Ultimate Analysis				
Carbon %	32.0	41.3	36.56	48.1
Hydrogen %	2.60	3.35	2.81	3.70
Nitrogen %	0.67	0.88	0.67	0.88
Chlorine %	0.03	0.03	0.02	0.02
Sulphur %	0.39	0.51	0.43	0.57
Ash %	30.6	39.5	23.7	31.2
Oxygen (by difference)%	11.21	14.43	11.8	15.53
HHV MJ/kg	12.48	16.10	14.38	18.92

Mineral Analysis of Ash %	Steamplant Coal	Gasifier Coal
SiO_2	51.8	-
Al_2O_3	28.9	-
TiO_2	1.0	-
Fe_2O_3	8.6	-
CaO	3.1	-
MgO	1.7	-
Na_2O	1.78	-
K_2O	0.80	-
P_2O_5	0.25	-
SO_3	1.80	-
Mn_3O_4	1.13	-
V_2O_5	0.06	-

Hardgrove Grindability Index 45-50

Ash Fusibilities, °C	Range	
IT (Reducing)	1140-1500+	-
ST	1190-1500+	-
HT	1230-1500+	-
FT	1270-1500+	-
IT (Oxidizing)	1300-1500+	-
ST	1330-1500+	-
HT	1360-1500+	-
FT	1390-1500+	-
250 Poise Viscosity, °C	1500	-
Na_2O, % Dry Coal Basis	0.43	-
K_2O, % Dry Coal Basis	0.07	-

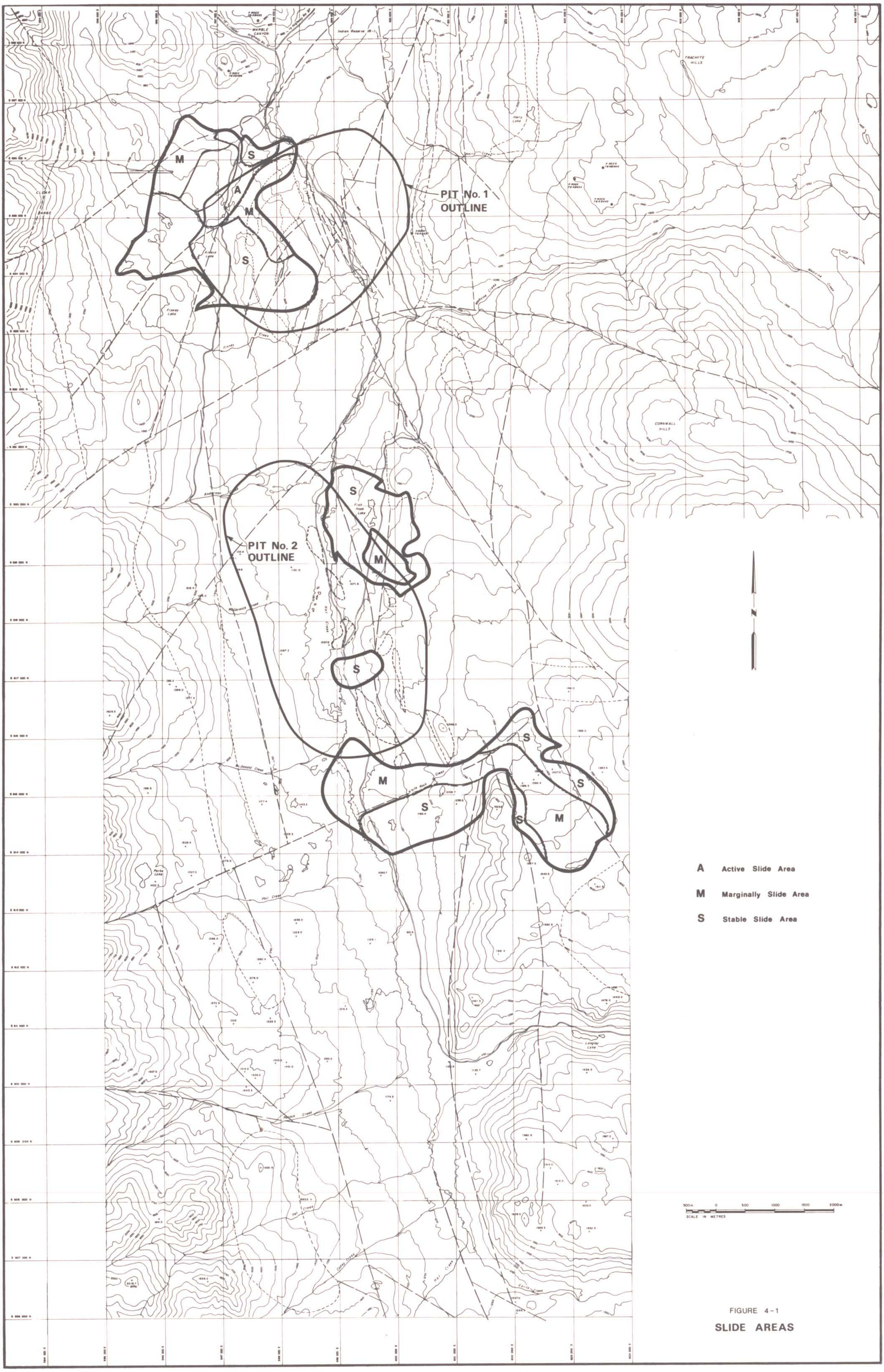


FIGURE 4-1
SLIDE AREAS

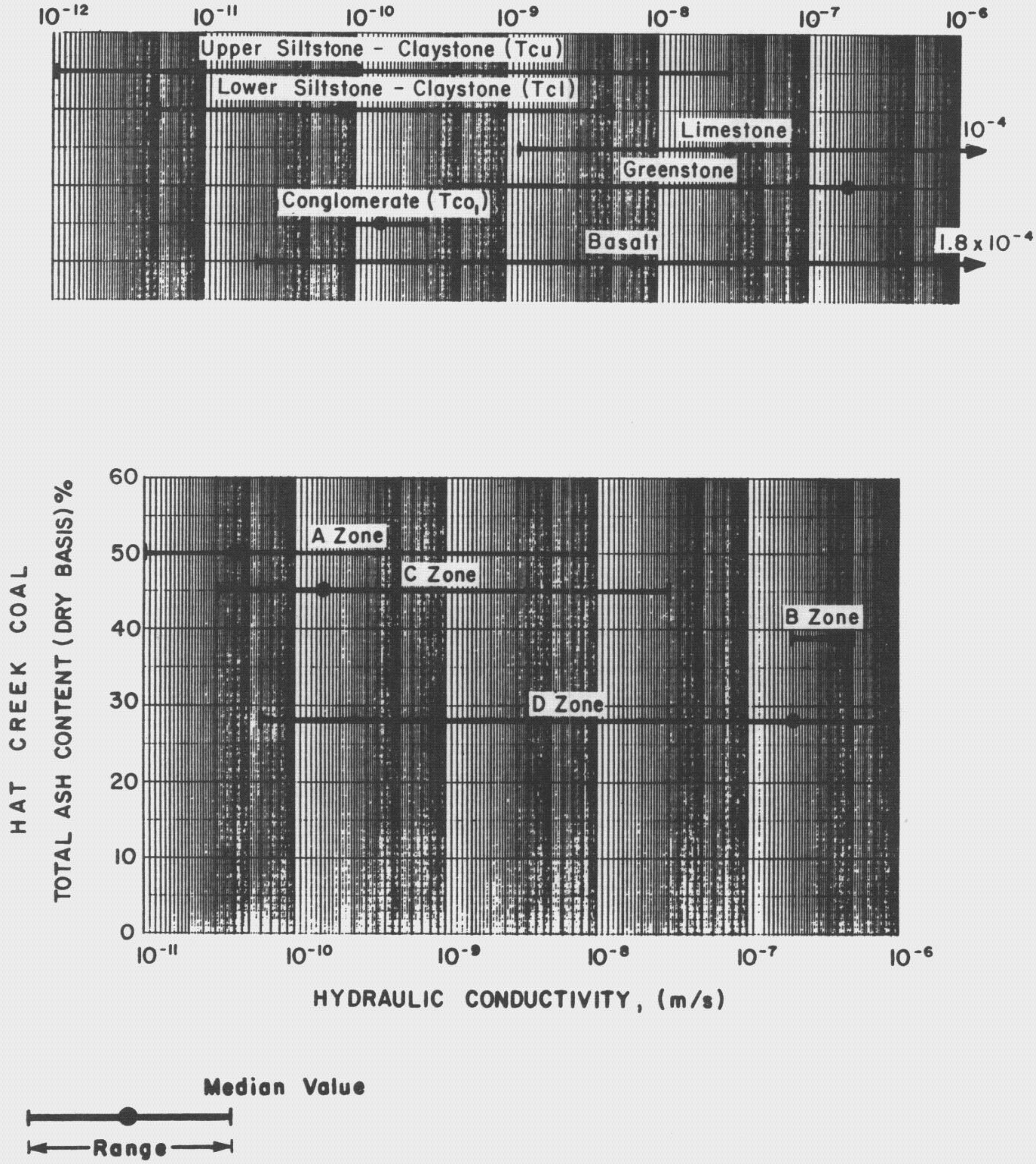


FIGURE 4.2

HYDRAULIC CONDUCTIVITY VALUES DETERMINED IN-SITU

SECTION 5

MINE OPERATION

5.1 GENERAL APPROACH

It is planned to mine the No. 1 Deposit first, with mine development commencing in Year -1 and coal production starting in Year 1 at 4.8 Mt and rapidly building up to 30 Mt in Year 5. Production continues at this level through Year 19 and then tapers off with coal extraction being completed in Year 21. Development of the No. 2 Deposit commences in Year 16 to remove approximately 150 Mm^3 of waste material prior to the start of coal production in Year 20. The coal requirements for the remaining years of the project life are supplied from No. 2 Deposit. Table 5-1 presents the production schedule for the project.

The selection of the bucketwheel excavator-conveyor as the major mining system was made because it is the only high-productivity system that can be applied to deposits of this size and configuration. This is not a textbook BWE application: the limited area of the deposits and their great depth requires careful sequencing of the operation and periodic relocation of the systems to different mining benches. Use of trucks and shovels to move such high volumes of material would require: very large fleets of equipment causing severe congestion in the pit; a much larger workforce; and consume large quantities of diesel fuel.

In developing the plan of mine operation for this study it has not been possible to evaluate the quality of coal that will be produced on an annual basis. This would have required considerably more time than was available for this study in the case of No. 1 Deposit and insufficient exploration data is available on No. 2. For No. 1 Deposit it has been assumed that with the quantity of coal scheduled for removal by supplementary shovels and trucks this combined system will be sufficiently flexible in its application to overcome any problem in meeting quality specifications.

5.2 CREEK DIVERSION

5.2.1 General

Hat Creek is the central watercourse in the Hat Creek Valley and is fed by numerous tributary streams draining an area of some 363 km^2. The creek flows through both the No. 1 and No. 2 Deposit areas and must be diverted around the coal deposits before any significant mine development can begin.

For much of the year Hat Creek is a rather small stream, having an average annual discharge of 0.67 m^3/s, that in a dry year can dry up completely. During the annual snowmelt period considerably greater volumes of water flow and major diversion works must be installed to handle these quantities. B. C. Hydro's Hydroelectric Generation Projects Division have completed a prefeasibility study on the necessary works. This prefeasibility study was based on the premise that all waters collected and diverted around the mine would be returned to the Hat Creek streambed downstream of the mine.

It is understood that the diversion of water from the Upper Hat Creek Valley into Oregon Jack Creek is being studied on behalf of the Ministry of Environment. Should that project proceed no serious conflict is anticipated. It is likely that some benefits could be gained by integrating the two projects.

The Hat Creek diversion scheme would be developed in two stages. The first stage to bypass No. 1 Deposit would be constructed during the pre-production period and become operational late in Year -2. If the construction period is extended the completion of Stage 1 should be advanced by at least one year and possibly two. The second stage around No. 2 Deposit would be completed in Year 15 to permit development of the deposit to commence in Year 16.

5.2.2 Stage 1

The Stage 1 diversion facilities, as shown in Figure 5-1 would comprise a headworks dam immediately downstream of Anderson Creek; 3.8 km of diversion canal generally following the 975 m contour on the

East side of the valley; 2.4 km of concrete lined tunnel adjacent to Pit No. 1; 460 m of open channel; 2.1 km of buried multiplate, corrugated steel conduit returning the diversion flow to Hat Creek; a pit rim dam, pumphouse and pipeline immediately upstream of Pit No. 1 to collect precipitation and discharge it to the canal; and a 2.2 km canal and outlet works to divert Finney Creek into the headworks reservoir.

The Hat Creek diversion works were designed to handle, as an emergency condition with minimum freeboard, a 1000-year snowmelt flood discharge of 27 m^3/s and, as a normal operating condition, a 100-year snowmelt flood discharge of 18 m^3/s. For the Probable Maximum Flood (PMF) having an estimated peak of 106 m^3/s, spillway facilities were provided to permit discharge into the pit since it was considered that the diversion works would have been too costly if designed for the PMF. The spillway facilities were designed for a discharge of 79 m^3/s, the difference between the PMF and the canal capacity.

5.2.3 Stage 2

The Stage 2 diversion facilities as shown on Figure 5-2 would comprise a small headworks dam near the point where the old Hammond Diversion via Oregon Jack Creek was made; 11.3 km of diversion canal on the West side of the valley; 2.5 km of buried conduit discharging into the Stage 1 headoworks reservoir; and a pit rim dam, pumphouse and pipeline immediately upstream of Pit No. 2.

The general principles of the Stage 2 facilities are similar to Stage 1. Items worth noting include:

(a) Major embankments for the canal would be required for crossings at Pocock, Phil and McDonald Creeks.

(b) The No. 2 pit rim dam drains an area of about 30 km^2, approximately 10 times as great as the area for No. 1 pit rim dam. This dam is located about 1 km upstream of No. 2 Pit. Because of the depth of permeable materials in this area it is proposed to keep the reservoir pumped down as much as possible to reduce seepage into the pit. A substantial pumping installation is provided for.

5.3 MINING METHODS

5.3.1 Pre-production Development

Pre-production development is scheduled to start immediately upon receipt of construction authorization which is assumed to be at the beginning of Year -3. Much of this work can proceed in parallel with work on the Creek Diversion; but there are a number of key items that cannot be undertaken until the diversion is operational. The most critical of these items, and the most difficult to schedule around is the construction of the water treatment and holding lagoons. Should the current tight schedule be maintained very careful planning and scheduling will be required in the feasibility stage.

The first task associated with mine development is the establishment of a mine area drainage and dewatering scheme. Numerous small lakes and ponds along the west side of the valley must be drained to reduce the risk of potentially unstable slide masses being activated. This will be achieved through a network of ditches. In addition a series of dewatering wells must be established to reduce groundwater flows in the pit when excavation starts. Groundwater removal also helps to improve pit slope stability, but needs as much time as possible to be effective. Water from the drainage and dewatering program will require at least treatment in sedimentation lagoons prior to discharge. This will entail the construction of temporary lagoons to be operated until the permanent lagoons are constructed.

Site preparation and construction of the mine support and surface facilities will take place between Year -3 and Year -1. These facilities include the lagoons, the conveyor distribution station, the overland conveyor, the crushing and screening plant, the blending stockpile area, and the necessary maintenance shops, warehouse and offices.

Mine development starts in Year -2 with excavation for the first conveyor loading station and to prepare the bench for the first BWE which is scheduled to start mining in Year -1. The pit development begins close to the centre of the deposit in the valley bottom where the coal comes to within 10 to 15 metres of the surface. The waste materials excavated during this phase are used to build haul roads from the initial pit to Houth Meadows Waste Dump 2.5 km away and 1.5 km to the No. 1 Loading Station. In addition, conveyor ways must be constructed from the West side of the deposit to the distribution point and from the distribution point to the waste dump areas. When these tasks are completed, the bases for the waste retaining embankments at Houth Meadows and Medicine Creek must be constructed.

5.3.2 No. 1 Deposit

The development of No. 1 Deposit commences in Year -2 and the economic resource is mined out in Year 21. Over this twenty three year period 566 million tonnes (378 Mm^3) of coal and 1012 Mm^3 of waste will be mined. Peak annual production rates are over 75 Mm^3 of total material to be mined and transported. This quantity of material will be mined with a combination of bucketwheel excavator and truck-shovel systems with the BWEs handling the majority of the work.

The shovel-truck system will be used primarily in areas where the BWE cannot operate efficiently and also to provide mobility and flexibility in operation to assist in the delivery of consistent fuel quality. Trucks and shovels will be used initially to develop the mine: prepare BWE benches; construct roads and conveyor ramps; and construct the base of waste embankments. They will also be used to remove material on the west side of the pit above the 1025 m elevation, which has insufficient quantity to warrant using the BWE. As the BWE become established the trucks and shovels will concentrate on deepening the centre of the pit.

The pit is divided into two along the valley bottom. This line coincides with the point where the eastern limb of the main syncline is steeply dipping and close to surface as is the faulted anticline structure. In this area, the attitude of the coal seams exposes several different qualities of coal in a limited area, which should permit the blending of a uniform quality of coal from the start of commercial production.

The initial development with the BWE is primarily concentrated on the west side of the deposit. A total of five BWE in the 60,000 m^3/day class are proposed. This rating is the production capability operating for 24 hours at 80% operating efficiency, which only allows for routine servicing, changing teeth and the normal minor operating delays and inefficiencies that occur. No major repairs or weekly service day are allowed for. For the purpose of this study it has been assumed that each BWE is capable of operating 4000 hours per year at 3000 m^3/h, or 12 million m^3/year. This assumption may be on the conservative side but it reflects all the downstream delays in the system and allows that a BWE may be shut down when fully available for such reasons as meeting short term coal quality targets or adverse climatic conditions. It is expected that on some occasions the BWE could be called upon to run 5000 hours in a year.

The selected BWE system shown in Figure 5-3 includes the following equipment:

(1) BWE 2300: 5 x 32; buckets 2.3 m^3, machine can excavate 5 m below its working elevation and 32 m above.

(2) 1800 mm wide conveyor belts transporting the material to the conveyor distribution point at the pit head.

The sequence of mine development which is shown in Figure 5-4, is based upon the first BWE being commissioned at the start of Year -1 with the other four following at six month intervals.

Sequence of BWE operations:

(1) Year -2: shovel-truck system prepares 980 W and 935 W benches for installation of conveyor sytems and space for BWE start.

(2) Year -1: BWE 1 starts excavation on 980 bench; continues to Year 7; total material excavated 87 Mm^3.

(3) Year -1: shovel-truck system prepares 890 W and 935 E benches.

(4) Year -1 (midyear) BWE 2 starts excavation on 935 W; continues to Year 12; 140 Mm^3.

(5) Year 1 BWE 3 starts excavation on 890 W; continues to Year 14; 164 Mm^3.

(6) Year 1 (midyear) BWE 4 starts excavation on 935 E; continues to Year 5; 44 Mm^3.

(7) Year 2 BWE 5 starts excavation on 845 W; continues to Year 15; 160 Mm^3.

(8) Year 5 BWE 4 moves to 800 W; continues to Year 15; then moves to No. 2 Pit.

(9) Year 7 BWE 1 moves to 755 W; continues to Year 15; 115 Mm^3.

(10) Year 12 BWE 2 moves to 845 E; continues to Year 18; then moves to No. 2 Pit.

(11) Year 14 BWE 3 moves to 845 E; continues to Year 20; then moves to No. 2 Pit.

(12) Year 15 BWE 1 moves to 755 E; continues to Year 19; then moves to No. 2 Pit.

(13) Year 15 BWE 5 moves to 800 E; continues to Year 19; then moves to No. 2 Pit.

In examining this sequence it should be apparent that every effort is made to minimize the number of moves made by the BWE system. The five BWE systems installed in No. 1 Deposit are involved in a total of six moves within the twenty years of operation. In the production schedule each move is assumed to lose the total output of the system for 4 months. The major time loss in this outage is caused by the relocation of the conveyor system, which must be dismantled, relocated and reassembled. The physical relocation of the BWE to the new bench could be accomplished in a day or two. It would be normal to take advantage of this extended outage to perform any major overhaul work required or substitute the machine for another one requiring overhaul.

The development sequence for the mine is based on each BWE taking a total cut height of 45 m. This height would be taken on three stages as shown in Figure 5-3:

(1) High cut 25 m: machine travels 10 m above conveyor elevation.

(2) High step 10 m: machine travels at conveyor elevation.

(3) Deep step 10 m: machine travels 10 m below conveyor elevation; cut is taken behind the conveyor.

When these three stages are completed, the conveyor is advanced towards the face and the process repeated.

In the course of mining each BWE would excavate coal, waste suitable for embankment construction and unsuitable waste. This provides a considerable challenge to the mine planner who must not only provide a supply of each type of material when required by the schedule but must also produce coal of a reasonably consistent, acceptable quality on a day to day basis. At this level of study it is not possible to confirm the ability to meet coal quality specifications in the short term. Taking a broad view it is reasonable to conclude that suitable coal quality control can be achieved by having access to a wide range of coals at any time through the exposure of numerous coal quality subzones.

The truck-shovel development is not presented in detail because of the much greater flexibility of this system. The equipment selected for application in the mining of No. 1 Deposit is:

(1) 14.5 m^3 hydraulic shovels similar to the Demag 241. Annual capacity (coal) 4.04 Mm^3. Fleet size: 2-5 units.

(2) 154-t trucks will be used starting in the pre-production period when most of the material to be hauled is waste over long haul distances. Fleet size 7-9 units.

(3) 91-tonne trucks will be phased in to replace the larger units starting in Year 3, when the emphasis shifts more to coal and the shorter haul distances.

The truck-shovel system will work 15 m high benches. All material below the 755 elevation is scheduled for mining with this system together with that required for mine development and pit deepening operations. Approximately a quarter of the total material is scheduled for removal by truck and shovel. With the exception of certain waste materials, such as that above 1025 elevation, which will be hauled directly to the Houth Meadows Waste Dump, all materials mined by truck and shovel will be hauled to the central conveyor ramp, dumped through a 400 mm square grizzly and loaded onto one of two conveyors for transport to the conveyor distribution station. The dump station grizzlies will be equipped with hydraulic hammers to break up oversize material. Lumps that cannot be broken will be removed with a clamshell and trucked away. As the pit is deepened, the conveyors are extended and over the life of the pit three additional dump stations are installed.

This approach minimizes the truck haulage costs. Where practical, the aim is to position the dump station so that coal is hauled no more than 2 benches (30 m) up or down. Where there is insufficient tonnage to justify the construction of the dump station the longer hauls must be accepted. Such a situation exists towards the end of the pit life with coal coming from the bottom benches.

Figure 5-5 showing the ultimate No. 1 Pit identifies the BWE and shovel-truck benches as well as truck dump stations etc.

In addition to the principal production equipment, a substantial fleet of support equipment: bulldozers; scrapers; graders; 32-t trucks; front end loaders; drills; cranes; service vehicles etc.

This equipment will be used for road construction and maintenance; pit clean up, conveyor moving and general maintenance. Table 5-2 summarizes the peak equipment requirements.

5.3.3 No. 2 Deposit

Pre-production development of the No. 2 Deposit commences in Year 16 with coal production starting in Year 20 and continuing until termination of the project in Year 35. The plan presented shows a rapid phasing out of production from No. 1 Deposit and rapid build up for No. 2. The rate of transition should be examined more closely in the feasibility and detailed design stages of the project. Over the twenty year period of development and production 340 million tonnes of coal are produced and 1021 million m^3 of waste removed. The peak annual production rates are over 100 Mm^3 of materials.

In the No. 2 Deposit the BWE systems are planned to move essentially all of the materials with trucks and shovels used in a support role for development work and handling material unsuitable for BWE mining. The major application identified for the truck-shovel system, other than development work, is the removal of an area of volcanic material in the southeast area of the pit. Considering the present state of knowledge of this deposit it is possible that other areas will be found that require drilling and blasting and should be mined with trucks and shovels. All material mined with the shovel-truck system will be hauled to a dumping point at the north end of the pit. This entails truck hauls up to 3-4 km, which makes trucks in the 154-t size range appropriate.

The five BWE used in No. 1 Deposit will be relocated to No. 2 as they become available. Two additional machines in the 100,000 m^3/day class are scheduled to become operational in Year 17. The general method of operation is the same as in No. 1 Deposit with each BWE feeding a conveyor train which transports the material to a conveyor distribution point at the pit head. Because the pit shape that is the most economic for producing 340 M tonnes of coal is long and narrow, a higher frequency of BWE relocation will be necessary. At the end of its planned life, the pit is open to expansion laterally and in depth. As the pit deepens more coal becomes available on each bench because of the anticlinal shape of the deposit.

Individual BWE complete their assignments from Year 30 to Year 35. In the case of the five BWE purchased initially for No. 1 Deposit they can be considered to have expended their useful lives, although they should still be capable of producing for many additional years. The two larger machines used only in No. 2 Deposit should have many useful years left. Re-sale opportunities are likely to be very limited.

5.3.4 Waste Disposal and Handling

5.3.4.1 General

Approximately 80% of the total material mined during the life of the project is waste. Because of the great thickness of the coal deposits waste material must be transported out of the pit to a waste disposal area, which is typically 6-8 km away from the point of excavation. Because of the topography in the Hat Creek Valley, the waste disposal areas available require that waste be elevated typically 200-400 m above the pit head elevation. The power requirements for raising material 1 m is approximately 20 times that required for the same horizontal travel. This demonstrates the magnitude of the waste disposal problem and indicates the necessity for optimization and careful planning of the waste disposal operation. Optimization has not been attempted in this study. Decisions have been made based on judgement and experience.

5.3.4.2 Assignment of Waste Disposal Areas

The waste material from No. 1 Deposit will be placed in the Houth Meadows and Medicine Creek waste disposal areas. These areas are closer to the mine mouth and are at a lower elevation than the Anderson and Ambusten Creek areas. The east end of the Medicine Creek area is set aside for the disposal of Powerplant and Coal Liquefaction Plant ash. The calculated storage volume requirements for waste from No. 1 Deposit and ash is 1439 Mm^3 against a combined design capacity of 1572 Mm^3 for Houth Meadows and Medicine Creek. Because of the limited data on which the swell factors are based, this small difference should be kept in reserve. There is a further reserve available in Medicine Creek, which has not been designed to the limit of the dump design criteria.

For the waste from No. 2 Deposit the backfilling of the No. 1 Pit is clearly the most economic approach: it is at a lower elevation; its shape and size are conducive to the placement of large volumes before the equipment must be relocated; using the confinement of the pit walls could make the deposition of thicker lifts possible. The use of No. 1 Pit for waste disposal appears to be mandatory because alternatives with sufficient capacity are not apparent.

During the development phase of No. 2 Deposit, it has been assumed that waste must be disposed of in either the Anderson or Ambusten Creek areas. With detailed planning and scheduling it may prove possible to backfill in one part of No. 1 Pit while mining in another. The waste storage volume requirements for No. 2 Deposit to the end of Year 21 is approximately 350 Mm^3. Because the Ambusten Creek disposal area cannot contain this volume, it has been assigned to Anderson Creek. A split between the two areas should be considered in optimization studies. Waste should be routed to the No. 1 Pit as soon as possible.

5.3.4.3 Waste Dump Development and Construction

The initial stages of waste dump development are the preparation of the base for the waste retaining embankments, the installation of the conveyor system to deliver the waste and the assembly of the waste stacker for placing the material.

The location of the embankment base will be cleared of vegetation and any unstable soil. The base will be constructed of free draining sand and gravel delivered by trucks and will be compacted. In the Medicine Creek dump this base will be built to the 1040 elevation to fill the narrow valley and provide a practical working area for the stackers. Lagoons will be constructed at the "toe" of each waste embankments to collect any leachates.

The waste conveyor systems will be installed running from the conveyor distribution station to each stacker location. For the mining of No. 1 Deposit three waste disposal systems will be installed: two for Medicine Creek and one for Houth Meadows. An additional two systems will be installed for the mining of No. 2 Deposit. Each system will consist of a series of 2400 mm wide overland conveyors, a shiftable conveyor on the dump, a belt tripper to transfer the material from the shiftable conveyor to the stacker, an an ARS 2400 (60 + 90) 27 stacker. Each system is capable of handling the output from two of the 60,000 m^3/day BWE or one of the 100,000 m^3/day BWE.

In operation the spreader will start dumping waste from the initial location of the shiftable conveyor. The first spreading pass will be on the downhill side of the conveyor, where a 20-m lift will be placed bringing the filled area up to the elevation of spreader tracks. This lift will be levelled and its surface compacted by bulldozers to prevent moisture penetration. This operation continues until the spreader has completed placing the lower lift. The spreader is then relocated to the uphill side of the shiftable conveyor, where it places a 15-m lift of waste above its operating elevation. When this upper lift is completed, the shiftable conveyor is moved towards the embankment on top of the previously placed 20-m lift. The new location for the conveyor is not closer than 25 m to the crest of the fill.

The cycle is then repeated with the placing of the lower 20-m lift, then the upper 15-m lift, followed by advancing the conveyor. This process continues with general mine waste until the Conveyor-Spreader System reaches the upstream face of the embankment.

When the Conveyor-Spreader System reaches the embankment, the operation continues in the same manner, but the materials transported and placed must be the approved construction materials: sand and gravel uncontaminated by bentonitic clays. On completion of the embankment section of the 35-m lift, the shiftable conveyor system is dismantled and re-erected on a new conveyor pad constructed at the planned elevation of the next lift.

Waste dump construction is illustrated in Figure 5-6.

The face of the embankment must be trimmed to the designed 2.5:1 slope ready for revegetation. Revegetation of the face and surface of the dumps will be carried out progressively as the areas become available through the life of the operation. This dumping sequence prevents the ponding of water between the general mine waste and the embankment. Routine grading of the dump surface and ditching will be required to collect surface runoff and direct it into the main drainage treatment and disposal system.

The stackers included in this system have the capability of constructing the dumps in thicker lifts: up to 70 m thick using a lower lift of 45 m and an upper lift of 25 m. This would clearly be advantageous if it could be achieved; however, at this time there are concerns about the waste material being sufficiently stable to permit lifts this thick. Further studies are required to resolve this.

5.4 MINE DRAINAGE

Despite being located in a semi-arid area (average annual precipitation = 317 mm/a), mine drainage and dewatering represents a significant aspect of the project. The drainage plan for this project must meet the following objectives:

(1) Keep the mine sufficiently dry to ensure continuous operation;

(2) Prevent flood damage to excavation, facilities and equipment;

(3) Ensure the stability of slopes and embankments;

(4) Protect the environment by providing for the continuity of existing streams, preventing the discharge of harmful water-borne contaminants and ensuring that all applicable regulations are observed.

5.4.1 Mine Water Sources

5.4.1.1 Direct Precipitation and Runoff

Annual precipitation at the mine site is low, averaging 317 mm/a of which 55% is received as rain and the balance as snow. Summer and winter are the wettest seasons with spring and fall being somewhat drier. Approximately 25% of the precipitation appears as direct runoff because of high infiltration and evapotranspiration capacity. Mining activity would be expected to result in increased runoff.

5.4.1.2 Creeks, Lakes and Ponds

The control and diversion of the principal creeks in the Hat Creek Valley has been discussed in Section 5.2.

Most lakes and ponds occur along the west side of the Valley. There are approximately 80 small lakes and ponds in the area of No. 1 Deposit which are believed to contribute to the instability of the slide masses. The majority of these lakes and ponds, including Aleece Lake and probably Finney Lake, must be drained at the outset of the project to improve pit slope stability on the west and southwest pit walls. The 15 to 20 small lakes and ponds in Houth Meadows area will also require draining prior to being covered with waste. In the No. 2 Deposit area there are also a number of ponds and Fishhook Lake to be drained prior to development in that area.

5.4.1.3 Groundwater

The glaciofluvial and alluvial deposits will be the principal source of groundwater to be removed. Groundwater flows from the coal and bedrock are expected to be minimal due to the low permeability of these members.

The two main aquifers in the No. 1 Deposit area are a small alluvial aquifer along the central valley and a buried bedrock channel on the east side. The Marble Canyon aquifer is not expected to influence mining operations.

5.4.1.4 Mine Wastewater

The principal sources of mine wastewater are:

1. Effluent from the Mine Services area.

2. Runoff and leachate from coal handling areas and waste dumps.

3. Runoff and seepage from coal and bedrock strata in the pit.

These water flows require separate handling and treatment due to the predicted high levels of dissolved salts. Leachate quantities are expected to be small due to the low permeability of the materials.

5.4.2 Mine Drainage System

5.4.2.1 System Philosophy

The philosophy for the conceptual design of the Mine Drainage System is to provide for the handling of separate streams of differing quality in order to maintain the highest standards of effluent water quality with a minimum of treatment. Essentially this requires that significant quantities of water will not be mixed with flows of lower quality.

5.4.2.2 Perimeter Drainage

The perimeter drainage system is designed to prevent runoff originating outside the project area from entering. Perimeter drains will be constructed around the Houth Meadows, Medicine Creek and Anderson Creek waste disposal areas and around the two pits. The majority of this flow will be delivered to various points in Hat Creek or its diversion works without treatment. In areas where there is heavy traffic the flow will be directed to sedimentation lagoons.

5.4.2.3 Slide Area Drainage

Perimeter ditches will be dug to minimize the recharging of the groundwater by surface runoff. The lakes and ponds will be drained by deepening existing natural drainage channels. Additional ditches will be dug where necessary.

5.4.2.4 Groundwater

A series of dewatering wells will be established to dewater the surficial materials in advance of mining. It is expected that this water will be discharged to Hat Creek via sedimentation lagoons. Groundwater remaining after pumping will be handled by the pit drainage system.

5.4.2.5 Pit Drainage

The pit drainage system must handle surface runoff and mine seepage water. Two separate systems will be established in each pit: one to handle waters in the surficial materials; the other to handle water in the coal and bedrock areas of the pit.

In the surficial areas, which are expected to produce the greatest water quantity, gravity drainage will be used where possible. Water originating at lower elevations will be collected in small pump sumps and pumped up to the gravity drainage system. This water will be discharged to Hat Creek via sedimentation lagoons.

Runoff and seepage in the coal and bedrock areas will be collected in sumps near the main conveyor inclines. A major pumping system will be installed to transfer this waste to the leachate storage lagoons.

5.4.3 Water Disposal

The drainage system proposed will produce three different qualities of water requiring separate handling, treatment and disposal.

1. Diverted Water

This includes both the major creek diversions and perimeter drainage. With proper design of the works there would be no project impact on the quality of the water. This water would be discharged to Hat Creek untreated.

2. Surficial Water

This includes surface runoff, seepage and groundwater pumped from the surficials, as well as other waters requiring treatment in sedimentation lagoons prior to discharge.

3. Mine Wastewater

This includes runoff and seepage from the coal and bedrock areas of the pit, waste dump and coal pile leachates, etc. that would require major treatment to meet water quality objectives. It is proposed that this stream be retained in a Zero Discharge System. This water would be pumped or drained to a leachate storage lagoon and disposal of through evaporation by using the water for dust control operations on the pit roads.

Lagoon systems are provided for as follows:

(1) North valley sedimentation and leachate storage lagoons to receive water from No. 1 Pit, Houth Meadows and Mine Services Area.

(2) Medicine Creek sedimentation and leachate storage lagoons.

(3) No. 2 Pit sedimentation and leachate storage lagoons to handle water from the Pit and Anderson Creek.

5.5 MINE SUPPORT FACILITIES

5.5.1 Introduction

The facilities required to support the operation of the Hat Creek mine include an administration building, a mine dry, a maintenance complex, service and supply facilities, emergency services, together with water and electrical supply systems.

All buildings and service facilities are grouped into a Mine Services Area, shown on Figure 5-7 which for No. 1 Pit is located northeast of the pit, covering an area of about 40 ha. For No. 2 Pit a smaller Mine Services Area, with some buildings provided on a reduced scale, and covering an area of about 30 ha has been planned. Construction of these facilities would commence in Years -3 for No. 1 Pit and Year 15 for No. 2 Pit. The locations of these areas are shown on the Project Layout Map, Figure 2-1. Water requirements for the mining operation are not expected to be large, the principal requirements being for potable water, fire protection, irrigation and dust control. The sources of water are noted in Section 5.5.3.

The electric power supply system for both No. 1 and 2 Pits has been planned to take power from the 60kV busbar at the proposed powerplant. This power supply would be distributed throughout the mine area by a ring main system.

Construction power would be supplied from an existing 60 kV line near Highway 12 north of the mine area.

5.5.2 Mine Buildings

The major mine buildings which are grouped in the Mine Services Area are:

Administration Building - a two storey facility to house staff in administration, accounting, personnel, purchasing, engineering, etc.

Mine Dry - the main point of reporting and assignment for mine operations personnel. The facility will be equipped with lockers, showers and laundry equipment to provide for the estimated labour force.

Maintenance Complex - a large integrated complex designed to handle all the routine and emergency maintenance requirements anticipated in a large mining operation utilizing a wide variety of equipment ranging from large bucketwheel excavators, hydraulic shovels, large diesel - electric trucks and materials handling equipment to small vehicles and service equipment.

The complex comprises truck and tractor repair and service bays, steam cleaning bays, welding and fabrication shops, machine and hydraulic repair shops, electrical and radio repair shops. Also included are warehouses and tool crib, supervisors, planning and maintenance record offices, a training centre, and a lunchroom.

The first aid centre, ambulances and fire trucks are also located in the complex. A layout of the complex for No. 1 Pit is shown in Figure 5-8. A smaller complex has been included for No. 2 Pit to service the needs of that operation, however, the complex at No. 1 Pit would still be operational, catering to large repairs, rebuilds, etc.

Mine Services Building - includes shops for carpenters, painters, pipefitters, plumbers, sheetmetal workers as well as vehicle and material storage areas.

Rubber Repair Shop - includes facilities for repairs to tires, conveyor belt and trailing cables.

Laboratories - includes environmental and assay laboratories equipped to analyze coal, oil and water samples.

Fuel and Lube Storage - separate facilities are provided for the bulk storage of diesel fuel, gasoline, waste oil and antifreeze; and of lube oils and greases. Integrated day storage facilities are provided at in-pit locations.

Other facilities and buildings are also provided; these include field maintenance centres, and mobile workshops for major repairs to bucketwheel excavators and stackers. Equipment erection and material storage areas are also provided. Maintenance facilities for the screening, crushing, blending and reclaim equipment are included in the screening and crushing building.

The buildings will be complete with the necessary power, water, heating, ventilating, air conditioning, fire protection, and sewage disposal services.

5.5.3 Water Supply

The total water requirements of the mine operations are not large as there are no major processing facilities involved. Potable water, fire protection, irrigation and dust control would be the main requirements. Estimated quantities, pro-rated from earlier studies, and their possible sources are:

Facility	Daily Requirements (m^3)	Source
Mine Service Area	500	Powerplant
Nursery	500	Pit Rim Reservoir
Roads & Stockpiles (dust control)	3000	Leachate Storage Lagoons

During construction, water would be supplied from groundwater wells, however, when the powerplant water supply system from the Thompson River is commissioned potable and fire protection water would be supplied from that source.

5.5.4 Power Supply

The power supply for the mine area will be taken from the 60 kV busbars at the powerplant and distributed via a ring main system of overhead transmission lines to the various locations within the mine area.

Distribution to the equipment and service areas will be at voltages ranging from 60 kV to 120 kV. A single line schematic diagram of the distribution network is shown on Figure 5-9.

A summary of estimated loads for both Pits No. 1 and No. 2 is given in Table 5-3.

Due to the large connected loads of excavating and conveying equipment further studies are required to ensure provision of a reliable power distribution system.

TABLE 5-1

HAT CREEK COAL LIQUEFACTION PROJECT
PREFEASIBILITY MINING STUDY

PRODUCTION SCHEDULE

QUANTITIES IN (10^6)

DESCRIPTION	PRE-PRODUCTION PERIOD -6	-5	-4	-3	-2	-1	PRODUCTION PERIOD 1	2	3	4	5	6	7	8	9	10	11	12
SHOVEL/TRUCK COAL - PIT No 1 (TONNES)									4.5	10.7	11.4	11.4	9.2	5.4	5.4	5.4	5.6	7.4
BWE COAL - PIT No 1 (TONNES)							4.8	15.7	20.5	18.6	18.6	18.6	20.8	24.6	24.6	24.6	24.4	22.6
PIT No 2 (TONNES)																		
SHOVEL/TRUCK WASTE - PIT No 1 (BM³)						8.8	8.9	8.9	9.2	5.1	4.8	4.4	6.1	8.6	8.6	8.6	10.4	11.6
BWE WASTE - PIT No 1 (BM³)						18.0	37.8	40.5	46.5	47.5	47.6	47.6	42.1	43.6	43.6	43.6	43.7	42.3
- PIT No 2 (BM³)																		
COAL MINED - PIT No 1 (TONNES)							4.8	15.7	24.8	29.3	30.0	30.0	30.0	30.0	30.0	30.0	30.0	30.0
COAL MINED - PIT No 2 (TONNES)																		
CUMULATIVE COAL (TONNES)							4.8	20.5	45.3	74.6	104.6	134.6	164.6	194.6	224.6	254.6	284.6	314.6
WASTE MINED - PIT No 1 (BM³)						26.8	41.7	49.4	55.7	52.7	52.4	52.0	48.2	52.2	52.2	52.2	54.1	53.9
WASTE MINED - PIT No 2 (BM³)																		
CUMULATIVE WASTE (BM³)						26.8	68.5	117.9	173.6	226.3	278.7	330.7	378.9	431.1	483.3	535.5	589.6	643.5

DESCRIPTION	13	14	15	16	17	18	19	20	21	22	23	24
SHOVEL/TRUCK COAL - PIT No 1 (TONNES)	7.8	11.9	20.1	14.4	14.4	14.4	20.3	20.9	18.5			
BWE COAL - PIT No 1 (TONNES)	22.2	18.1	9.9	15.6	15.6	15.6	9.7	2.5	0.1			
PIT No 2 (TONNES)								6.6	11.4	30.0	30.0	30.0
SHOVEL/TRUCK WASTE - PIT No 1 (BM³)	11.3	10.6	5.1	7.7	7.8	7.8	4.5	4.6	4.4			
BWE WASTE - PIT No 1 (BM³)	45.2	39.5	46.9	37.6	37.6	37.6	31.9	23.1	5.2			
- PIT No 2 (BM³)				5.3	42.4	53.0	44.8	60.6	69.4	80.5	76.0	82.0
COAL MINED - PIT No 1 (TONNES)	30.0	30.0	30.0	30.0	30.0	30.0	30.0	23.4	18.6			
COAL MINED - PIT No 2 (TONNES)								6.6	11.4	30.0	30.0	30.0
CUMULATIVE COAL (TONNES)	344.6	374.6	404.6	434.6	464.6	494.6	524.6	554.6	584.6	614.6	644.6	674.6
WASTE MINED - PIT No 1 (BM³)	56.5	50.1	52.0	45.3	45.4	45.4	36.4	27.7	9.6			
WASTE MINED - PIT No 2 (BM³)				5.3	42.4	53.0	44.8	60.6	69.4	80.5	76.0	82.0
CUMULATIVE WASTE (BM³)	700.0	750.1	802.1	858.7	940.5	1038.9	1120.1	1208.4	1287.4	1367.9	1443.9	1525.9

DESCRIPTION	25	26	27	28	29	30	31	32	33	34	35	TOTAL
SHOVEL/TRUCK COAL - PIT No 1 (TONNES)												219.1
BWE COAL - PIT No 1 (TONNES)												347.5
PIT No 2 (TONNES)	30.0	28.0	28.0	28.0	28.0	28.0	28.0	8.6	8.6	8.6	8.6	340.4
SHOVEL/TRUCK WASTE - PIT No 1 (BM³)												167.8
BWE WASTE - PIT No 1 (BM³)												844.0
- PIT No 2 (BM³)	79.3	81.1	71.8	57.0	48.2	44.0	44.0	44.5	27.5	9.2	0.7	1021.5
COAL MINED - PIT No 1 (TONNES)												566.6
COAL MINED - PIT No 2 (TONNES)	30.0	28.0	28.0	28.0	28.0	28.0	28.0	8.6	8.6	8.6	8.6	340.4
CUMULATIVE COAL (TONNES)	704.6	732.6	760.6	788.6	816.6	844.6	872.6	881.2	889.8	898.4	907.0	907.0
WASTE MINED - PIT No 1 (BM³)												1011.9
WASTE MINED - PIT No 2 (BM³)	79.3	81.1	71.8	57.0	48.2	44.0	44.0	44.5	27.5	9.2	0.7	1021.5
CUMULATIVE WASTE (BM³)	1605.2	1686.3	1758.1	1815.1	1863.3	1907.3	1951.5	1995.8	2023.3	2032.5	2033.2	2033.2

TABLE 5-2

SUMMARY OF MINING EQUIPMENT-PEAK REQUIREMENTS

Coal Liquefaction Project - Prefeasibility Study 1980 - Mining

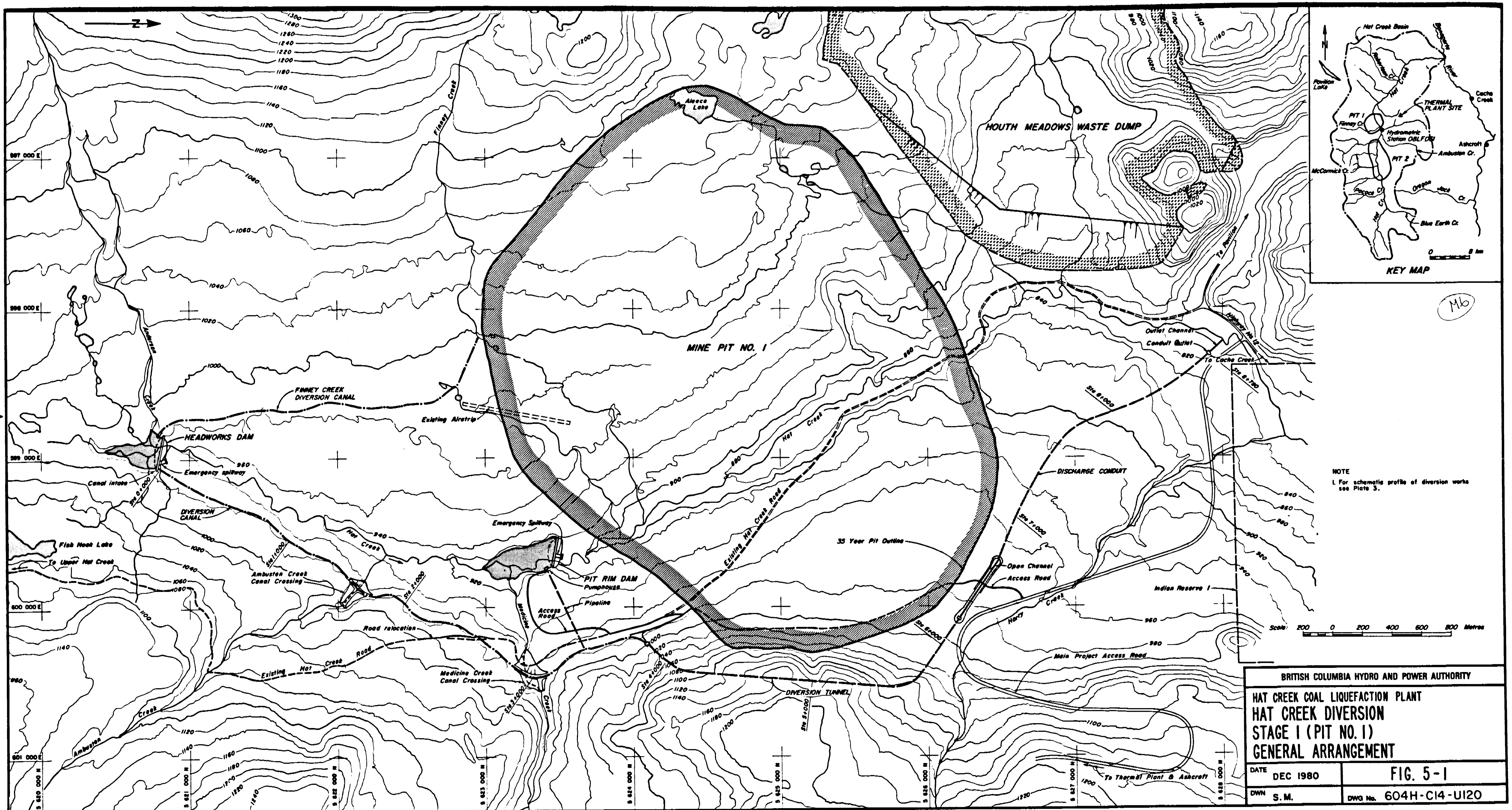
Item	Number Pit No. 1	Number Pit No. 2
Bucketwheel excavator- 60,000 m^3	5	5
100,000 m^3	-	2
Hydraulic Shovels 14.5 m^3 bucket capacity	5	2
Trucks 154-tonne	9	5
91-tonne	25	-
32-tonne	4	2
Scrapers 24 LCM	6	2
Graders	6	5
Dozers track	19	20
wheeled	3	3
Front-end loaders 11.5 m^3	-	2
9.6 m^3	2	2
5.4 m^3	2	2
Drills - Auger, Rotary, Rotary Percussion	2	2
Blasting Truck	2	2
Compactors	2	2
Gradall	1	1
Backhoe 1 m^3	1	2
Water wagon	3	1
Mobile crusher	1	1
Mobile cranes 5 to 70-tonne	9	10
Mobile service vehicles	20	20
Emergency vehicles	4	4
Light vehicles	100	100
Truck unloading stations	4	1
Crawler mounted waste spreaders	3	5
Tripper cars	3	5
Rail mounted stackers	2	2
Bridge type bucketwheel reclaimers	2	2
	Length	
Mine conveyors (to distribution station)	23000 m	32000 m
Coal transfer conveyors in preparation area	16015 m	16015 m
Overland coal conveyors to preparation plant	9600 m	15400 m
Waste conveyors	20270 m	36270 m

TABLE 5-3

ELECTRICAL LOADS

HAT CREEK COAL LIQUEFACTION PROJECT-PREFEASIBILITY STUDY 1980-MINING

Description	Connected Load (MVA)	
	Pit No. 1	Pit No. 2
Buildings & Services	7.5	9.0
Equipment		
Shovels	9.0	2.0
Incline Conveyors	11.5	1.0
BWE	19.0	32.4
Stackers & Trippers	11.4	19.0
Coal Conveyors (BWE System)	20.1	32.4
Waste Conveyors (BWE System)	79.4	64.6
Overland Coal Conveyor	31.0	31.0
Crushing & Screening Plant	4.5	4.5
Blending/Reclaim/Delivery (Powerplant)	3.7	3.7
Blending/Reclaim/Delivery (C.L. Plant)	8.4	8.4
	205.5	208.0



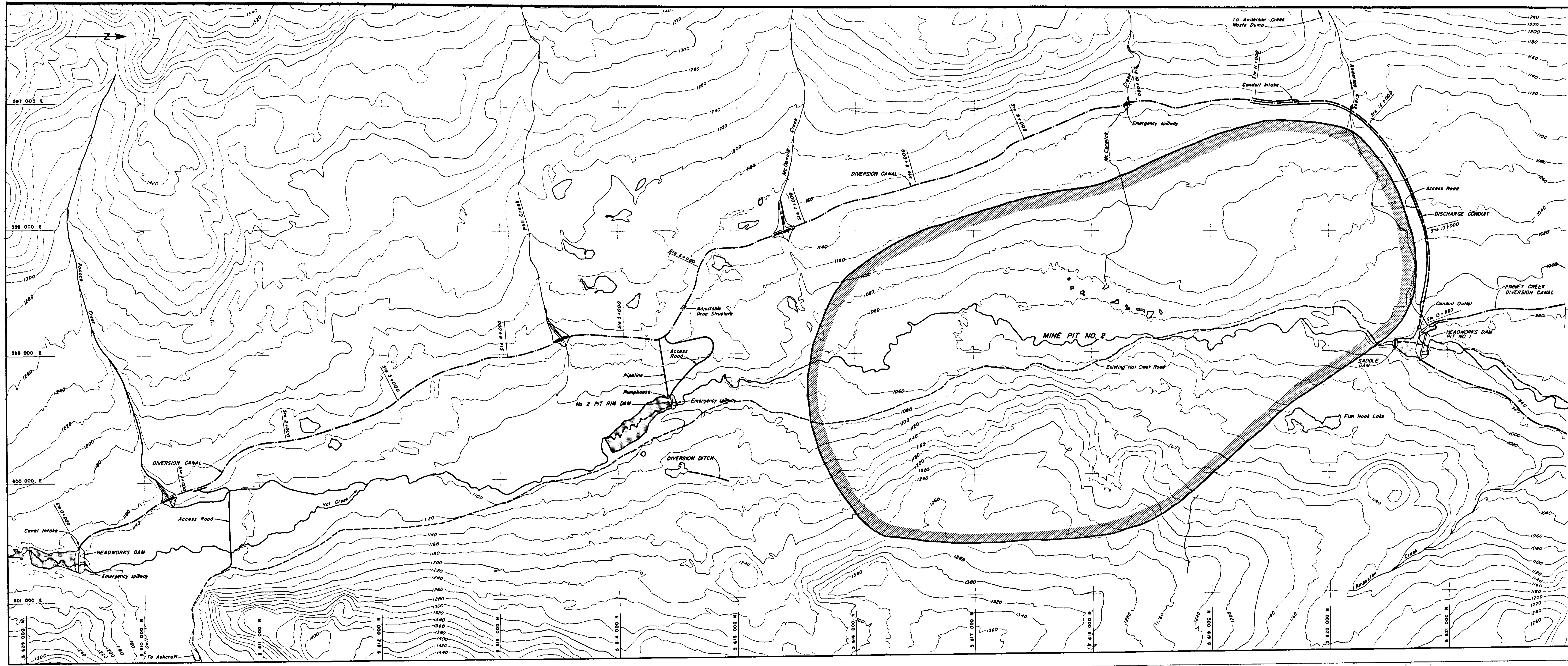
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

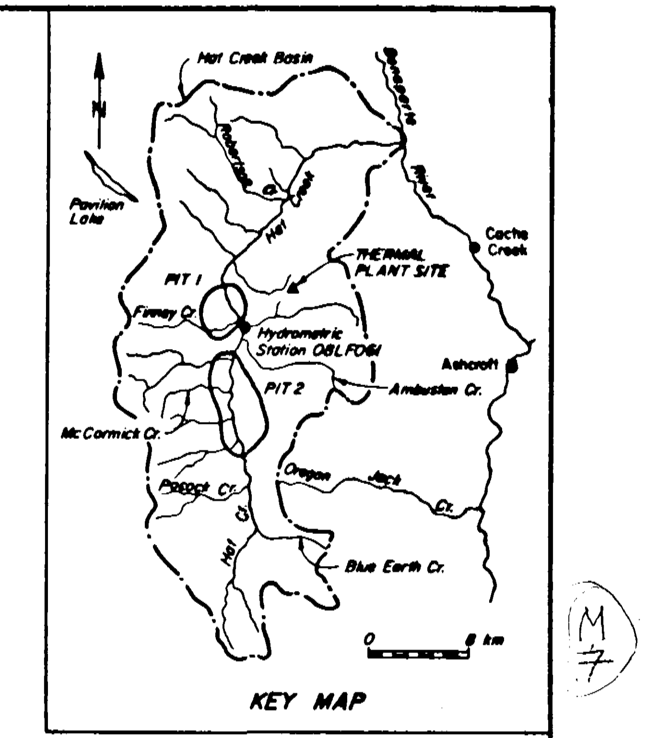
HAT CREEK COAL LIQUEFACTION PLANT

HAT CREEK DIVERSION
STAGE I (PIT NO. I)
GENERAL ARRANGEMENT

DATE DEC 1980 | FIG. 5-1

DWN S.M. | DWG No. 604H-C14-U120





KEY MAP

NOTE
1. For schematic profile of diversion works see Plate 3.

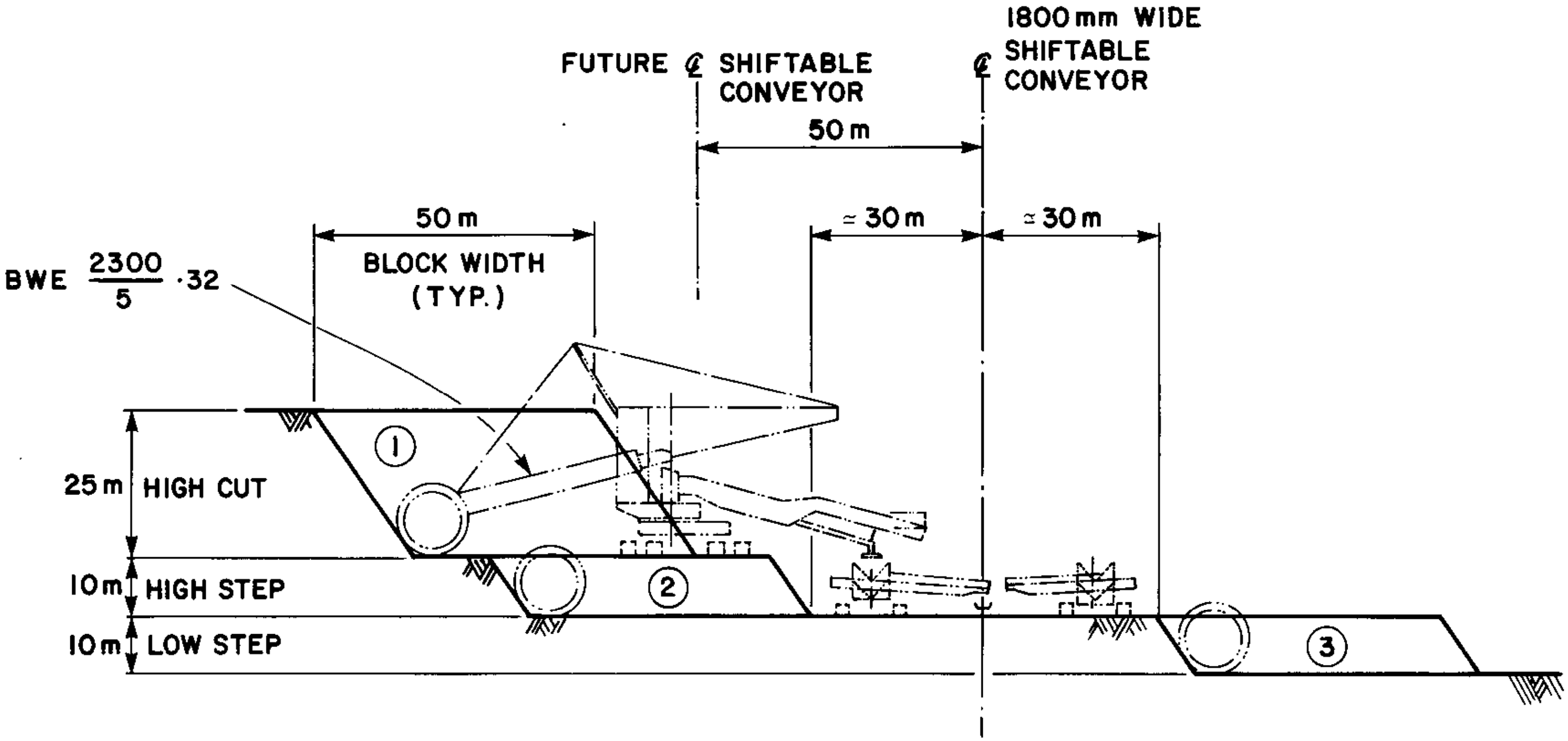
Scale: 200 0 200 400 600 800 Metres

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK COAL LIQUEFACTION PLANT
HAT CREEK DIVERSION
STAGE 2 (PIT NO. 2)
GENERAL ARRANGEMENT

DATE	DEC 1980		FIG. 5-2
DWN	S.M.	DWG No.	604H-C14-X121

REPORT No. H 1322



(1) BWE EXCAVATES HIGH CUT THEN RAMPS DOWN TO EXCAVATE HIGH STEP

(2) BWE EXCAVATES HIGH STEP THEN MOVES TO OTHER SIDE OF CONVEYOR AND RAMPS DOWN TO EXCAVATE LOW STEP

(3) BWE EXCAVATES LOW STEP THEN TRAVELS BACK VIA RAMPS TO BEGIN CYCLE AGAIN AFTER CONVEYOR IS SHIFTED TO NEW POSITION

FIGURE 5 - 3

SEQUENCE OF BWE OPERATIONS
No. 1 DEPOSIT

FIGURE 5-4
Pit Development Sequence Pit No. 1

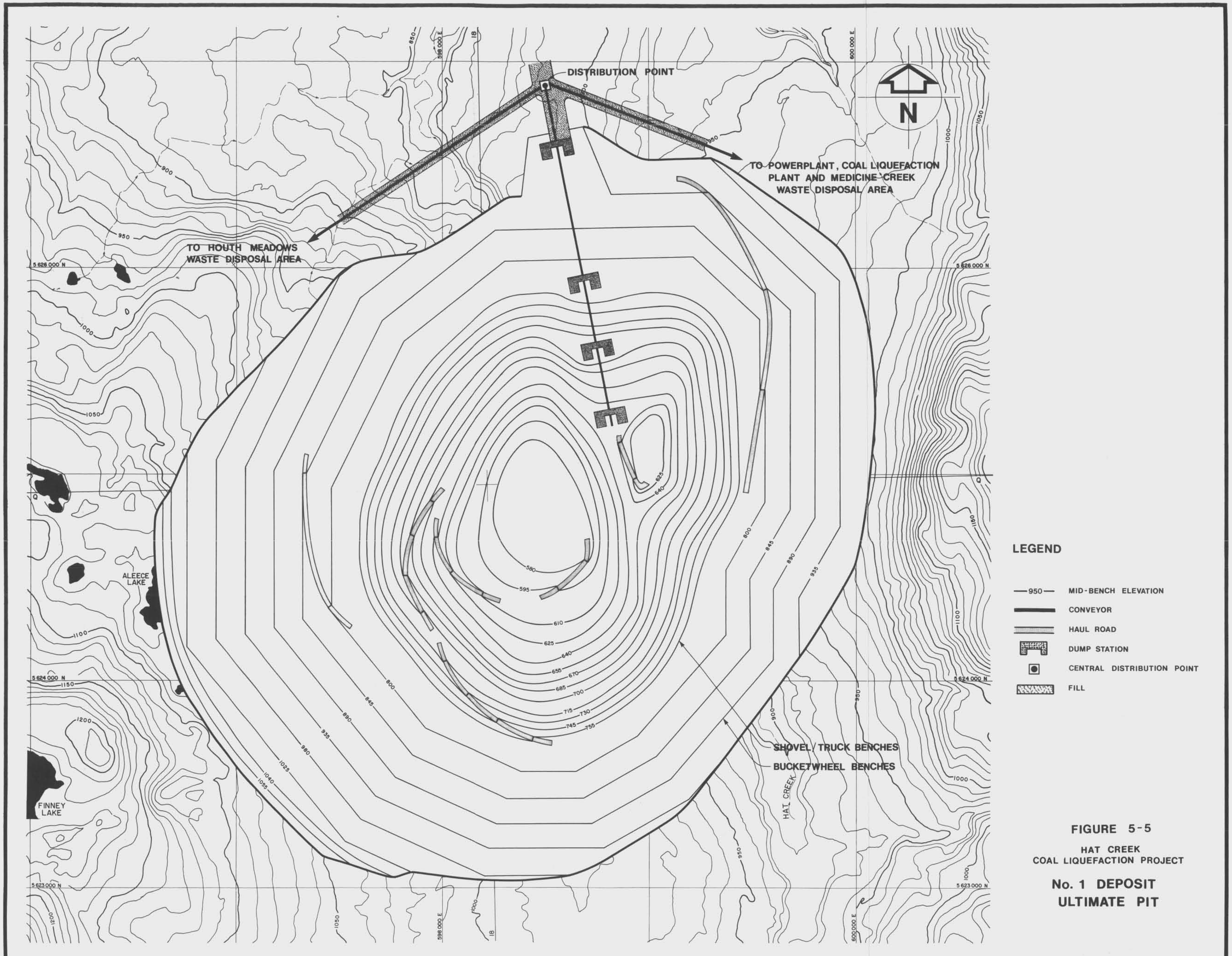
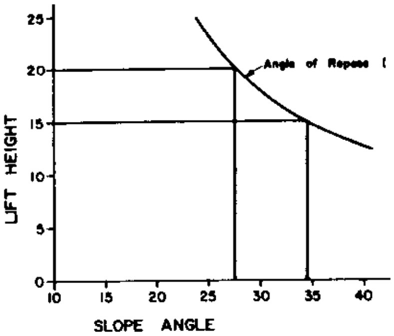


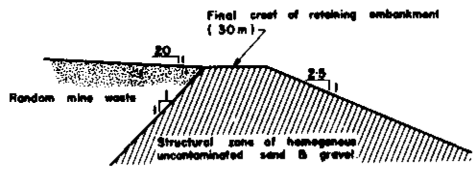
FIGURE 5-5

HAT CREEK
COAL LIQUEFACTION PROJECT

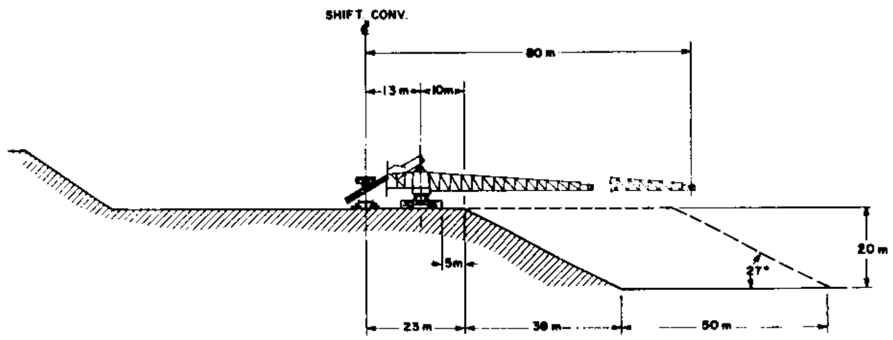
No. 1 DEPOSIT
ULTIMATE PIT



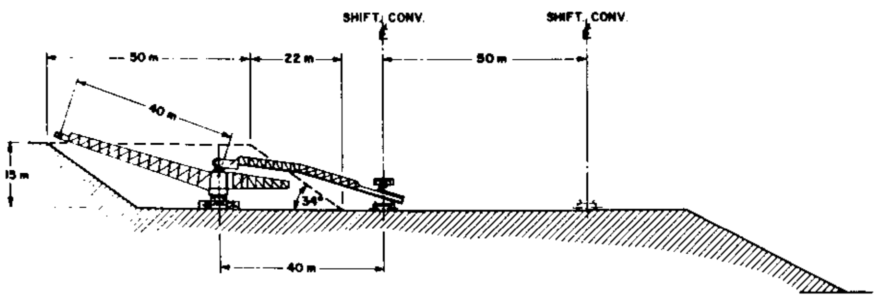
(a) WASTE MATERIAL ANGLE OF REPOSE



WASTE EMBANKMENT SLOPE ANGLES



(b) SPREADER POSITIONED TO DEPOSIT WASTE BELOW THE SHIFTABLE CONVEYOR



(c) SPREADER POSITIONED TO DEPOSIT WASTE ABOVE THE SHIFTABLE CONVEYOR

FIGURE 5-6

Waste Dump Construction

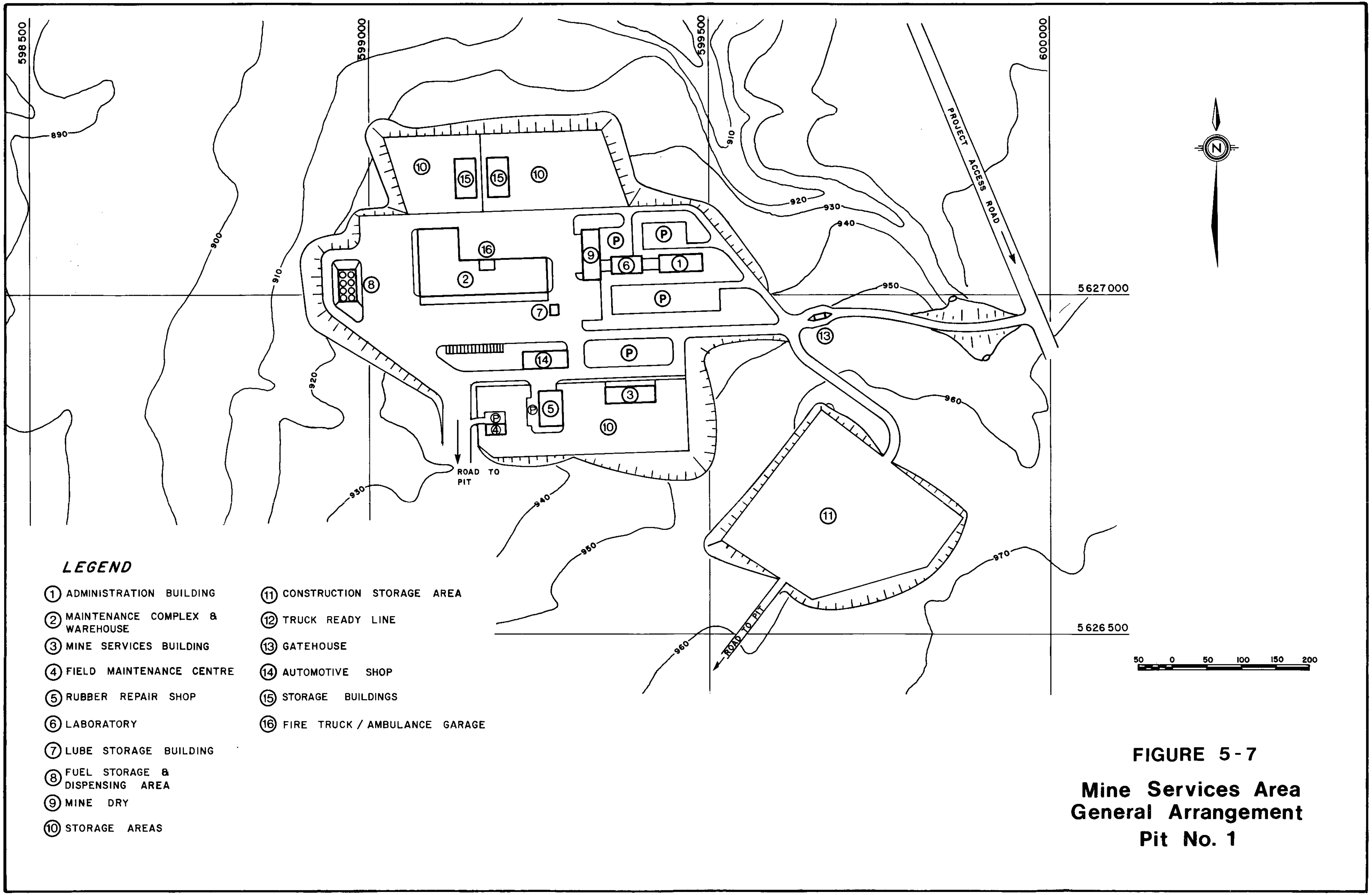


FIGURE 5-7

Mine Services Area General Arrangement Pit No. 1

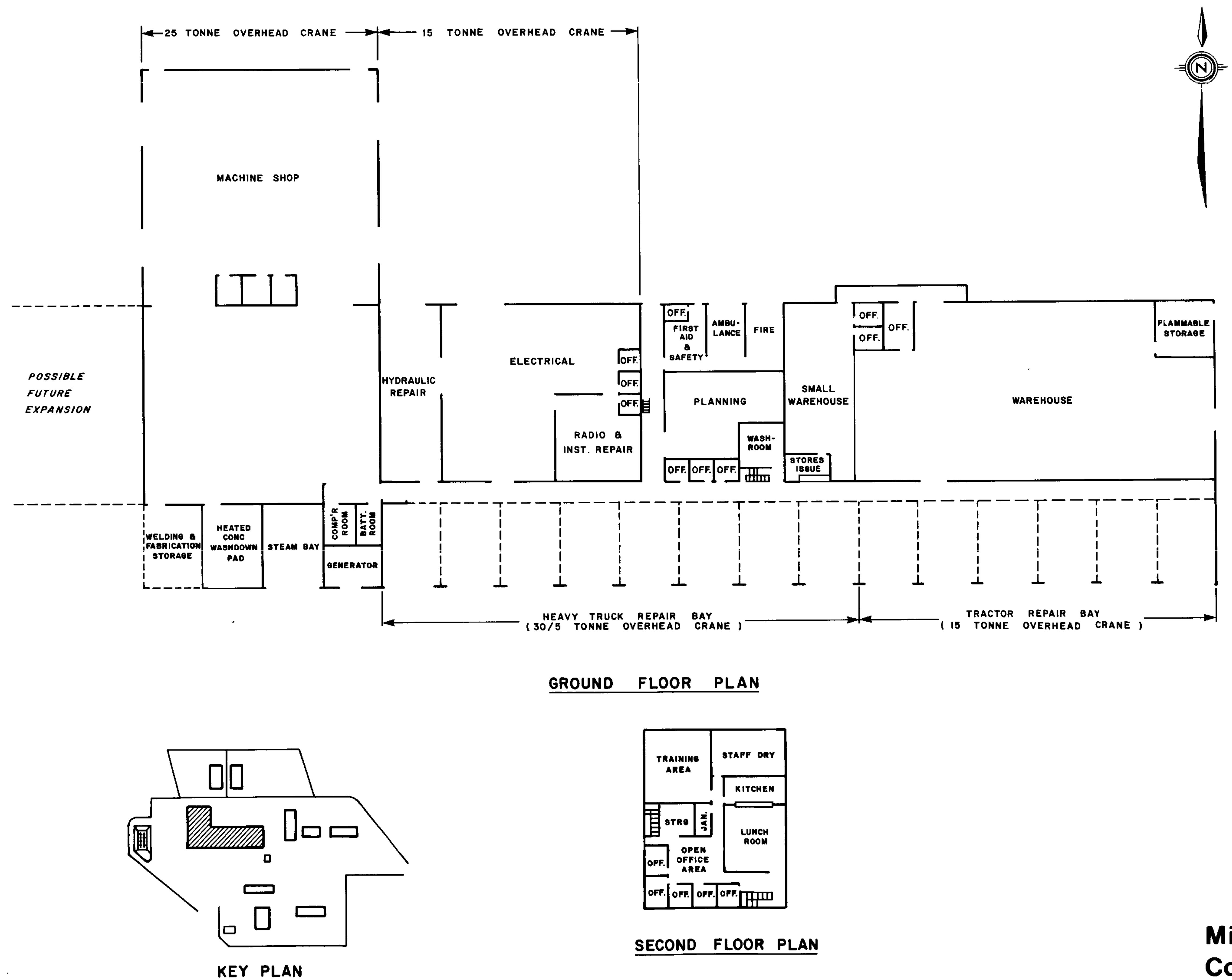


FIGURE 5-8

Mine Maintenance Complex Pit No. 1

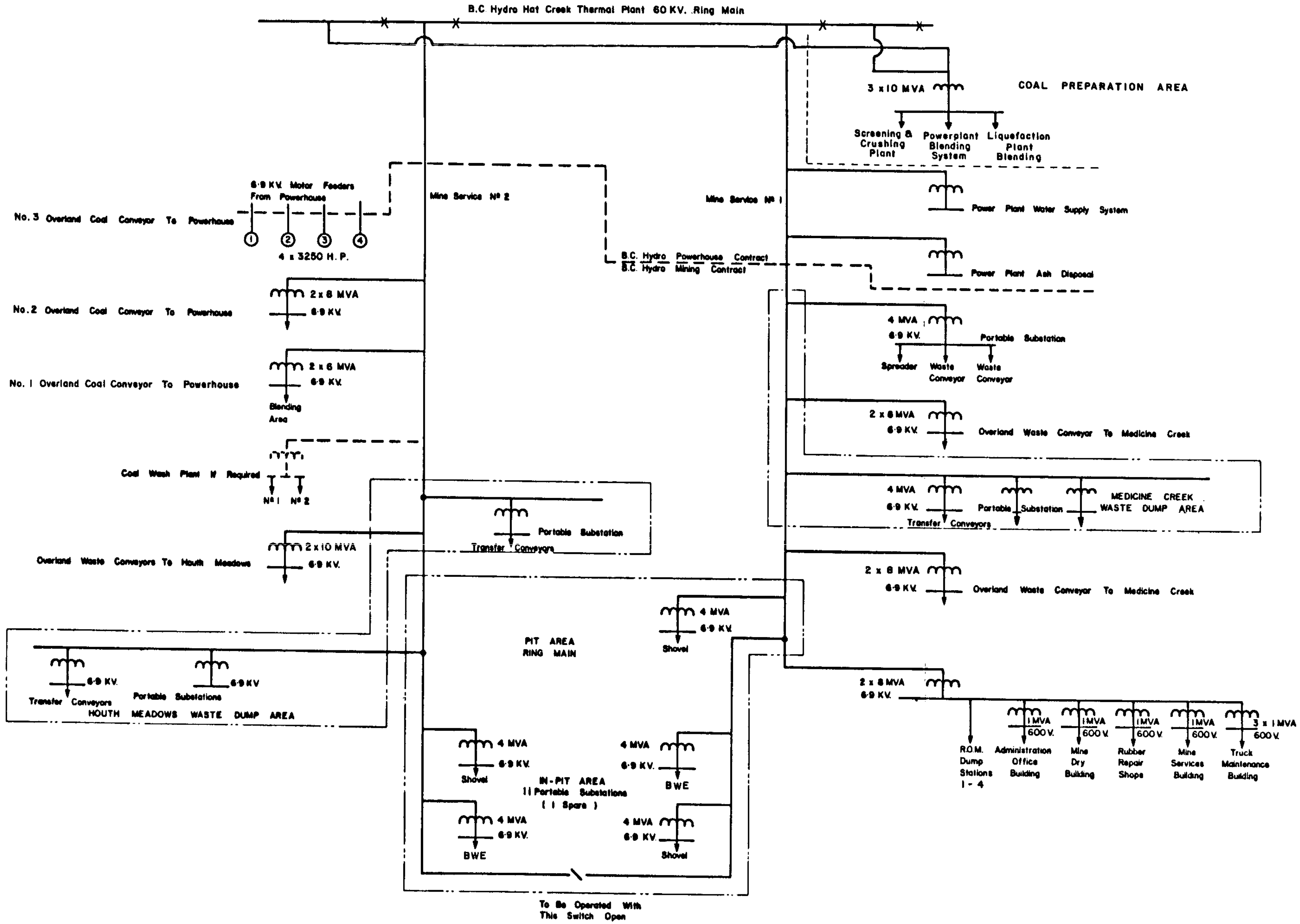


FIGURE 5-9

MINE POWER DISTRIBUTION NETWORK

SECTION 6

MATERIALS HANDLING

6.1 INTRODUCTION

The function of the materials handling system is to transport coal and waste from the area of excavation, perform any necessary processing and deliver the products to their appropriate locations: waste to the waste dumps; coal to the powerplant and the coal liquefaction plant.

This project presents a materials handling challenge of considerable magnitude. Over the life of the project approximately 2.6 billion m^3 (5 billion tonnes) of coal and waste must be transported over distances averaging 6-8 km. During the first twenty years of the project while No. 1 Deposit is being mined, the typical annual quantity to be moved is 140 million tonnes using over 100 individual conveyors totalling over 65 km in length. When No. 2 Deposit comes on stream in the later years of the project, annual quantities will exceed 200 million tonnes.

The conceptual design for the materials handling system presented in this report was planned to ensure a reliable supply of specification coal to each of the three plants: powerplant, liquefaction plant gasifiers and steamplant. This is largely achieved through the provision of redundant and backup systems and the use of alternative coal streams. For example, gasifier coal could be delivered to the streamplant in an emergency.

Waste disposal systems are discussed in Section 5.3.4.

6.2 IN-PIT CONVEYING

The in-pit conveying systems are shared between coal and waste with the separation and control of the materials being effected by planning and scheduling of the excavation, the operation of a conveyor distribution station at the pit head and close supervision.

Excavation of materials in Deposit No. 1 is performed by five bucketwheel excavators (BWE) and four hydraulic shovels. Each of these types of equipment can mine in coal, waste or both in a given shift. Typically two or three of each will be operating.

Each BWE is supplied with its own conveyor train running from the excavator to the conveyor distribution station. Because of the cutting action of the BWE it is usually unnecessary to provide a grizzly to restrict the size of material flowing onto the conveyor. The selected BWE's have a rated output of 3000 m^3/h but under favourable digging conditions can achieve short term peaks more than double this. The conveying system is sized to handle 5000 m^3/h or 7500 t/h in coal.

Trucks loaded by the hydraulic shovels haul coal and waste to one of two conveyors installed on the central incline. Material is dumped from the trucks through a grizzly with 400 mm square openings into a 300 t capacity surge hopper prior to being loaded onto the conveyor using pan feeders. Four of these truck dumping stations are provided at different elevations as the mine is developed to full depth. Oversize material on the grizzlies can either be broken by a boom mounted hydraulic hammer or removed with a clamshell and hauled away by truck. Figure 6-1 shows the layout of a dumping station.

Seven conveyors: five from the BWE's and two fed by trucks, supply material to the conveyor distribution station. Material is carried away from the station by three waste and two overland coal conveyors. Each of the seven incoming conveyors can discharge material onto any of the five outgoing conveyors. The re-direction of the material flow from one conveyor to another can be accomplished in under a minute, while the flow is cut-off at the loading point for that length of time to permit the change to be made. Radio communication between the excavator/conveyor loading operator and the conveyor controller is essential to coordinate this operation.

Conveyor belt widths have been standardized as far as practical to enhance the interchangeability of parts. Further work will be required in this final design stage to standardize motores and drives to minimize inventories of spare components. Two sizes of conveyor have been selected to work in No. 1 Pit: 1800 mm wide belt transporting materials from the BWE's to the pit head and 1400 mm wide belt to carry the shovel-truck system materials. Table 6-1 provides a list of the conveyors proposed.

In-pit conveyors will not be covered but will be equipped with such standard safety features as emergency trip cords, sequential starting and stopping interlocks, underspeed and side travel switches and overload protection. On inclined conveyors special safety features required are: skirting boards to restrain lumps and holdbacks to prevent a loaded conveyor from running backwards when not under power.

6.3 COAL HANDLING

The coal handling system, as presented in Figure 6-2, is designed to transport run-of-mine coal from the pit head and prepare and deliver specification products to the powerplant, the gasifiers and the steamplant in the liquefaction plant. The coal supply must be dependable because of the economic penalty of a plant shutdown. There are two streams of coal produced in the mine: the A, B & C zone stream and the D-zone (higher quality, lower sulphur) stream. These streams must be handled separately to produce the specified products. The two streams are split by dry screening into size fractions with each fraction being routed to a particular product stream.

In developing the conceptual design of the coal handling system the following were the principal criteria to be met:

(1) The powerplant performance fuel specification, previously established, should be maintained as well as the ability to supply higher quality, lower sulphur coal to permit operation of the proposed Meterological Control System (M.C.S.) when required.

(2) Gasifier coal to be sized to -100 mm +13 mm and contain less than 2% of -2 mm fines. To minimize the production of fines requires that storage and handling of this product must be minimized after sizing.

(3) The powerplant and steamplant coals to be blended to minimize short term quality fluctuations and sized to -50 mm.

(4) Reliability of supply.

6.3.1 Conveying

Coal is transported from the conveyor distribution station at the pit head by overland conveyor to the Crushing and Screening Plant adjacent to the Powerplant complex. Two parallel overland conveyor lines which can be loaded from each of the seven conveyors entering the distribution station are provided. Each of the overland conveyor lines consists of three flights totalling 4750 m in length and a vertical lift of 535 m. Each of these systems is rated to carry 7500 t/h to match peak BWE capacity. Each of these conveyors is

2000 mm wide to reduce belt tension and help improve reliability. The overland conveyors, which delivery coal to the Crushing and Screening Plant is covered to reduce dust emissions.

6.3.2 Coal Crushing and Screening

The Crushing and Screening Plant consists of eight lines of feeders, crushers, screens and conveyors each rated for a throughput of 1000 t/h. The plant is divided into two sections of four lines with each section nominally dedicated to the A, B & C-zone or the D-zone coal stream, but with sufficient flexibility retained to permit interchanging the streams. A coal flow diagram is presented in Figure 6-3.

The two overland conveyors deliver the coal to the centre of the top floor of the plant and discharge onto either of two shuttle conveyors which each feed a bank of 4 x 500 t bins. Coal is withdrawn from each bin by a pan feeder and conveyed to a scalping screen which directs the +100 mm fraction through a crusher. The crushed product is combined with the -100 mm fraction and conveyed to be screened on multi-deck screens to effect the final separation of the coal into -13 mm and +13 mm components. The screen products are discharged onto the appropriate collector conveyor to transport the material out of the plant to its planned destination. Duplicate collector conveyors are provided to ensure reliability. A limited surge pile of 10,000 t capacity is provided for each half of the plant to smooth out short term imbalance between the Mine and Plant.

Screening Plant Product	Gasifiers Direct Feed	Powerplant Blending Stockpile	Steamplant Blending Stockpile
ABC +13 mm	X	-	-
ABC -13 mm	-	X	X
D +13 mm	X	X	-
D -13 mm	-	X	X

Summary of screening plant products and utilization.

The conceptual design presented here has been developed in sufficient detail to establish the feasibility of the approach and a reasonable basis for estimating capital and operating costs. Final design optimization can be performed at a later date, when sufficient data on the screening characteristics of Hat Creek coal has been accumulated from field testing.

The scale of the proposed crushing and screening plant is dictated by the lack of significant surge capacity between the plant and the gasifiers. With the proposed layout it should be practical to maintain adequate feed to the gasifiers with one half of the plant unavailable and three of four lines operating. The plant has the capability to: handle short term peaks in mine output; replenish stockpiles while meeting normal output requirements; and maintain throughput when wet weather creates difficult screening conditions.

6.3.3 Blending, Storage and Delivery

The blending area provides for four blending stockpile bays with each providing for a pile 550 m long 50 m wide and 20 m high. Under normal operation two of these bays would be dedicated to blending Powerplant coal in 300,000 t piles which provide for one week of operation at maximum capacity rating. The other two bays will be used for blending steamplant coal in 80,000 t piles with the remainder of each bay occupied by 200,000 t compacted dead storage piles of blended Powerplant fuel.

Two slewing, luffing, rail-mounted stackers and two bridge-type, rail-mounted bucketwheel reclaimers are planned. Each system has the capacity to supply the combined requirements of the Powerplant and the Steamplant.

Each stacker will work two bays of the blending area from the centre conveyors and build the blending piles using the windrow method. While one pile is being constructed the other pile is reclaimed. The dead storage piles are built or replenished by stacker to meet Powerplant fuel specifications and compacted to prevent spontaneous combustion. All coal delivered by the Crushing and Screening Plant for Powerplant and Steamplant consumption is usually routed to the blending piles, although provision is made to deliver the coal directly to the Powerplant feed conveyors or flow through the blending area to the CLP Steamplant conveyors.

The bucketwheel reclaimers also work two blending bays: one reclaimer on Powerplant fuel and the other on Steamplant fuel and recovering coal from the compacted dead storage when required. Upon completion of work in one bay the reclaimer is moved on a transporter car to its next assignment. The two reclaimers can be interchanged using this method.

Coal reclaimed for Powerplant use is conveyed to the Powerplant feed conveyors and delivered into a 600 t surge bin. Steamplant coal is reclaimed and delivered to the CLP feed conveyors. When reclaiming from dead storage for delivery to the Powerplant the reclaim conveyors are reversed to deliver coal back to the Powerplant feed conveyors. Gasifier coal is delivered directly from the Crushing and Screening Plant to the CLP feed conveyors.

The CLP feed conveyor system consists of three parallel 1400 mm wide conveyors each rated at 3000 t/h. One conveyor is dedicated to gasifier coal, one to steamplant coal and the third is shared. Given the high degree of reliability of conveyor systems this should provide ample assurance of coal supply to the CLP.

TABLE 6-1

CONVEYOR LIST

Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

CONVEYOR NO.	WIDTH mm	LENGTH m	LIFT m	SPEED m/s	CAPACITY t/h	INSTALLED HP
In-Pit (Pit #1)						
T1-A	1400	330	33	4.5	5000	1000
B	1400	330	33	4.5	5000	1000
T2-A	1400	660	120	4.5	5000	3000
B	1400	660	120	4.5	5000	3000
T3-A	1400	330	60	4.5	5000	1500
B	1400	330	60	4.5	5000	1500
T4-A	1400	330	60	4.5	5000	1500
B	1400	330	60	4.5	5000	1500
Overland						
OC1-A	2000	1950	115	5	7500	5700
1-B	2000	1950	115	5	7500	5700
OC2-A	2000	1100	245	5	7500	8600
2-B	2000	1100	245	5	7500	8600
OC3-A	2000	1700	175	5	7500	6500
3-B	2000	1700	175	5	7500	6500
OC4-A	2000	65	-	5	7500	75
4-B	2000	65	-	5	7500	75
Screening Plant						
"A,B,C" Stream						
1 A	1400	35	4	2.5	1000	75
1 B	1400	35	4	2.5	1000	75
1 C	1400	35	4	2.5	1000	75
1 D	1400	35	4	2.5	1000	75
3 A	1400	125	25	2.5	1000	200
3 B	1400	125	25	2.5	1000	200
3 C	1400	125	25	2.5	1000	200
3 D	1400	125	25	2.5	1000	200

TABLE 6-1

CONVEYOR LIST

Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

CONVEYOR NO.	WIDTH	LENGTH m	LIFT m	SPEED m/s	CAPACITY t/h	INSTALLED HP
5 A	1800	65	4	2.5	1000	100
5 B	1800	65	4	2.5	1000	100
7 A	1800	35	4	2.5	3000	75
7 B	1800	35	4	2.5	3000	75
Bypass & Recl.						
9 A	1400	125	25	4.2	3000	400
9 B	1400	60	10	2.5	1000	75
"D" Stream						
2 A	1400	35	4	2.5	1000	75
2 B	1400	35	4	2.5	1000	75
2 C	1400	35	4	2.5	1000	75
2 D	1400	35	4	2.5	1000	75
4 A	1400	125	25	2.5	1000	200
4 B	1400	125	25	2.5	1000	200
4 C	1400	125	25	2.5	1000	200
4 D	1400	125	25	2.5	1000	200
6 A	1800	35	4	2.5	3000	75
6 B	1800	35	4	2.5	3000	75
Bypass & Recl.						
8 A	1400	125	15	4.2	3000	300
8 B	1400	60	10	2.5	1000	100
Powerplant Blend & Supply						
10 A	1400	350	4	4.2	3200	300
10 B	1400	350	4	4.2	3200	300
12	1400	670	10	4.5	3200	600
14 A	1400	670	6	4.2	3000	500
14 B	1400	670	6	4.2	3000	500
16	1400	135	4	4.2	3000	150
18	1400	70	4	4.2	3000	75
20	1400	210	4	4.2	3000	175

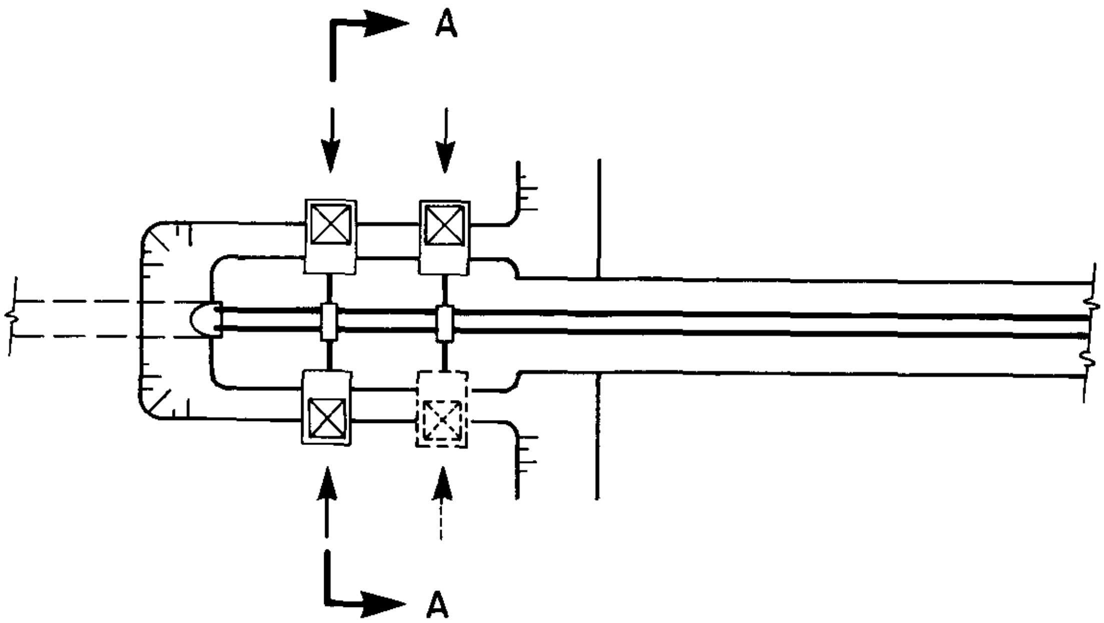
TABLE 6-1

CONVEYOR LIST

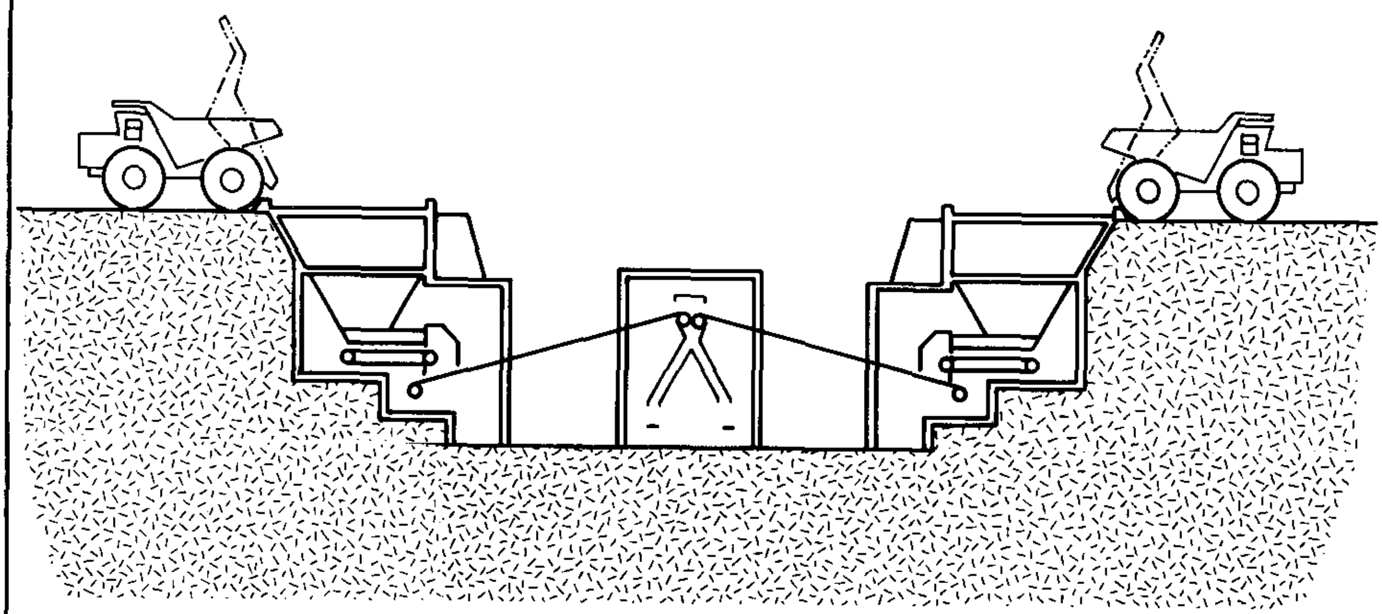
Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

CONVEYOR NO.	WIDTH mm	LENGTH m	LIFT m	SPEED m/s	CAPACITY t/h	INSTALLED HP
22 A	1400	180	35	4.2	3000	550
22 B	1400	180	35	4.2	3000	550
Gasifier & Steam Plant						
Blend & Supply						
11 A	1400	500	4	4.2	3000	350
11 B	1400	500	4	4.2	3000	350
13	1400	670	10	4.5	3200	600
15 A	1400	670	6	4.2	3000	500
15 B	1400	670	6	4.2	3000	500
17 A	1400	400	-	4.2	3000	250
17 B	1400	400	-	4.2	3000	250
19 A	1400	600	40	4.2	3000	900
19 B	1400	600	40	4.2	3000	900
21 A	1400	1750	50	4.2	3000	1500
21 B	1400	1750	50	4.2	3000	1500
21 C	1400	1000	40	4.2	3000	1200
23 A	1400	550	10	4.2	3000	500
23 B	1400	550	10	4.2	3000	500
BWE System:						
(Pit No. 1)						
In-pit Conveyors	1800	19260 (total)	-	5.2	6500	-
Waste Conveyors	2400	20270 (total)	-	5.2	13000	-
(Pit No. 2)						
In-Pit Conveyors	1800	4000*	-	5.2	6500	-
In-Pit & Waste Conveyors	2400	16000*	-	5.2	13000	-

*Additional to lengths for Pit No. 1



PLAN - TYPICAL TRUCK DUMP STATION



SECTION A-A

FIGURE 6-1
Inpit Truck Dump Station

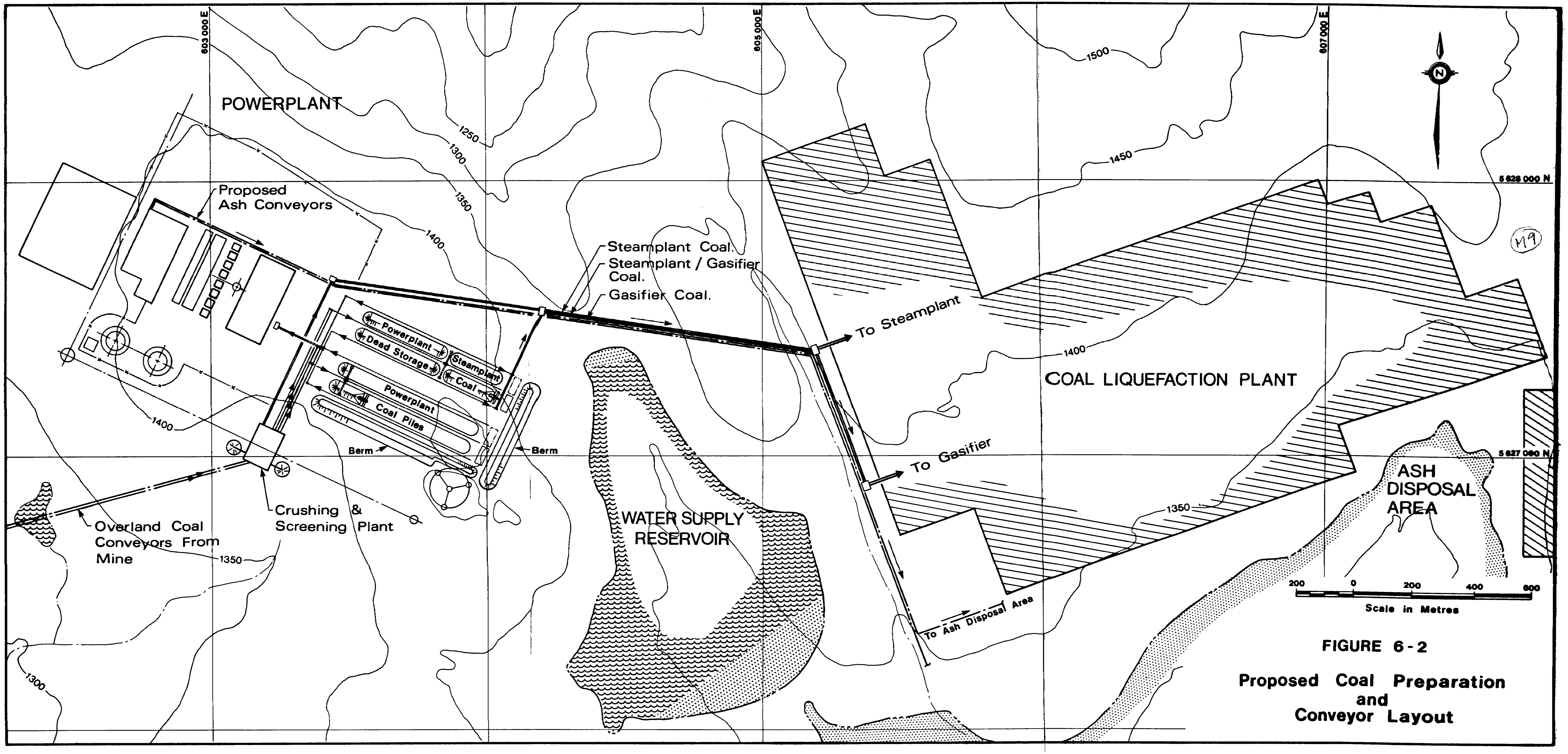


FIGURE 6-2

Proposed Coal Preparation and Conveyor Layout

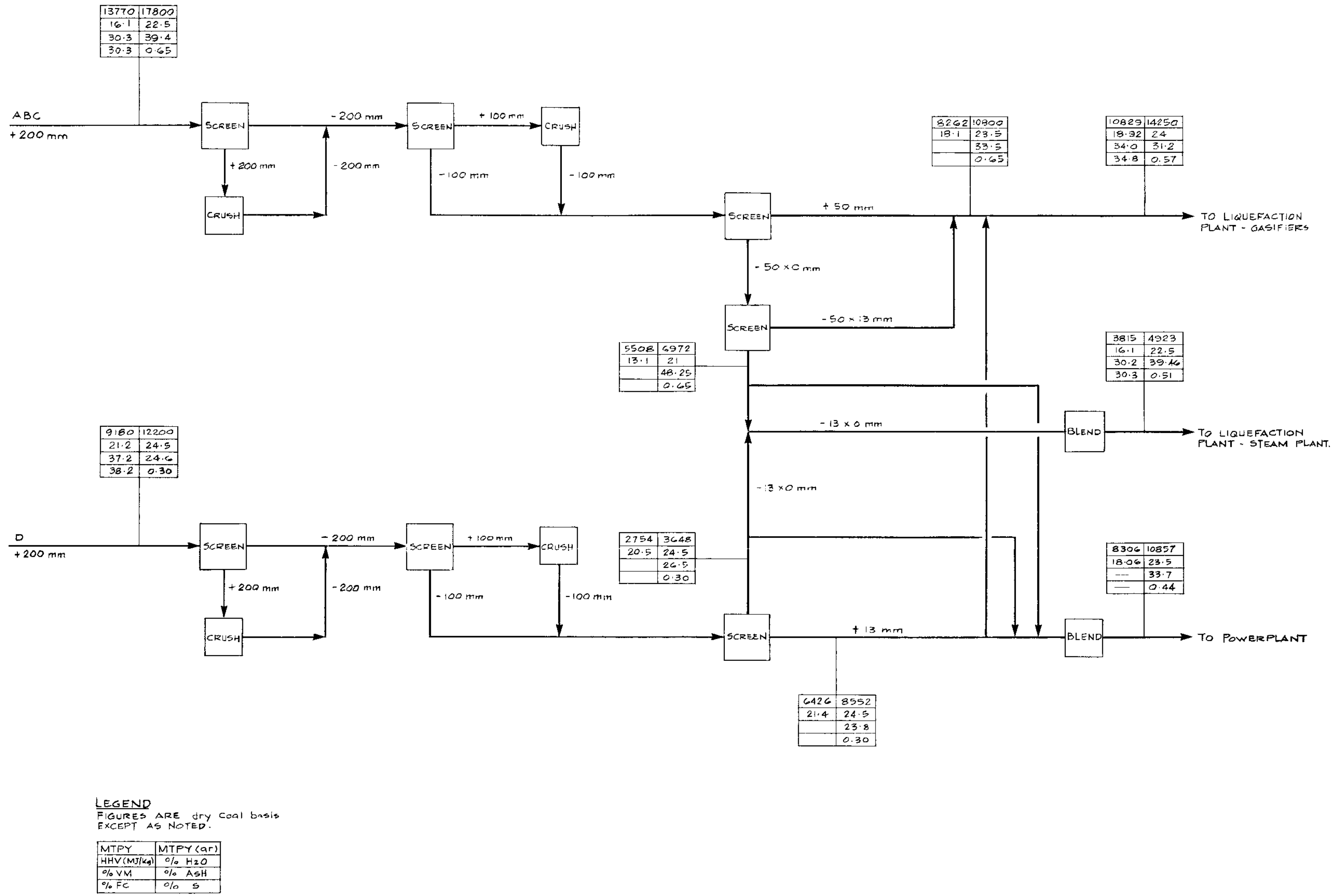


FIGURE 6-3

COAL FLOW DIAGRAM

SECTION 7

ENVIRONMENTAL PROTECTION & RECLAMATION

7.1 INTRODUCTION

The project area is situated with the Hat Creek drainage basin. Several small creeks, Medicine, Finney, Ambusten, Houth, Anderson, McCormick and McDonald drain into Hat Creek, which flows north and then east to the Bonaparte River, from where it joins the Thompson River System just north of Ashcroft. The water bodies of significance in the general project area are Aleece Lake, Finney Lake and Fishhook Lake.

The regional climate is classified as continental, and is typified by long, cold winters and short, warm summers. Semi-arid conditions prevail; average precipitation is 317 millimetres per annum, of which about half falls as snow. Winds behave according to the mountain/valley topography and are channelled predominantly upslope from the north to the south and southwest during the day, and the reverse at night.

The objective of the Reclamation and Environmental Protection Plan is to protect land, water, and air during the construction and operation of the mine. After the mine closes, it is planned, within practical limits, to restore the land to the same conditions as it was before mining started. While the mine is being built and operated, the control of drainage will be of paramount importance in order to protect the aquatic environment downstream. The same considerations apply to the control of noise and dust. It is equally important to ensure that any measures taken to replant disturbed land should be continued for however long it may take to restore the land to a self-sustaining, stable and useful condition.

The plan makes provision for both restoration and extended care under three major reclamation and environmental protection priorities:

(1) Drainage control during and after mining;

(2) The effective replanting of disturbed land areas; and

(3) The development of a safe pit abandonment scheme.

7.2 DUST

Previous studies have indicated that the B. C. Pollution Control Board Objectives for total suspended particulates of 60 $\mu g/m^3$ and 150 $\mu g/m^3$ for annual and 24-hour averages respectively could be met. In order to meet the objectives specific design and operating requirements were established, for example:

(1) Material Handling System:

- (a) Coal pile orientation.
- (b) Construction of windbreak berms.
- (c) Installation of water guns.
- (d) Windrow blending pile construction method using telescopic chutes.
- (e) Dust suppression equipment at transfer points.
- (f) Covering conveyors where practical, such as the overland conveyors from the mine and the CLP feed conveyors. Covering the stacking and reclaiming conveyors in the blending yard is not possible due to the operation of the equipment.

(2) The area stripped of surface soils will at all times be minimized to reduce erosion potential. In addition, overburden removal would be continued until non-friable (i.e. low dusting potential) material was reached if possible.

(3) Binding agents would be used to control erosion where appropriate.

(4) Areas that would remain exposed after overburden removal for extended periods of time would be revegetated.

7.3 NOISE

Existing sound levels have been measured and compared with those likely to arise from operation of the mine. Findings show that the Hat Creek Valley may be affected by noise from the project, though not significantly.

Present noise levels in the valley vary from about 30 to 40 decibels in the areas away from Highway 12. Adjacent to the highway, noise levels range from 44 to 41 decibels. By comparison, a soft whisper would produce a sound level of 30 decibels, and a quiet wind through the trees would be around 50 decibels.

Noise from construction would, of course, be transitory, whereas noise from the mine operation essentially constant throughout the mine's productive life. The latter would stem principally from heavy equipment moving in and around the pit, with intermittent additional noise from the coal stacker-reclaimer, conveyors and crushers. Only two of the five Hat Creek ranches are expected to be affected by construction activity noise. Maximum noise levels on these ranches would reach 47 decibels which is close to the 45 decibels typically set at a nighttime level by many communities.

The southwestern portion of the Bonaparte Indian Reserve may be affected by mining and coal preparation noise. The area involved contains at present one dwelling with four to six residents. The two ranches nearest to the pit might experience intermittent noise levels up to 63 decibels; the next two, levels of between 45 and 49 decibels; and the two furthest away, levels of 41 to 42 decibels. As the natural background level is 35 to 40 decibels, the occasional level of noise from the mining operation is not expected to cause annoyance to anyone reasonably disposed. It is planned to minimize blasting activity. When blasting is required it would be restricted to daylight hours.

7.4 MINE DRAINAGE AND WATER QUALITY

Drainage measures insofar as they affect reclamation may be summarized by noting that all lagoons, diversions, ditches, and reservoirs linked with wetland and riparian habitats will be left intact and revegetated wherever possible within the constraints imposed by mining. Drainage control structures will be grass-seeded, and, where erosion or flow capacity is not involved, with a mixture of shrubs, trees and grasses.

Laboratory and field tests on materials which would be encountered during mining have been run to determine the concentrations of leachable materials. Based on these data and the water quality and hydrology of the water bodies to be affected by this project, the main drainage plan has been devised. Essential elements of the plan are:

(1) All water suitable for simple diversion without any form of treatment, such as Hat Creek, would be redirected around the project and returned to its natural downstream watercourse;

(2) Runoff contaminated with suspended solid material would undergo sedimentation to reduce the concentration of suspended solids to less than 50 milligrams per litre;

(3) All water of unsuitable quality for discharge would be collected in leachate pond and disposed of on site by reuse in dust control or by spray evaporation on waste dumps. If it proves necessary, this water could be neutralized before reuse.

This drainage scheme would remain in service during the 10-year post-abandonment period to ensure that water quality values downstream of the project would be maintained. The Hat Creek diversion scheme, headworks dam, and the pit rim dam would be developed to reestablish a suitable wetland habitat in the early stages of the project. All drainage ditches would be revegetated to reduce suspended solids contamination.

7.5 LAND RECLAMATION

On-site Reclamation Testing

Both laboratory and on-site testing has been undertaken to determine the properties of the waste materials as growth media and to evaluate a variety of grass and legume species for revegetation at Hat Creek.

Initial laboratory (greenhouse) studies were followed by detailed on-site reclamation testing, making use of materials generated during the 1977 Bulk Sample Program. These latter tests have demonstrated most effectively that the revegetation of waste materials is feasible at Hat Creek consistent with proposed goals for reclamation. These may be summarized as follows:

(1) Short-term goals - Control of wind and water-borne erosion,
- Aesthetics,
- Stabilization of waste;

(2) Long-term goals - Self-sustaining vegetation,
- Suitable end use - mixed agriculture and wildlife.

The field tests comprised two major programs, one to examine the revegetation potential of slopes at different angles of repose, and the other to examine the different materials and determine their characteristics as growth media. All waste dumps associated with the 1977 Bulk Sample Program were also reseeded and provided facilities for further testing.

Results of tests on simulated embankment slopes at Houth Meadows and Medicine Creek demonstrate that slopes up to 30^{o} are stable and can be reclaimed.

Revegetation of surficial materials such as colluvium (till), gravel, and baked clay can be readily achieved. Further, these soils are suitable for reclamation purposes without the addition of topsoil. This result is noteworthy: in the case of colluvium, both biomass production and seedling emergence were lower on the topsoil-treated part of the plot. Plants were healthy and showed little sign of chlorosis. The implication here is clearly that the separate stripping of topsoil has been shown to be unjustified in the presence of suitable quantities of other surficial materials.

Revegetation of non-seam mine waste, gritstone (sandstone/claystone), and bentonitic clay proved to be more difficult to achieve in the short term.

It is considered that a surface capping of surficial material would be required to satisfactorily revegetate these waste materials.

The dramatic growth in the water retention furrows constructed in bare carbonaceous shale and bentonitic clay clearly identifies the lack of moisture as a most important factor in revegetation at Hat Creek, where the annual precipitation totals only 317 millimetres.

Vegetation Species

In total 16 different species of grass and legume have been tested in these revegetation trials. The species were selected on the basis of their known characteristics and adaptation to the soils and climatic conditions at Hat Creek. To ensure that the species were both viable and available, only agronomic species were considered. Seed mixed of four and five species were devised and, in some instances, species were used individually.

Results of these field tests have identified eight species including two legumes which could be used for reclamation purposes at Hat Creek. Fall ryegrass proved to be an excellent species for short-term (1 year) revegetation. However it is an annual, and because it is particularly tall-growing and vigorous, its use would be restricted to those occasions where short-term revegetation - for example, for dust control - is required.

In addition to these agronomic species, native shrubs and forbs considered essential in the reclamation of wildlife habitats will need to be transplanted and/or propagated in the project nursery.

The selection of species for revegetation of waste dumps and related areas at Hat Creek will be largely based on these results. Mixes of approximately five species, of which three would be grasses, would be selected and seeded, mostly by harrow-seeding methods. Only in areas too steep for harrow-seeding would hydro-seeding be used. Due to the low precipitation, seeding would be carried out in late fall (September-November) or early spring (April-May), the former period being favoured in order that maximum use could be made of moisture accumulating over the winter months. Legumes may benefit from early spring seeding to reduce losses by winter kill.

Waste Dumps and Embankments

Rapid revegetation of embankments and waste dumps will stabilize exposed surfaces against erosion. Temporary reclamation will be carried out on all areas of dump surfaces left inactive for a number of years. Retaining embankments will be constructed in lifts which allow for long-term reclamation concurrently with construction. Waste dump surfaces will be reclaimed as soon as the final surface elevation is reached, to an end use similar to that which now exists: mixed wildlife and agriculture. It is anticipated that reclamation of the Houth Meadows and Medicine Creek areas will be completed by Year 25.

Areas disturbed due to construction of facilities such as the transportation corridors will be reclaimed as soon as possible following construction. Trees will be planted where appropriate to screen the development and enhance the aesthetic appearance of the project.

Reclamation Upon Project Termination

All above-ground developments not required for other purposes will be dismantled and removed. The disturbed areas will be contoured to blend into the surrounding terrain and revegetated.

At the end of the proposed project life it is anticipated that Pit No. 1 will be totally reclaimed and returned to productive use. It may prove possible to reestablish Hat Creek in its approximate original location at that time. In the No. 2 Pit area the upper benches will be resloped to 26° to provide a safer perimeter and lessen the visual impact. No resloping will be done in the lower levels of the pit. After resloping fertilizer and seed will be aerially broadcast on all pit benches. In time revegetated overburden and slide areas may be expected to creep and slide into the pit.

A protective fence to restrict access will surround the pit perimeter and those areas which may be susceptible to failure. Trees will be planted at selected points on the perimeter to screen the pit.

7.6 DISTURBED LAND

Over the life of the project it is expected that some 5770 hectares of land will be disturbed by mining, waste and ash disposal operations. It is anticipated that ultimately over 80% of this land will be reclaimed and returned to productive use - much of this reclamation will be accomplished during the life of the project. The largest portion of the land that will not be reclaimed is represented by the lower portion of No. 2 Pit, which would still contain economically recoverable coal resources. Other unreclaimed areas include transportation corridors, lagoons, and reservoirs.

A summary of disturbed land is shown in Table 7-1.

TABLE 7-1

HAT CREEK - COAL LIQUEFACTION PLANT/PREFEASIBILITY STUDY - MINING

PRELIMINARY ESTIMATE OF MAJOR AREAS DISTURBED

	AREA (hectares)
Open Pit No. 1	920
Open Pit No. 2	1,300
Houth Meadows Waste Disposal Area	680
Medicine Creek Waste Disposal Area	1,670
Anderson Creek Waste Disposal Area	900
Miscellaneous (facilities, lagoons, conveyor ways, roads, etc.)	300
Total	5,770

SECTION 8

PROJECT DEVELOPMENT PLAN AND SCHEDULE

8.1 INTRODUCTION

The objective of this section is to define the sequence of major activities that must be performed to carry the project from the completion of the prefeasibility studies through its successive stages to the continuing operation of an efficient coal mining complex. These activities are linked together to provide a schedule for this program.

The activities discussed in this section are essentially mining tasks, although it is fully recognized that close cooperation and integration with the other elements of the total project will be required.

8.2 ORGANIZATION

The first activity that must commence is the establishment of a mining organization to plan, direct, control and integrate the early project field investigations and feasibility studies. This initial group would be primarily project management and technical supported by procurement, contracts and accounting staff.

As the project progresses through feasibility to final design, construction, mine development and operation this nucleus would be expanded to provide the mine operating organization. This approach to a gradual expansion of the organization provides the continuity in the accumulation of project specific knowledge and experience that is essential to ensure the efficient, cost effective delivery of the project on schedule.

Without continuity and commitment of the staff, the schedule will be disrupted at the commencement of each major phase of the project, to permit orientation of new project personnel. At such times it is critical to ensure that major changes to established project concepts are not undertaken unnecessarily.

8.3 FIELD INVESTIGATIONS

The program of field investigations proposed has the objective of providing an adequate level of data to permit the feasibility and design stages of the project to be completed with an acceptable level of confidence. In this program the principal effort is concentrated on the No. 2 Deposit, which, unlike the No. 1 Deposit, has not been well defined by previous B. C. Hydro programs. Although mining activity is not expected in the No. 2 Deposit until after fifteen years of operation in No. 1, it is important to establish at the feasibility stage that there are no factors that jeopardize the technical and economic viability of the total project.

8.3.1 Geological Drilling

The first phase of the geological drilling program on No. 2 Deposit, which must be completed prior to the feasibility study, would have the objective of improving the definition of the limits of the deposit, its structure, stratigraphic correlation and the distribution of coal quality. The field program would consist of a surface geophysical gravity survey; 20,000 m of diamond core drilling; down-the-hole geophysical logging; preparation of descriptive geophysical logs and the preparation of samples for laboratory analysis. This would be followed by the analytical work, the interpretation of results and the preparation of geological plans and sections. The estimated duration is five to six months for the field program and a further six months for the office work.

The second phase program should be conducted prior to the feasibility study. The scope of this program would be established after the results of the first program have been evaluated.

A budget allowance of $2 million has been made for each program phase.

8.3.2 Geotechnical and Hydrology

The purpose of the first phase of this program recommended by Golder Associates would be to identify the geotechnical and hydrological factors which would be of significance in the economics of the development of No. 2 Deposit. The second phase program would be directed to solving the particular design problems identified in the first program to enable final design to be conducted.

The scope of geotechnical work in the first program includes:

(a) Investigation of interim and final slopes of the proposed pit.

(b) Investigation of the steeper slopes beyond the west side of the pit.

(c) Investigation of proposed waste embankment sites at Anderson and Ambusten Creeks.

(d) Investigation of the interaction between a waste dump at Ambusten Creek and the eastern slopes of the proposed pit.

(e) Laboratory testing of the strength of materials from No. 2 Deposit.

Hydrological work includes:

(a) Field investigations to monitor flow measurements at active discharge locations, obtain samples for water chemistry and monitor stream flows, where applicable, to determine groundwater storage possibilities.

(b) Install piezometers to measure groundwater pressures, permeabilities and investigate flow systems.

(c) Perform pumping tests in screened developed wells to assess potential inflow to the mine and to determine hydraulic characteristics for dewatering and depressurizing the bedrock.

(d) Permeability testing in coal exploration holes.

In addition to the holes specifically drilled for this program the work would be coordinated with the geological drilling program to gain the maximum data for the least cost. Data gathered in previous investigations of No. 1 Deposit will also be used where applicable.

The estimated cost of this first phase program is $2 million. For the second phase program an allowance of at least $1 million should be made.

8.3.3 Overburden Drilling

The safe disposal of waste materials at Hat Creek is predicated on the storage of weak materials behind engineered retaining embankments. The selected bucketwheel excavator-conveyor primary mining system imposes a requirement for more complete and thorough planning because of the inflexibility of the system relative to the more common truck-shovel mining systems. To permit this planning, and the scheduling of the flow of materials for waste embankment construction, further data is required prior to the feasibility study.

The program recommended for gathering this data includes detailed geological mapping of the surficial deposits and a drilling and sampling program using a Becker Hammer Drill. A budget allowance of $500,000 has been made.

8.3.4 Creek Diversion

The diversion of Hat Creek is a critical item in the project development schedule that must receive high priority attention to permit the development of the project on schedule. The diversion scheme required for this program involves the use of a 3 km diversion tunnel around the east side of No. 1 Deposit and also diversion of the Creek around the west side of No. 2 Deposit. This is a significant change from B. C. Hydro's base scheme, which, because of the smaller pit excavation, was able to handle the diversion using an open canal and No. 2 Deposit was not included in the mining plan. As a result additional field investigations must be completed to permit preparation of preliminary designs in time for inclusion in the feasibility study report.

8.3.5 Bulk Sample Program

Production of a 10-15,000 tonne sample of Hat Creek coal will be required for shipment to SASOL for testing in an operating scale Lurgi gasifier. The timing of this program is not established, but presumably would have to take place in time to provide final design data for the Coal Liquefaction Plant.

The cost and duration of this program would depend upon the agreed specifications of the coal to be tested and the depth of overburden to be removed in order to gain access to the coal. An assessment of some of the most accessible sources of coal has identified three locations that could yield satisfactory samples:

(1) An extension of the existing Trench "B" in No. 1 Deposit. This is considered the most favourable location. The existing excavation in "D" zone coal would be extended to the east and deepened, requiring dewatering of the trench, relocation of the road, power line and telephone line.

(2) An extension of the existing Trench "A" in No. 1 Deposit. This would be a straightforward mining operation with the trench being extended either north or south and deepened. In order to produce coal of the required quality it is anticipated that more careful selective mining and more extensive screening of the coal would be required.

(3) The third location identified is in No. 2 Deposit in the vicinity of DDH 75-89 and 75-62.

It may be desirable to produce a blend of the coals from two or more locations. Prior to commencing excavation, the proposed areas would be investigated by 'close spaced drilling' to ensure that quality objectives could be met.

It is expected that the total program to produce and ship the sample could be completed in six months with the first four months being consumed in selecting the site, planning, preparing specifications, tendering and mobilization of the work force and equipment. Budget cost for the on-site program would be $1 million, subject to adjustment depending on the actual site selected.

8.3.6 Crushing and Material Handling Tests

These tests are required to establish criteria for the selection and design of coal and waste conveying systems and transfer points, coal crushers and bins and silos. In addition, because of the need for careful size control on the coal fed to the gasifiers, extensive testing of coal screening characteristics will be required.

This testing program would be performed in conjuction with the bulk sample program. Estimated cost $250,000.

8.4 FEASIBILITY STUDY

The objective of the mining feasibility study is to fully define the scope of the project and establish its technical and economic viability. The feasibility report is the principal document to support project approval, financing and control of the project design, construction and development. Clearly, if the feasibility report is to meet these requirements it must be thorough, complete and based upon sound data.

The selection of Bucketwheel Excavator-Conveyor systems as the principal mining method imposes a further constraint on the completion of the feasibility study. Because of the production capacity of each of these systems, their relatively inflexible sequence of operations and the variability of coal quality in the deposit, it is necessary to plan the mine excavation in more detail than is customary in a feasibility study. As discussed in Section 8.3, adequate data for a feasibility study is available for the No. 1 Deposit but not for No. 2.

To complete the feasibility study will require 12 months. It is probable that the BWE portion of the study would be performed by German consultants because it is unlikely that sufficient qualified personnel could be assembled in North America. Budget allowance for feasibility study: $3 million.

8.5 PROJECT SCHEDULE

A major milestone chart, presented in Figure 8-1, has been developed to meet the in-service dates for the Powerplant and the Coal Liquefaction Plant adopted for this study. The capital and operating cost cash flows have been developed to match this schedule.

The schedule is extremely tight for the proper planning, design construction and commissioning of a very large, and complex mining operation: substantially larger than any existing mining operation in British Columbia. The schedule could not be classified as impossible with the level of planning completed to date. However, in order to meet the required in-service dates it will be necessary to initiate the programs identified in Section 8.3 immediately. There are risks attached to developing a large scale project on a crash basis:

(a) Insufficient planning creates errors in design and construction resulting in costly modifications.

(b) Proper management and cost controls, which require time to establish, are easily overlooked in the anxiety to get the job done.

(c) Many activities must proceed in parallel, some in advance of when they are really needed; so that should the project be terminated at an intermediate stage it is likely that more work than strictly necessary has been committed.

A review of the schedule indicates that the majority of the problems occur in the first two to three years. The principal schedule conflicts identified are:

(1) There is no allowance for project organization, program planning, consultant selection, preparing contract bid documents, tendering, bid evaluation, financial approvals.

(2) A feasibility study conducted before the completion of many of the programs identified in Section 8.3 is of limited value and may not provide a suitable basis for project financing.

(3) The time allowance for project approval appears inadequate.

(4) Bucketwheel excavator procurement is a critical item. Three years must be allowed from order placement to commissioning of the first unit to allow for design, manufacture, delivery and erection. It is unlikely that this time could be shortened significantly. Prior to placing the order, a minimum of nine months is required to prepare bid specifications, call tenders and evaluate the bids. On this schedule, these activities overlap the feasibility study and must be completed prior to the project approval date.

(5) The diversion of Hat Creek is another tight area on the schedule. The present completion date in late 1985 only allows a few months to complete necessary preparations for the start up of the first BWE. This may be possible, but probably difficult. It should be possible to advance this schedule by accelerating the early field investigations and committing to final design ahead of project approval. Because of the seasonal nature of much of the diversion work, it will probably be necessary to accelerate the diversion works schedule by a year.

It is recommended that the project development schedule be extended by at least one year, preferably two. Even with a two year extension it is still necessary to get started quickly on project organization and field investigation; delays at any stage awaiting project approvals would jeopardize completion on schedule. This would permit at least the first phase of drilling on No. 2 Deposit to be completed for inclusion in the feasibility study.

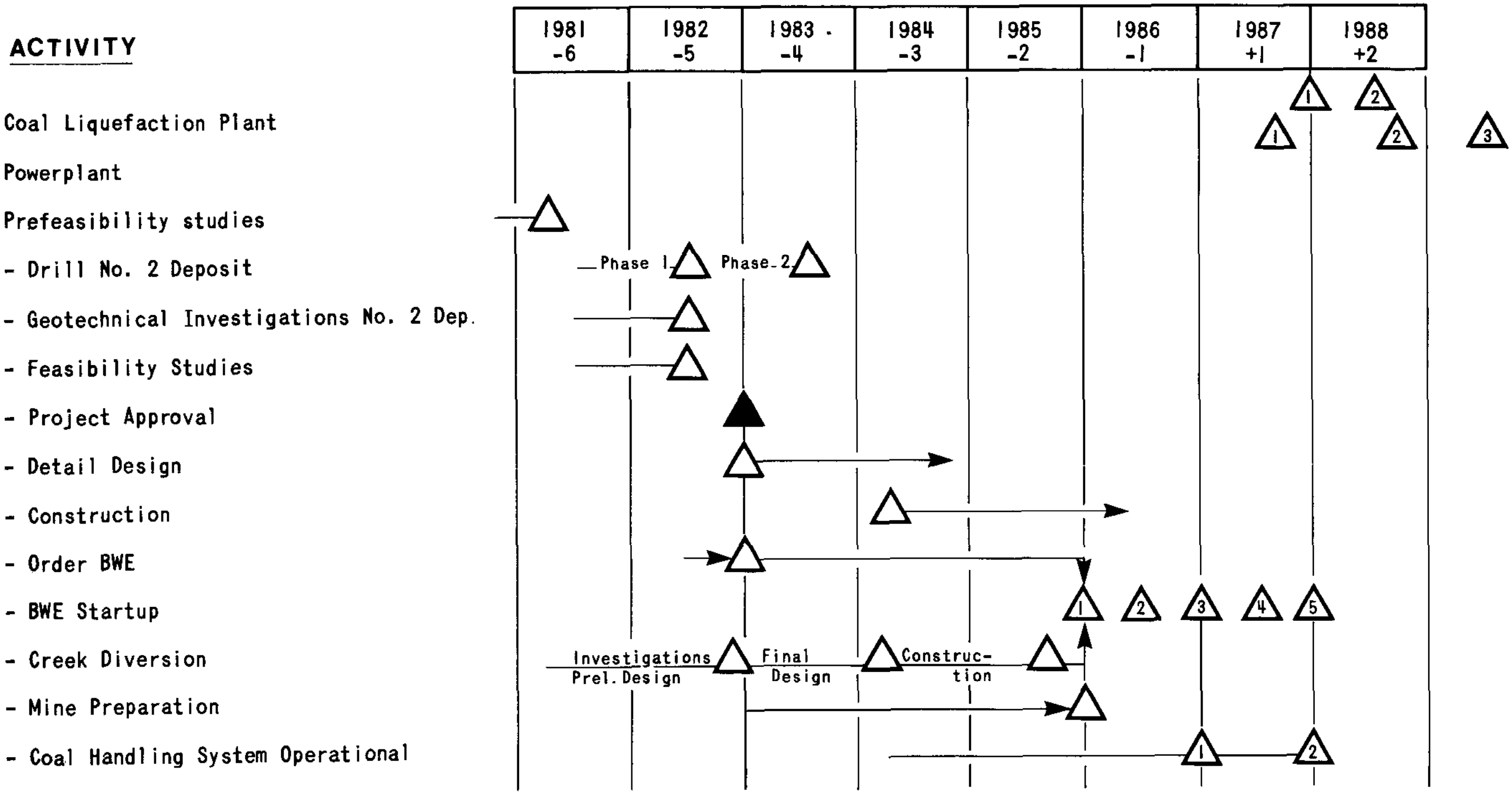


Figure 8-1

PROJECT MAJOR MILESTONE SCHEDULE - MINING

SECTION 9

COST ESTIMATES

9.1 ESTIMATING METHODS

The starting point for the development of capital and operating costs for the project is the Annual Production Schedule (Table 5-1). When the total material movement requirements and destinations are known, appropriate mining systems can be evaluated. In general terms there are two types of mining systems which must be treated somewhat differently for costing purposes. They are:

(1) The "fixed" system, which incurs costs that vary little with its production rate: a conveyor, which requires the same manpower for transporting material at 1000 t/h as at 5000 t/h;

(2) The "variable" system, which usually has smaller increments of productive capacity: a shovel-truck system operating with four trucks can easily have one truck shut-down to match a reduced production requirement and stop incurring operating costs immediately.

The fixed system is also appropriate for handling overhead and other costs that do not vary directly with output. The variable system is useful for handling equipment which may be used in many different tasks or cost centres from an equipment pool.

Figure 9-1 is a flowchart showing how operating and capital costs are developed. The process is as follows:

(1) Equipment productivity standards are established for each type of equipment to be considered. Table 9-1 shows how a sample equipment productivity standard is developed.

(2) The productivity standards are matched against the production schedule to establish the level of use and number of pieces of equipment required in each year.

(3) The annual equipment requirements are entered in Table 9-2 and, based on the service life of the equipment, an equipment purchase and replacement schedule is developed which forms the basis for establishing the capital costs.

(4) Standard operating costs/shift are developed for each type of equipment based on its estimated requirement for: operating labour, operating supplies, maintenance labour, maintenance material, maintenance overhead and fuel or power. An example of the development of standard operating costs is shown in Table 9-3.

(5) The level of equipment use in each year developed in Step (2) with the standard operating costs yield the total operating cost for the equipment. Manpower requirements are developed from the same process.

(6) Following the final selection of the type of equipment to be used, costs for auxiliary equipment, facilities and other expenses are established.

While this system only handles costs of the variable consumption type, these are usually the most difficult to handle and maintain consistency between different estimators.

Costs for the remaining functions are established using estimates of manpower and material requirements at current cost levels. For equipment such as BWE and conveyors: maintenance costs are estimated based on a percentage of the capital cost; crew requirements, power and materials consumption are developed based on experience with the equipment selected. The estimates for the BWE-conveyor-stacker systems were developed from information supplied by Krupp Canada Inc.

9.2 ESTIMATING CRITERIA

9.2.1 General

All cost estimates were developed in fourth quarter 1980 Canadian dollars. Exchange rates of C$1.15 to US$1.00 and 1.6 Deutschmark per Canadian dollar were used where appropriate. Estimates are developed on a calendar year basis with Year 1 being 1987: the year in which the first Powerplant 500 MW unit is commissioned. Interest during construction has not been included.

9.2.2 Capital Costs

Capital costs for major mining, conveying and processing equipment were based on current manufacturers' budget quotations and include freight, installation, duty and taxes. Much of this equipment is exempt from federal sales tax; but provincial sales tax was included at 4%.

Cost estimates for buildings and civil works were generally based on previous estimates for similar facilities, adjusted for inflation and pro-rated for scale. The cost estimates for the Hat Creek Diversion was developed by B. C. Hydro's Hydroelectric Generation Projects Division.

A summary of the capital costs and estimated service lives of major mining equipment is presented in Table 9-4.

9.2.3 Operating Costs

9.2.3.1 Mine Operating Schedules

Costs are developed based on the mine operating 365 days per year and twenty-four hours per day. Operations and maintenance crews will work 3-8 hour shifts per day with one swing shift crew: a total of four crews assigned to each function. General service personnel are assumed to work a 5-day, 40-hour week.

9.2.3.2 Payroll Burden

Payroll burden has been estimated at 28% of wage rates to cover the costs of a typical comprehensive benefits package.

	% of Payroll
Company Pens	6.0
Vacations	6.0
Statutory Holidays	4.2
Sick Benefits	3.0
Workers' Compensation	2.5
Group Life Insurance	0.7
Income Continuance	0.6
U.I.C.	1.3
C.P.P.	1.0
Medical Services Plan	0.8
Extended Health Care Plan	0.1
Dental Plan	0.8
Miscellaneous	1.0
Total	28.0%

9.2.3.3 Staff Salaries

Staff salaries were developed based on the 1980 salary survey conducted by the Mining Association of British Columbia, as well as in-house experience of current salary levels. Salaries used in this study are presented in Table 9-5.

9.2.3.4 Hourly Labour Rates

After a review of current wage scales in the mining industry of British Columbia, a pacesetting agreement in the Highland

Valley was selected as a basis for the study. A hierarchy of jobs appropriate to the study was established and a base hourly rate assigned. From the base rate an average rate was developed incorporating shift and weekend differentials where necessary and an allowance for overtime. This average rate was increased by 28% to produce a fully burdened, all-inclusive average hourly rate for use in estimating. The rates developed are presented in Table 9-6.

9.2.3.5 Power

Power costs developed for the various cost categories were based on a rate of 20.3 mills/kWh. Annual power costs and energy consumed (GWh) are presented in Tables 9-8 and 9-9.

9.2.3.6 Fuel

Equipment operating costs/shift were developed for fuels using the following quoted bulk rates (October 1980):

Diesel Fuel	$0.22/litre
Gasoline	$0.24/litre

Estimated annual costs and consumption of these fuels are presented on Table 9-9.

9.3 CAPITAL COST CENTRE DESCRIPTIONS

A summary of capital costs is presented in Table 9-7.

Account 90000 Engineering and Construction

The engineering and construction costs include the capital costs of project design and construction management for the seven year development period; development drilling, surveys and testing programs; construction site costs for temporary buildings, power, roads, etc.; construction camp operating costs; insurance; security; and an allowance for the owners staff to direct and control the work.

Account 91000 Mine Property Development

Includes the construction costs of permanent roads in the service area, mine water supply, sewage and drainage systems, electrical distribution to the in-pit substations, fuel distribution stations and site improvements. Land costs are not included.

Account 92000 Buildings & Structures

This account includes capital costs, at both No. 1 and No. 2 Deposits, for the following: administration office; maintenance shops and warehouses; mine dry; mine service buildings; bulk fuel and lube storage; and equipment, tools and furnishings.

Account 93000 Mining Equipment

This account includes estimated capital costs for initial and replacement purchase of mobile mining; auxiliary and support equipment; bucketwheel excavators; and initial spare parts purchase.

Account 94000 Coal Conveying, Screening and Blending

Account 95000 Waste Disposal Equipment

These accounts include estimated capital costs for all conveying, screening, crushing, stacking, blending, reclaiming and sampling equipment; as well as civil and construction costs for this equipment. An allowance for initial spare parts purchase is also included.

Account 96000 Reclamation and Environment Protection

These costs provide the facilities and equipment for a reclamation program starting with early construction and ending 10 years after mining ceases. It includes the buildings, laboratory and field equipment, and the revegetation greenhouses. Equipment for major earthmoving tasks is provided for under Mining Equipment (Account 9300).

Account 97000 Creek Diversion

Capital cost of investigations, design and construction of diversion works.

Account 98000 Construction Camp

Capital costs only.

Account 99000 Housing

Covers cost of down payment on 1400 housing units for employees. It is assumed that remaining costs will be covered by mortgage to be paid by employees.

Account 100000 Land Acquisition

Cost of acquisition of surface rights on land held in fee simple. Estimated 2,250 hectares @ $2,000 per hectare. It is assumed that Crown land will be obtained under coal lease. Lease payments provided for in operating costs.

Account 110000 Previous Studies

This represents sunk costs for expenditures made in drilling, evaluating and testing the coal deposits.

Account 120000 Contingency

The contingency allowance is designed to compensate for omissions in estimating caused by limits on the degree of detail that can be evaluated in a study at this level. For the No. 1 Deposit phase of the project a contingency of 15% has been assumed. Because there is a lower level of knowledge of No. 2 Deposit a 20% contingency has been used for the later years of the project life.

9.4 OPERATING COST CENTRE DESCRIPTIONS

Summaries of the operating costs are presented in Tables 9-8 and 9-9.

Account 100 and 200 Drilling and Blasting

The costs in these accounts cover all the equipment, manpower and materials charges for drilling and blasting an estimated 10% of the waste materials in No. 1 Deposit and 5% in No. 2.

Account 300 Loading

This account accumulates the costs of operation of the bucketwheel excavators and the hydraulic shovels in excavating coal and waste and discharging it onto conveyors and trucks respectively.

Account 400 Hauling

This account accumulates the operating costs of trucks carrying coal and waste to the truck dumping stations or direct to the waste dumps.

Account 500 Coal Handling System

This account includes all operating costs (except power), for the coal conveying, screening, crushing, blending reclaiming and sampling systems supplying coal to the powerplant and coal liquefaction plants.

Account 600 Waste Handling System

This account includes all operating costs, (except power), for the screening conveying and stacking of all waste materials into the respective dump areas.

Account 700 Auxiliary Equipment

This account includes the operating costs of the fleet of mobile auxiliary equipment used in a wide variety of applications in all areas of the mine operations.

Account 800 Power

This account includes the cost of electrical power for bucketwheel excavators, loading shovels, coal and waste handling equipment, and mine buildings.

Account 900 General Mine Expense

901 Pit Dewatering and Drainage

Provides for the operation of the mine dewatering wells and the in-pit sumps and pumps.

902 Electrical Maintenance

This account includes the costs of maintenance and repairs to the power distribution system, both in-pit and ring main, as well as maintenance to all electrical services to the mine facilities.

903 Road Construction and Maintenance

Costs for mine roads: in the pit; on waste dumps; conveyor access roads. Includes excavation and placing of material, crushing, snow removal, etc.

904 Mine Service Vehicles

This account includes the operating costs for mine mobile support equipment such as service trucks, cranes, welding machines, etc.

905 Field Lubrication and Fuelling

This account includes the cost of both operating and maintenance labour plus supplies and parts for the fuel islands, fuel trucks and lube trucks.

906 Land Reclamation

Includes costs of staff, greenhouses and storage buildings, surface regrading, placement of soil and buffer materials, and seeding and fertilizing.

907-909 Mine Supervision and Engineering

Covers costs of mine and maintenance supervision and engineering: salaries and operating expenses. The exception is first line mobile equipment maintenance supervision which is charged directly to the equipment.

910 Mine Communications

Operation of mine dispatch system and maintenance of mobile radios.

911-912 Mine Transportation

Operating and maintenance cost of mine buses used for transporting personnel from mine dry to work place. Costs of operating pickups etc. used by mine staff.

913 Mine Training

Salaries and expenses of operating the mine training department

914 Quality Control Drilling

Costs of operating drilling equipment to gather data from closely spaced holes for short term production scheduling and quality control.

Account 1000 Mine Overhead

1001-1004 Management and Administration

Salaries, supplies and other operating expenses for the management, administrative and human resources functions. Administration services provided for encompass: security, fire protection, building maintenance and janitorial service, garbage disposal, water and sewage etc.

1005 Local Taxes

An allowance for local property taxes has been made of 1½% of the replacement cost of capital assets. Cost figures were supplied by the Ministry of Industry and Small Business Development.

1006 Insurance

Allowance for all risk insurance premiums is 0.25% of total capital assets.

Account 1100 Head Office Expense

A lump sum provision of $5 million per year for the operation of a corporate head office.

Account 1200 Creek Diversion

Operating and maintenance costs of the system. Principally pumping costs from the pit rim dam.

Account 1300 Royalties and Lease Payments

Provincial coal royalties are payable at 3½% of mine head value. Calculation by the Ministry of Industry and Small Business Development. Annual coal lease and licence payments are included in this category.

Account 1400 Provincial Capital Tax

Calculated as 0.2% of each year's value of capital. Calculation of the tax was by the Ministry of Industry and Small Business Development.

Account 1500 Contingency

The contingency allowances in this account are the same as those in the capital costs, i.e. 15% for No. 1 Deposit and 20% for No. 2 Deposit. The allowance was applied to all account categories except royalties and head office expenses.

9.5 MANPOWER REQUIREMENTS

9.5.1 Operating Manpower

Operating manpower starts a gradual build up almost immediately after project approval. In the first year of coal production the total estimated manpower is 923, including a 10% contingency allowance. At this stage in the operation waste production is at a much higher than average ratio to coal and the emphasis is on training crews and debugging the operation. During the full production years of No. 1 Deposit, Years 3-15, the manpower required is in the 1250-1300 range. Manpower requirements reach a peak when No. 2 Deposit is being developed and No. 1 Deposit is being mined with a high percentage of the coal being produced by truck and shovel. During this period, Years 16-20, total manpower is estimated at 1564. Thereafter the manpower requirements are reduced.

Table 9-10 presents a summary of the project operating manpower. Table 9-11 provides a detailed breakdown.

9.5.2 Manpower Availability

The projected availability of the necessary skills has not been investigated. Based on a general knowledge of conditions in the industry it must be assumed that manpower, particularly in specific trades such as heavy duty mechanics, will be in short supply and project planning must incorporate specific programs directed to solving the problem.

These programs would include:

(1) Establishing competitive benefits and housing.

(2) Extensive formal training programs must be established for operators, tradesmen and supervision. Consideration should be given to a joint effort in training with the Powerplant and Coal Liquefaction Plant or other mines in the area who all will have some common needs.

9.5.3 Construction Manpower

Because of the tight schedule field activity must start immediately after project approval is given resulting in a rapid build up of construction manpower. The initial activity will be in the development of the camp infrastructure, camp construction, site preparation and final site investigations requiring an estimated 440 construction workers in Year -4. The requirement peaks at 670 in Year -3 and drops off to 530 in Year -2. The construction manpower requirements are presented in Table 9-12.

TABLE 9-1

EQUIPMENT PRODUCTIVITY STANDARD

14.5 m^3 Hydraulic Shovel - Loading Coal

HAT CREEK COAL LIQUEFACTION PROJECT - PREFEASIBILITY STUDY 1980 - MINING

Material density = 1.5 t/bank m^3

Bucket size (14.5 m^3) x Swell factor ($\frac{1}{135\%}$) x Fill factor (0.9) = 9.7 m^3/cycle

Cycles/hour (3600/35 s/cycle) x 9.7 m^3/cycle = 1000 m^3/h output

Hours/shift (8) x Operating efficiency (0.70) = 5.6 h/shift productive time

Output/shift = 5600 m^3

Scheduled shifts/year (1062) x Physical availability (.80) x Use of availability (.85)

= 722 productive shifts/year

Annual output = 722 shifts x 5600 m^3

= 4.0 million m^3 (6.0 million tonnes)

LEGEND

DENOTES NEW PURCHASE — $^{}2^{}$ — DENOTES REPLACEMENT PURCHASE; DENOTES TOTAL NUMBER ON SITE

(M10)

TABLE 9-2
HAT CREEK COAL LIQUEFACTION PROJECT
PREFEASIBILITY MINING STUDY

PURCHASE AND REPLACEMENT SCHEDULE

DESCRIPTION		PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																																			TOTAL
		-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	
Drills	-Truck Mounted						$^{1}1^{0}$				$^{0}1^{1}$				$^{0}1^{1}$				$^{0}1^{1}$				$^{0}1^{1}$	$^{1}2^{0}$			$^{0}2^{1}$	$^{0}2^{1}$				$^{0}1^{1}$				$^{0}1^{1}$				$^{0}1^{1}$			11
	-Air-Trac						$^{1}1^{0}$				$^{0}1^{1}$				$^{0}1^{1}$				$^{0}1^{1}$				$^{0}1^{1}$	$^{1}2^{0}$			$^{0}2^{1}$	$^{0}2^{1}$				$^{0}1^{1}$				$^{0}1^{1}$				$^{0}1^{1}$			11
Shovel	-Demag H241				$^{2}2^{0}$					$^{1}3^{0}$	$^{1}4^{0}$				$^{0}4^{2}$				$^{1}5^{0}$	$^{0}5^{1}$	$^{0}5^{1}$				$^{0}5^{2}$										$^{0}1^{1}$								12
Front-end Loader	- 5.4 m³				$^{1}1^{0}$		$^{1}2^{0}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$			$^{1}2^{1}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$		25
	- 9.6 m³				$^{2}2^{0}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$		$^{1}3^{0}$	$^{0}3^{2}$		$^{0}3^{1}$			$^{0}1^{1}$			$^{0}1^{1}$			$^{0}1^{1}$			$^{0}1^{1}$			22
	- 11.5 m³																							$^{2}2^{0}$			$^{0}2^{2}$																4
Haulage Truck	- 32t						$^{2}2^{0}$	$^{1}3^{0}$					$^{0}3^{2}$	$^{0}3^{1}$				$^{1}4^{0}$	$^{0}4^{2}$	$^{0}4^{1}$				$^{0}4^{1}$	$^{2}6^{2}$						$^{0}2^{2}$						$^{0}2^{2}$						19
	- 91t (coal box)									$^{4}4^{0}$	$^{6}10^{0}$		$^{6}16^{0}$	$^{3}19^{0}$				$^{7}22^{0}$	$^{1}17^{6}$		$^{3}22^{2}$	$^{6}25^{0}$					$^{1}12^{0}$																45
	- 154t (rock box)				$^{7}7^{0}$		$^{2}9^{0}$																	$^{5}5^{0}$								$^{0}5^{5}$								$^{0}3^{2}$			21
Scraper	-Cat 637				$^{6}6^{0}$			$^{0}6^{6}$			$^{0}5^{5}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}2^{2}$			$^{0}1^{1}$			$^{1}2^{1}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$			$^{0}2^{2}$		42
Dozer (track)	- Cat 955				$^{4}4^{0}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}4^{4}$			$^{0}2^{2}$			$^{0}2^{2}$		48
	- Cat D7				$^{1}1^{0}$			$^{0}1^{1}$			$^{0}1^{1}$	$^{1}2^{0}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{1}2^{0}$			$^{0}1^{1}$			$^{0}1^{1}$			$^{0}1^{1}$			$^{0}1^{1}$		19
	- Cat D8				$^{2}2^{0}$		$^{2}4^{0}$	$^{3}7^{2}$	$^{2}9^{0}$	$^{0}9^{2}$	$^{0}9^{5}$	$^{0}9^{2}$	$^{0}9^{2}$	$^{0}9^{5}$	$^{0}9^{2}$	$^{0}9^{2}$	$^{0}9^{5}$	$^{0}9^{2}$	$^{0}9^{2}$	$^{0}9^{5}$	$^{0}9^{2}$	$^{0}9^{2}$	$^{3}12^{5}$	$^{0}12^{2}$	$^{2}14^{2}$	$^{0}14^{8}$	$^{0}14^{2}$	$^{0}12^{2}$	[illegible]15^{3}		$^{0}13^{2}$	$^{0}13^{11}$		$^{0}13^{2}$	$^{0}13^{11}$		$^{0}13^{2}$	$^{0}6^{4}$		$^{0}6^{2}$	$^{0}4^{2}$		122
	- Cat D9				$^{4}4^{0}$		$^{1}5^{0}$	$^{1}6^{4}$	$^{1}7^{0}$	$^{1}8^{1}$	$^{0}8^{5}$	$^{0}8^{1}$	$^{0}8^{2}$	$^{0}8^{5}$	$^{0}8^{1}$	$^{0}8^{2}$	$^{0}8^{5}$	$^{0}8^{1}$	$^{0}8^{2}$	$^{0}8^{5}$	$^{0}8^{1}$	$^{0}8^{2}$	$^{1}8^{4}$	$^{0}8^{1}$	$^{0}7^{2}$	$^{1}8^{4}$	$^{0}8^{1}$	$^{2}8^{0}$	$^{2}7^{2}$		$^{0}6^{2}$	$^{0}6^{4}$		$^{0}6^{2}$	$^{0}6^{4}$		$^{0}6^{2}$	$^{0}6^{4}$			$^{0}4^{4}$		87
Dozer (wheel)	- Cat 824B				$^{1}1^{0}$		$^{1}2^{0}$			$^{0}2^{1}$		$^{0}2^{1}$			$^{0}2^{1}$		$^{0}2^{1}$		$^{1}3^{0}$	$^{0}3^{1}$		$^{0}3^{1}$		$^{1}4^{1}$	$^{0}4^{1}$		$^{1}5^{1}$		$^{1}5^{1}$			$^{0}3^{1}$		$^{0}3^{2}$			$^{0}3^{1}$		$^{0}3^{2}$				22
Compactor	- Cat 825B				$^{1}1^{0}$										$^{0}1^{1}$										$^{0}1^{1}$										$^{0}1^{1}$								4
Grader	- Cat 14G				$^{1}1^{0}$		$^{1}2^{0}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$	$^{1}3^{0}$	$^{0}3^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$		$^{0}2^{1}$	$^{0}2^{1}$			25
	- Cat 16G				$^{2}2^{0}$			$^{0}1^{2}$		$^{1}2^{0}$	$^{0}2^{2}$		$^{0}3^{1}$	$^{0}3^{2}$		$^{0}3^{1}$	$^{0}3^{2}$		$^{1}4^{1}$	$^{0}4^{2}$		$^{0}4^{2}$	$^{0}4^{2}$	$^{1}5^{0}$	$^{0}5^{2}$	$^{0}5^{2}$	$^{1}6^{1}$		$^{1}3^{0}$	$^{0}3^{2}$		$^{0}3^{1}$	$^{0}3^{2}$		$^{0}3^{1}$	$^{0}3^{2}$		$^{0}3^{1}$	$^{0}3^{2}$		$^{0}3^{1}$		41
Miscellaneous	- Backhoe (1m³)				$^{1}1^{0}$						$^{0}1^{1}$						$^{0}1^{1}$				$^{1}2^{0}$		$^{0}2^{1}$				$^{0}2^{1}$	$^{1}2^{0}$					$^{0}2^{1}$	$^{0}2^{1}$					$^{0}2^{1}$	$^{0}2^{1}$			11
	- Water wagon (26kL)				$^{2}2^{0}$					$^{1}3^{2}$					$^{0}3^{3}$					$^{0}3^{3}$			$^{1}4^{0}$		$^{0}4^{3}$			$^{0}4^{1}$	$^{1}5^{0}$				$^{0}2^{1}$	$^{0}2^{1}$				$^{0}2^{1}$	$^{0}2^{1}$				21

TABLE 9-3

DEVELOPMENT OF STANDARD OPERATING COSTS

Hat Creek Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

300 - LOADING

Account item: 320 and 330 Loading Coal and Waste Material with 14.5 m^3 Hydraulic Shovel

Expense or Position Title	Basic Unit	Price Per Unit	Units Per Shift	Total Cost Per Shift	Man-Shifts Per Productive Shift
Operating Labour					
(a) Operator	MH	17.82	10.57	188.36	1.32
(b) Helper/Oiler	MH	16.29	10.57	172.19	1.32
Subtotal				360.56	
Operating Supplies					
(a) Wear Parts	$			76.44	
(b) Power	kW	0.025	2,486.5	62.16	
Subtotal				138.60	
Total Operating				499.16	(437.00) excl. power
Maintenance Labour					
(a) Repair	MH	18.20	11.20	203.84	1.40
(b) Service	MH	16.29	0.56	9.12	0.06
Subtotal				212.96	
Maintenance Parts and Supplies					
(a) Repair	$			327.15	
Subtotal				327.15	
Maintenance Overhead					
(a) Staff	$/repair labour hour	3.15	11.20	35.28	
(b) Repair	MH	18.20	1.34	24.46	0.17
(c) Supplies	$			49.28	
Subtotal				109.02	
Total Maintenance				649.13	

Total Operating and Maintenance Cost/Shift: $1,148.29/$1,086.13 (excl. power)

TABLE 9-4

EQUIPMENT CAPITAL COSTS AND SERVICE LIVES
Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

Item	Capital Cost 1980 $ FOB Hat Creek	Service Life Op. Hours
Drills		
Auger: truck-mounted	217,000	15,000
Air-Trac c/w Compressor	143,000	15,000
Shovels (hydraulic)		
Poclain 1000 CK	1,572,000	36,000
Demag H241	2,690,000	45,000
Front-end Loader		
5.4 m^3	362,000	15,000
9.6 m^3	720,000	15,000
11.5 m^3	840,000	15,000
Haulage Truck		
32 t	345,000	25,000
91 t (coal box)	587,000	33,000
154 t (rock box)	807,000	33,000
Scraper		
Cat 631	429,000	15,000
Cat 637	496,000	15,000
Dozer (track)		
Cat 955	116,000	15,000
Cat D7	230,000	15,000
Cat D8 with ripper	300,000	15,000
Cat D9 with ripper	426,000	15,000
Dozer (Wheel)		
Cat 824B	255,000	25,000
Compactor		
Cat 825B	293,000	25,000
Vibratory (towed)	50,000	20,000
Grader		
Cat 14G	220,000	20,000
Cat 16G	315,000	25,000
Miscellaneous		
Backhoe (1 m^3)	212,000	30,000
Water wagon 26 kL	327,000	25,000

TABLE 9-5

Annual Salaries of Mine Staff
Coal Liquefaction Project
Pre-feasibility Study-Mining
1980

Position	Base Rate Per Annual	Payroll Burden 28%	Rate Per Annum
Mine Manager	64,400	18,400	82,800
Asst. Mine Manager	58,650	16,100	74,750
Superintendent, Mine	48,300	13,570	61,870
Superint. Plant & Maint.	44,275	12,420	56,695
Superint., Engineering	44,275	12,420	56,695
Superint., Administration	41,400	11,615	53,015
Superint., Human Resources	41,400	11,615	53,015
Asst. Mine Superintendent	41,400	11,500	52,900
Asst. Mtce. Superintendent	41,400	11,500	52,900
Chief Accountant	37,950	10,580	48,530
Chief Geologist	37,950	10,580	48,530
Pit Engineer	37,950	10,580	48,530
Superint. Envir. & Recr.	37,950	10,580	48,530
Senior Accountant	35,650	10,005	45,655
Purchasing Agent	35,650	10,005	45,655
Wareh. & Stores Superint.	35,650	10,005	45,655
Personnel & Labour Rel.	35,650	10,005	45,655
Sen. Mine Eng./Geologist	35,650	10,005	45,655
Chief Electrician	35,650	10,005	45,655
Master Mechanic	35,650	10,005	45,655
Environm. Pollution Eng.	35,650	10,005	45,655
Mine Shift Supervisor	35,650	10,005	45,655
Electrical/Mechanical Eng.	35,650	10,005	45,655
Mine Pit Foreman (Shift Boss)	32,775	9,200	41,975
Assay Lab. Superint.	32,775	9,200	41,975
Chief Surveyor	32,775	9,200	41,975
Chief Sampler - Gr. Off.	32,775	9,200	41,975
Safety Supervisor, Training	32,775	9,200	41,975
Public Rel. Information	29,900	8,395	38,295
Roads & Pioneer Foreman	29,900	8,395	38,295
Crushing Plant Foreman	29,900	8,395	38,295
Payr. Acct. Cost. Supervisor	29,900	8,395	38,295
Secr. Sr., Confidential	27,600	7,705	35,305
Jr. Engineer/Geologist	26,450	7,360	33,810
Chief Security Superint.	25,300	7,130	32,430
First Aid Off. & Ch. Fireman	24,150	6,785	30,935
Sr. Technician/Surv./Geol.	31,050	8,740	39,790
Sr. Tech. Draftsman, Design	28,750	8,050	36,800
Surveyor-Transitman	26,450	7,360	33,810
Payroll clerks/dr. Wareh.	25,300	7,130	32,430
Accounting Clerks	25,300	7,130	32,430
Technicians/Draftmen	24,150	6,785	30,935
First Aid Att./Fireman	24,150	6,785	30,935
Training Officers-Asst.	24,150	6,785	30,935
Warehouse Clerks (Shipper)	25,150	6,785	30,935
Security Guards	23,000	6,440	29,440
Samplers	22,310	6,210	28,520
Rodmen-Surv. Helper	22,310	6,210	28,520
Secretaries	20,930	5,865	26,795
Mail Truck Driver	18,400	5,175	23,575
Typists/Stenogr.	17,998	5,002	23,000
Timekeeper/Mail Clerk	17,595	4,945	22,540
Keypunch Op. Payr. Clerk	17,940	5,060	23,000

TABLE 9-6

LABOUR RATES FOR MINE OPERATIONS STAFF
(October 1980 Dollars)
Coal Liquefaction Project-Prefeasibility Study 1980 - Mining

	Continuous Shift Hourly Rate[1]	Day Shift Hourly Rate[2]
Journeyman Tradesman BWE Operator	18.20	17.60
Shovel Operator Waste Stacker Operator	17.82	17.20
Crane Operator Rotary Driller Coal Stacker & Reclaimer Operator Conveyor Controller BWE Oiler	17.43	16.81
Blaster Secondary Crushing Operator Tireman	17.06	16.44
Production Truck Driver Dozer Operator Grader Operator Front-end Loader Operator	16.68	16.04
Shovel Helper Drill Helper Service Truck Driver Airtrac Driller Primary Crushing Operator Lube Serviceman	16.29	15.64
Utility Truck Driver Blaster's Helper Warehouseman Conveyor Patrolman	15.91	15.24
Counterman	15.54	14.87
Pumpman	15.15	14.48
Labourer	14.77	14.08

[1]Hourly rates include: trade base rate; shift differential; overtime allowance; payroll burden.

[2]Hourly rates include: trade base rate; overtime allowance; payroll burden.

(M11)

TABLE 9-7

HAT CREEK COAL LIQUEFACTION PROJECT
PREFEASIBILITY MINING STUDY

SUMMARY OF CAPITAL COSTS BY MAJOR COST CENTRE

ACCOUNT CODE	DESCRIPTION	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																																					TOTAL
		-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	
90000	ENGINEERING AND CONSTRUCTION	6215	9252	11728	15315	14646	16442	4491	2031												1900	400	400																				82820
91000	MINE PROPERTY DEVELOPMENT				17134	22547	9763	798	200	320	860	403	450		1150	950	500		100	350	5260	15342	14790	8490	990	790	700				50	290					150						102377
92000	BUILDINGS AND STRUCTURES				10400	10500	3500	150	120		220	120		240	220	500	150		120		150	240	5350	5850	90		120		150	120	70	240	120		260	650		150	120		220	60	40200
93000	MINING EQUIPMENT			17400	50884	53152	37536	36885	2334	9925	17707	2342	7532	12271	11230	4179	11309	6875	11578	15024	8245	32163	39450	45473	16280	10759	16251	4350	17933	2400	4891	13063	2625	4323	13520	2937	4251	6654	2894	4661	5741	344	567369
94000	COAL CONVEYING SCREENING & BLENDING			2835	4270	43037	39638	53128	22223	2016						1264	1293					1264	7225		11200	11200	15530	3400	13100														232623
95000	WASTE DISPOSAL EQUIPMENT			26065	41030	79890	47782	34885	2016	2016						1264	1264					50167	57517	60360			11050		11000														426306
96000	RECLAMATION				179	254	65	66	31	118	144	26	5	26	149	113	31	26	5	144	144	5	126	91	133	159	26	5	26	169	113	31	26	5	144	144		5		138	128	5	3005
97000	CREEK DIVERSION	500	570	2800	15680	11550													80	1310	6560	8200	250																				47500
98000	CONSTRUCTION CAMP			5000	5000																																						10000
99000	HOUSING			5600	5600	5600	5600	5600																																			28000
100000	LAND ACQUISITION	4500																																									4500
110000	PREVIOUS STUDIES	17500																																									17500
120000	CONTINGENCY	1007	1473	10714	24824	36176	24049	20400	4343	2159	2840	434	1198	1881	1912	1241	2182	1035	1782	2524	3339	21556	25022	24053	5793	4582	8735	1551	8442	558	1025	2725	554	866	2785	746	880	1362	603	960	1218	82	259591
	TOTAL CAPITAL COSTS	29722	11295	82142	190316	277352	184375	156403	33298	16554	21771	3325	9185	14418	14661	9511	16729	7936	13665	19352	25596	129337	150130	144317	34486	27490	52412	9306	50651	3227	6149	16349	3325	5194	16709	4477	5281	8171	3617	5759	7307	491	1821791
	TOTAL CUMULATIVE CAPITAL COSTS TO FULL PRODUCTION	29722	41017	123159	313475	590827	775202	931605	964903	981457	1003228	1006553	1015738	1030156	1044817	1054328	1071057	1078993	1092658	1112010	1137606	1266943	1417073	1561390	1595876	1623366	1675778	1685084	1735735	1738962	1745111	1761460	1764785	1769979	1786688	1791165	1796446	1804617	1808234	1813993	1821300	1821791	1821791

CANADIAN $000 OCTOBER, 1980

TABLE 9-8

HAT CREEK COAL LIQUEFACTION PROJECT
PREFEASIBILITY MINING STUDY

SUMMARY OF OPERATING COSTS BY MAJOR COST CENTRE

ACCOUNT CODE	DESCRIPTION	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																																			
		-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	TOTAL
100	DRILLING						153	239	282	318	300	300	297	276	300	300	300	309	308	323	286	297	259	490	490	437	388	286	230	230	230	230	230	230	230	230	230	230	118	118	59		9533
200	BLASTING						437	682	806	909	861	861	850	787	853	853	853	883	880	923	818	850	739	1400	1400	1248	1109	816	656	656	656	656	656	656	656	656	656	656	336	336	168		27218
300	LOADING				1268	1268	4937	8302	11087	13396	13449	13493	13421	13437	13405	13405	13405	13749	14200	14204	14601	14672	13288	22612	25398	23889	23656	21967	19751	21979	21979	21979	21979	21979	18844	16754	14664	14664	14298	8028	4710	2438	560555
400	HAULING				4241	4241	5332	5392	5592	7333	5858	5935	5105	4269	2627	2669	2636	2642	3207	2828	4749	7665	5551	7935	7747	9775	11067	10236	2708	2708	2708	2708	2708	2708	2708	2708	2708	2708	1354	1354	677		164875
500	COAL HANDLING SYSTEM						1821	3774	7096	9255	10254	10290	10374	10374	10374	10374	10510	10510	10510	10510	10510	10510	10235	11564	11926	11600	11741	11547	11205	11419	11419	11419	11309	11309	10822	10497	10172	8288	6388	5414	4927	4602	344849
600	WASTE HANDLING SYSTEM						3642	7210	10048	11850	11850	11850	11850	11850	11850	11850	11988	11988	11988	11988	11988	11988	11050	19187	21482	19780	19438	18076	18805	20808	20808	20808	20808	20808	17728	15675	13621	13621	13506	7347	4267	2213	495614
700	AUXILIARY EQUIPMENT			2500	5568	5646	6774	4711	5972	6205	6333	5918	5911	5909	5905	5909	5905	5997	6174	6180	6231	5836	6409	6595	7026	7218	7220	7166	4794	4794	4794	4794	4794	4794	4794	4794	4794	4794	2750	2750	2750	2750	210158
800	POWER			406	459	459	1825	3816	7141	8711	10012	10014	10010	10011	10010	10010	10279	10294	10312	10313	10329	10333	10035	14347	15590	14719	14548	13832	13535	14705	14705	14705	14705	14705	13090	12001	10919	10919	9000	5755	4126	3036	383721
900	GENERAL MINE EXPENSE			672	6318	9827	14001	15946	15787	16590	16732	17472	15487	15391	15388	15488	16076	15804	15886	16012	16075	16222	16512	16743	17599	18386	18346	18011	15785	15222	14227	15702	15705	14225	14802	15688	15683	15959	15307	13729	13734	*24210	596749
1000	OVERHEAD		444	1878	5922	11347	17843	21214	24100	24228	23056	22035	20905	19975	19199	18500	17511	16998	16449	16036	15749	15833	17354	18859	19893	19531	19182	19062	18555	18494	17892	17406	17063	16609	16245	16032	15699	15416	15187	14933	14734	14561	671928
1100	HEAD OFFICE EXPENSE		5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	5000	200000
1200	CREEK DIVERSION						100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	100	250	250	250	250	250	250	250	250	250	250	250	250	250	250	250	250	250	250	250	6450
1300	ROYALTIES AND LEASES		160	160	160	160	160	895	3502	5909	6465	6687	6688	6687	6686	6688	6416	6415	6416	6415	6415	6415	6416	6415	6416	6416	6416	6415	6415	6416	6416	6416	5925	5925	5924	5924	5924	5924	2018	2018	2018	2018	197274
1400	PROVINCIAL CAPITAL TAX	66	94	273	697	1281	1688	2039	2038	1823	1645	1452	1294	1165	1053	944	863	774	707	660	631	813	1015	1180	1106	1026	1006	903	894	792	708	655	582	522	492	441	398	366	329	300	278	246	35239
1500	CONTINGENCY		66	818	3567	4918	8515	10693	13156	14819	14803	14725	14132	13842	13487	13403	13420	13377	13487	13397	13700	14131	18287	23946	25710	25317	25339	24200	21205	22203	21884	22082	21991	21604	19984	19007	17830	17451	15649	11953	10030	*12186	620314
	TOTAL OPERATING COSTS	66	5764	11707	33200	44147	72228	90013	111507	126446	126700	126132	121424	119070	116237	115493	115262	114640	115624	114889	117182	120665	122250	156521	167033	164592	164706	157767	139788	145676	143676	144810	143705	141324	131569	125657	118548	116246	101490	79285	67728	*73510	4524477

* INCLUDES 10 POST PRODUCTION YEARS

CANADIAN $000 OCTOBER, 1980

TABLE 9-9
HAT CREEK COAL LIQUEFACTION PROJECT
PREFEASIBILITY MINING STUDY

COST SUMMARY
BREAKDOWN OF SPECIFIC COSTS

M13

ACCOUNT CODE	DESCRIPTION	PRE-PRODUCTION PERIOD -6	-5	-4	-3	-2	-1	PRODUCTION PERIOD 1	2	3	4	5	6	7	8	9	10	11	12	13
	HOURLY LABOUR																			
	OPERATING				3404	4219	8536	10653	14197	16658	16844	17340	16580	16390	16057	16097	16558	16498	16844	16816
	MAINTENANCE				2580	3375	5343	7399	10299	12218	12128	12043	11907	11733	11411	11440	11481	11637	11885	11837
	STAFF LABOUR			1573	4359	6933	8727	9263	9466	9611	9611	9460	9460	9460	9460	9460	9460	9460	9460	9460
	DIESEL FUEL																			
	OPERATING SUPPLIES (@ $0.22/litre)				1751	1772	2201	2388	2642	3083	2872	2679	2555	2387	2090	2114	2091	2170	2306	2261
	UNITS ($\times 10^3$ litres)				7960	8055	10005	10855	12010	14015	13055	12175	11615	10850	9500	9610	9505	9865	10480	10275
	GASOLINE																			
	OPERATING SUPPLIES (@ $0.24/litre)				192	288	432	576	576	576	576	576	576	576	576	576	576	576	576	576
	UNITS ($\times 10^3$ litres)				800	1200	1800	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400
	POWER																			
	POWER ($0.0203/Kwh)			406	459	459	1825	3816	7141	8711	10012	10014	10010	10011	10010	10010	10279	10294	10312	10313
	ANNUAL ENERGY (GWh)			16	18	18	73	153	286	348	400	401	400	400	400	400	411	412	412	413
	PROVINCIAL SALES TAXES (BASED ON CAPITAL COSTS)			1536	3833	6107	4109	3757	753	523	726	109	321	524	482	301	505	271	449	588

DESCRIPTION	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	TOTAL
HOURLY LABOUR																							
OPERATING	17341	17993	17072	21853	23215	22923	23186	22198	18564	19714	19714	19714	19659	19659	18069	17009	15949	15375	13551	10371	8613	7384	612,817
MAINTENANCE	12299	12839	12280	15522	16422	16344	16533	15894	13249	13958	13959	13960	13904	13908	12922	12269	11616	11039	9462	7470	6318	5442	436,322
STAFF LABOUR	9460	9460	9584	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	9721	*13159	361,324
DIESEL FUEL																							
OPERATING SUPPLIES (@ $0.22/litre)	2618	2908	2631	3191	3239	3680	3888	4395	1731	1731	1731	1731	1731	1731	1731	1731	1731	1731	1102	1102	987	870	85285
UNITS ($\times 10^3$ litres)	11900	13220	11959	14500	14720	16725	17676	19978	7869	7869	7869	7869	7869	7869	7869	7869	7869	7869	5009	5009	4485	3955	387656
GASOLINE																							
OPERATING SUPPLIES (@ $0.24/litre)	576	576	576	576	576	576	576	576	576	576	576	576	576	576	576	576	576	576	432	432	288	288	20208
UNITS ($\times 10^3$ litres)	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	2400	1800	1800	1200	1200	84200
POWER																							
POWER ($0.0203/Kwh)	10329	10833	10035	14347	15590	14719	14548	13832	13535	14705	14705	14705	14705	14705	13090	12001	10919	10919	9000	5755	4126	3036	383721
ANNUAL ENERGY (GWh)	413	413	401	574	624	589	582	553	541	588	588	588	588	588	524	480	437	437	360	230	165	121	15349
PROVINCIAL SALES TAXES (BASED ON CAPITAL COSTS)	429	2705	3235	3204	959	740	1375	275	1256	106	207	516	107	166	532	145	178	263	118	197	233	16	41856

CANADIAN $000 OCTOBER, 1980

TABLE 9-10

MINE OPERATING MANPOWER SCHEDULE - SUMMARY

Coal Liquefaction Project-Prefeasibility Study 1980-Mining

	-3	-2	-1	1	2	3	4	5	6-15	16-20	21-30	31-35
Management and Reclamation P.C.	24	34	48	50	50	50	50	50	50	53	53	53
Administration and Site Services	48	95	158	164	164	164	164	164	164	164	164	164
Human Resources	26	35	39	39	39	39	39	35	35	39	39	39
Mine Supervision-Engineering	15	25	32	37	39	39	39	39	39	39	39	39
Mine Supervision-Operations	7	13	24	31	36	39	39	39	39	39	39	39
Maintenance Supervision	12	22	33	38	42	42	42	42	44	44	44	40
Mine Operations-Labour	62	137	237	295	394	463	467	471	484	634	520	303
Maintenance-Labour	33	94	134	185	257	305	305	305	320	410	350	235
Subtotals	227	455	705	839	1021	1141	1145	1145	1175	1422	1248	912
Contingency-10%	23	45	70	84	102	114	114	114	117	142	124	91
Totals	250	500	775	923	1123	1255	1259	1259	1292	1564	1372	1003

TABLE 9-11

SCHEDULE OF OPERATING MANPOWER REQUIREMENTS

Coal Liquefaction Project-Prefeasibility Study 1980-Mining

	-3	-2	-1	1	2	3	4	5	6-15	16-25	26-35
1. Management											
Managers	2	2	2	2	2	2	2	2	2	2	2
Industrial Engineering	4	5	5	5	5	5	5	5	5	5	5
Public Relations	1	2	2	2	2	2	2	2	2	2	2
Computer Services	1	3	4	4	4	4	4	4	4	4	4
Laboratory Services	2	4	8	8	8	8	8	8	8	8	8
Environment Protection and Reclamation	7	7	10	12	12	12	12	12	12	15	15
Secretaries, Clerical, Stenos.	7	11	17	17	17	17	17	17	17	17	17
	24	34	48	50	50	50	50	50	50	53	53
2. Administration Supt.	1	1	1	1	1	1	1	1	1	1	1
Finance-Cost Accounting and Payroll	2	4	5	5	5	5	5	5	5	5	5
Materials Management-Purchasing, Warehouse	6	12	18	18	18	18	18	18	18	18	18
Security Services	3	10	10	10	10	10	10	10	10	10	10
Fire Department	2	6	6	6	6	6	6	6	6	10	10
Janitorial Svs. & Office Bldgs. Maintenance	8	10	18	18	18	18	18	18	18	18	18
Dry Facilities			6	6	6	6	6	6	6	6	6
Communications-Radio, Telephone, Telex	2	2	6	6	6	6	6	6	6	6	6
Water, Fuel, Sewerage; Wastewaters, Garbage Disposal	3	6	11	11	11	11	11	11	11	11	11
Transportation-Bus Service and Delivery Drivers	4	10	18	24	24	24	24	24	24	24	24
Surface Labour-Road Maintenance, Plant Yard	3	6	6	6	6	6	6	6	6	6	6
Warehouse	4	8	20	20	20	20	20	20	20	20	20
Secretaries, Clerical, Stenos	10	20	33	33	33	33	33	33	33	33	33
	48	95	158	164	164	164	164	164	164	164	164

	-3	-2	-1	1	2	3	4	5	6-15	16-25	26-35
3. Human Resources-											
Supt.	1	1	1	1	1	1	1	1	1	1	1
Labour Relations Dept.	2	3	3	3	3	3	3	3	3	3	3
Training School	7	10	10	10	10	10	10	6	6	6	6
Personnel Department	2	3	3	3	3	3	3	3	3	3	3
Safety and First Aid	6	8	12	12	12	12	12	12	12	16	16
Secretaries, Stenos.	8	10	10	10	10	10	10	10	10	10	10
	26	35	39	39	39	39	39	35	35	39	39
4. Mine Engineering-											
Supt.	1	1	1	1	1	1	1	1	1	1	1
Pit Engineer	1	1	1	1	1	1	1	1	1	1	1
Planning Engineers	4	6	8	8	8	8	8	8	8	8	8
Draftsmen	2	3	3	3	3	3	3	3	3	3	3
Samplers & Grade Officer	-	2	5	5	5	5	5	5	5	5	5
Surveyors	2	3	4	4	4	4	4	4	4	4	4
Geologists	2	3	5	6	7	7	7	7	7	7	7
Geotechnical Engineers and Technician	2	2	2	2	2	2	2	2	2	2	2
Helpers (Survey, Rodmen)	1	2	4	4	4	4	4	4	4	4	4
Secretary, Clerical, Stenos	1	3	3	3	4	4	4	4	4	4	4
	15	25	32	37	39	33	33	33	33	33	33
5. Mine Supervision (Operations)											
Supt. and Assistant	2	2	2	2	2	2	2	2	2	2	2
Shift Supervisors	1	2	3	3	4	4	4	4	4	4	4
Production-Shift Foremen	1	1	3	4	4	4	4	4	4	4	4
Production-BWE Foremen	-	1	3	4	4	4	4	4	4	4	4
Production-Dump Pocket Foremen, Crusher	-	1	1	2	3	4	4	4	4	4	4
Processing-Coal Plant Foremen	-	1	1	2	3	4	4	4	4	4	4
Processing-Waste Dump Foremen	1	1	3	4	4	4	4	4	4	4	4
Pit Maintenance-Roads Foremen	1	1	3	4	4	4	4	4	4	4	4
Pit Maintenance-Drainage Foremen	1	2	4	4	4	4	4	4	4	4	4
Clerical and Steno.	-	1	3	3	4	5	5	5	5	5	5
	7	13	26	32	36	39	39	39	39	39	39

	-3	-2	-1	1	2	3	4	5	6-15	16-20	21-30	31-35
6. Maintenance Eng'g (Supervision)												
Supt. and Assistant	1	2	2	2	2	2	2	2	2	2	2	2
Engineering Design-Engineers,Draftsmen	3	4	5	5	5	5	5	5	5	5	5	5
Maintenance Planning	2	3	4	4	5	5	5	5	5	5	5	5
Operations Maintenance-Mechanical	2	3	5	8	10	10	10	10	10	10	10	10
Operations Maintenance-Electrical	1	3	5	5	6	6	6	6	6	6	6	5
Pit Equipt. Maintenance-Mechanical Super.	1	2	4	6	6	6	6	6	8	8	8	6
Surface Yard and Carpenter Foremen	–	1	2	2	2	2	2	2	2	2	2	2
Clerical and Steno.	2	4	6	6	6	6	6	6	6	6	6	6
	12	22	33	38	42	42	42	42	44	44	44	42
7. Mine Operations												
Shovel-Operators	4	4	7	8	8	11	12	12	16	17	4	2
Shovel-Oilers	4	4	7	8	8	11	12	12	16	17	4	2
Haulage Truck Drivers	16	16	21	25	25	35	36	36	48	50	14	5
BWE Systems-Operators etc.	-	50	100	110	160	160	160	160	160	290	290	180
Conveying Coal	-	15	29	30	54	72	72	76	76	76	76	38
Conveying Waste (truck/shovel systm)	3	5	5	8	8	12	12	12	12	16	8	4
Drilling & Blasting	2	2	2	8	8	12	12	12	12	15	12	6
Heavy Equipment Operators	25	30	43	59	80	90	91	91	83	88	70	37
Service Truck Drivers	2	3	6	19	19	24	24	24	25	23	10	8
Mobile Crusher	2	2	2	4	4	8	8	8	8	8	4	4
Pit Dewatering and Drainage	2	2	4	6	8	8	8	8	8	14	14	9
Labourer	2	4	9	10	12	20	20	20	20	20	14	8
	62	137	237	295	394	463	467	471	484	634	520	303

	-3	-2	-1	1	2	3	4	5	6-15	16-20	21-30	31-35
8. Maintenance Labour												
Central Shops and Services:												
H.D. Mechanic	10	22	24	30	36	44	44	44	48	48	30	20
Auto Mechanic	2	5	6	8	8	12	12	12	14	16	12	8
Tiremen	2	4	6	8	8	8	8	8	8	8	4	2
Welder	2	3	7	7	10	10	10	10	12	14	8	6
Machinist	1	2	4	4	6	6	6	6	6	8	6	4
Carpenter	1	2	3	3	4	6	6	6	8	8	8	6
Pipefitter	1	1	3	3	3	3	3	3	3	3	3	3
Painter	-	1	2	2	2	2	2	2	2	2	2	2
Radio Technician	1	2	2	3	3	3	3	3	3	3	3	3
Electrician	6	12	18	23	23	23	23	23	23	23	23	23
Crane Operator	1	2	3	5	5	5	5	5	5	5	5	5
Labourer	2	8	10	10	12	12	12	12	12	16	14	6
Field Services:												
BWE Mechanic	-	10	20	35	76	88	88	88	88	160	152	100
H.D. Mechanic	2	6	8	8	18	20	20	20	22	22	10	8
Conveyor Mechanic	-	8	11	26	30	50	50	50	53	60	60	32
Belt Vulcanizer	-	2	2	4	4	4	4	4	4	4	4	4
Lube Servicemen	2	4	5	6	9	9	9	9	9	10	6	3
Total Labour	33	94	134	185	257	305	305	305	320	410	350	235

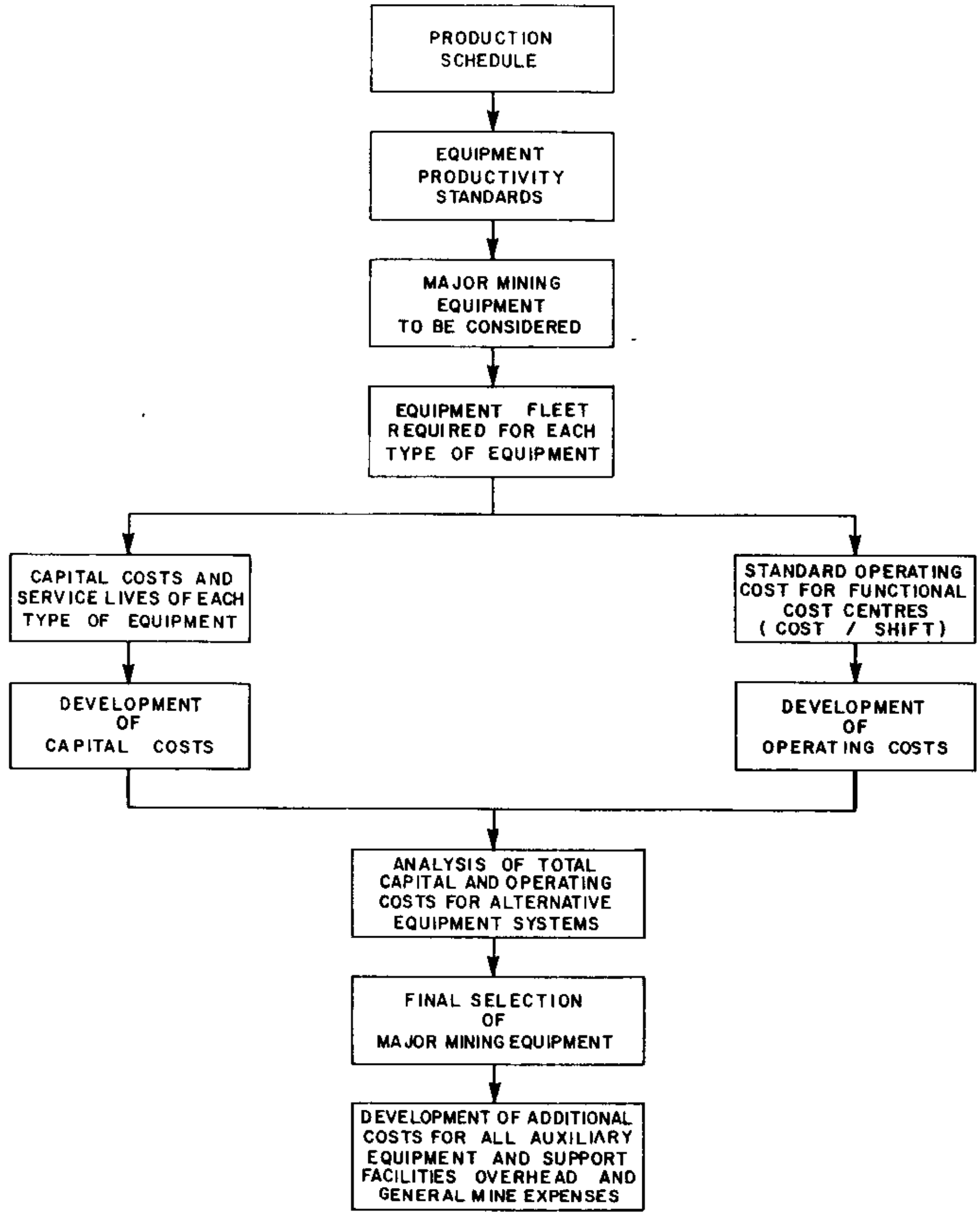


FIGURE 9-1

Overall Approach to Development of Cash Flow

SOURCE: British Columbia Hydro and Power Authority

SECTION 10

FINANCIAL ANALYSIS

10.1 INTRODUCTION

The financial analysis of the total mining costs was carried out by C. E. Wetton Associates Inc. under the direction of the Ministry of Industry and Small Business Development.

The objective of the analysis, given specified financial criteria, was to determine the unit price, in last quarter 1980 dollars, of Hat Creek coal delivered to both the Coal Liquefaction Plant and the B. C. Hydro 2000 MW (net) Powerplant. For the proposed base case, the price charged for Hat Creek coal must ensure a real rate of return of 10% (17.7% nominal) on investment before income taxes.

The price was then given to Fluor Inc. to incorporate into their analyses of the Coal Liquefaction Plant.

10.2 CRITERIA

The coal price calculation is dependent on the criteria, assumptions and methodology employed. The assumptions and financial criteria used in the analysis of the base case were as follows:

(1) Mine Operations - considered as a stand-alone private corporation subject to all applicable taxes, i.e. a single corporate entity-not part of a large corporate structure where tax burdens and advantages are shared with the parent company.

(2) Construction Period - for No. 1 Pit assumed to commence 1981 and ending in 1988 with substantial production. For No. 2 Pit commencing 1998 and ending in 2006 with substantial production.

(3) Production Period - full production assumed beginning 1 January 1989 to end December 2017, with reduced production, i.e. for powerplant only, from January 2018 to December 2021.

(4) Base Date - for economic calculations the base date was last quarter 1980 in Canadian dollars.

(5) Inflation - a general price inflation factor of 7% was applied to all capital and operating cash flows shown in Tables 9-7, 9-8 and 9-9.

(6) Rate of Return - the target before tax real rate of return was 10% (17.7% nominal if inflation included).

(7) Income Tax - taxes were estimated using 1980 rules for the federal and provincial tax systems.

(8) Debt: Equity Ratio - assumed to be 75:25. The interest rate charged on borrowed funds assumed to be 5% real (or 12.35 nominal if general price inflation is included).

(9) Royalties - calculated at 3.5% of minehead cost of coal.

10.3 ADJUSTMENTS

For the base case analysis certain adjustments were made to the costs presented in Tables 9-7, 9-8 and 9-9, For the before tax case these adjustments were as follows.

10.3.1 Capital Costs

An allowance for working capital was included in the capital costs, i.e. 15% of that years operating costs and contains the operating cost contingency factor.

10.3.2 Operating Costs

Royalties were removed from the operating cost cash flows as they are more accurately defined as tax revenues to the Provincial Government.

Other adjustments affecting the calculation of income taxes were also made, i.e. capitalizing operating costs incurred during the construction period; capital cost categories were grouped to take advantage of tax and depreciation allowances.

Further details of these adjustments and the tax calculations can be found in the Ministry of Industry and Small Business Development Report referenced in Section 11.

10.4 ANALYSIS

The present values of the cash outflows and inflows associated with the mine operation, including the necessary adjustments, were equated.

The cash outflows are the annual mine capital and operating cash requirements and the cash inflows are determined from the schedule of coal deliveries to the respective plants and the price of the delivered coal.

The analysis incorporated the inflation rate noted above, a discount rate of 10% real (at 17.7% nominal) was applied in the discounted cash flow technique. The time period covered in the study was 42 years, i.e. 7 for pre-production and 35 for production (the costs for the 10-year post-production reclamation period were included for convenience in the final production year costs, i.e. 2021).

10.5 BASE CASE RESULTS

The financial analysis of the base case, using the criteria and assumptions listed in Section 10.2, results in a delivered coal price, in last quarter 1980 dollars, of $11.26 per tonne which gives a real rate of return on investment of 10% (17.7% nominal). The after tax rate of return is calculated at 7.71% real (15.25% nominal). This estimated coal price will cover all capital, operating and tax requirements as well as give a return on owner's equity of 18.62% (nominal).

The 1980 dollar distribution of the $11.26 per tonne price is as follows:

Cash Flow Out Per Tonne

Capital Investment	$ 2.46
Operating Costs	$ 4.18
Taxes and Royalties	$ 2.61
Interest Expenses	$ 1.18
Debt Repayment	$ 0.54
Net Cash Flow (Profit)	$ 1.50
	$12.47*

Cash Flow In Per Tonne

Revenues (Sales)	$11.26
Borrowing	$ 1.20
	$12.46*

*Note: difference due to rounding of figures.

Sensitivity analyses were also carried out using: differing returns on investment and debt; equity ratios; real cost increases; alternate power cost; and varying contingency and escalation feactors.

The results of these analyses can be found in the Ministry of Industry and Small Business Development Report referred to earlier.

SECTION 11

REFERENCES

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6 BCH, Depositional Environment and Stratigraphic Subdivision, Hat Creek No. 1 Deposit, October 1979.

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9 BCH, Hat Creek Project, Geological Report, No. 1 Deposit, Volumes 1-2, November 1979.

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15 Krupp Industrie-Und Stahlbau Hat Creek Coal Mines Cost Study on Bucketwheel Technology, January 1981.

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