B.C. HYDRO AND POWER AUTHORITY
Mining Department — Thermal Division

HAT CREEK PROJECT

MINING REPORT —
Volume 1 (of 2)

DECEMBER, 1979.

Copy No.
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HAT CREEK PROJECT

MINING REPORT - VOLUME I

DECEMBER 1979

ERRATA

Page 1-1	1st para.	Should be "... 500 MW (net) unit ..."
	2nd para.	Should be "... 2000 MW (net) powerplant."
		Should be "... 4 x 500 MW (net) ..."
Page 2-4	Item (9)	Should read "... equivalent to 6.19 mills/kW.h at a capacity factor of 65%.
Page 3-1	2nd para.	Should be "... create some 875 steady jobs at the mine ..."
Page 3-6	1st para.	Should be "... circumference of around 8 km ..."
Page 4-15	Section 4.6.3.1	Should be "... to be 739 million tonnes ...".
	Section 4.6.3.2	Should be "... 746 million tonnes ..."
Page 4-35	Figure 4-5	Note that expressions for X and Y based on Heating Value in kJ/kg; graph is based on Heating Value of MJ/kg.
Page 5-27	Section (3)	Should read "... reduced from 101 million tonnes to 89 million tonnes."
Page 5-31	Section 4	Add this paragraph:- Maximum gradients used for the mine roads are: (1) Haul roads 8% (2) Service roads - 10%.
Page 5-42	Section 5.5.3.2	In the last sentence of the first paragraph it should read "... not closer than 23 m to the crest ..."

142

Page 5-44	Section (3)	Second and third sentences should read "... 305 million <u>bank</u> m^3 ..." and "... 134 million <u>bank</u> m^3 ..." respectively.
Page 5-45	3rd para.	Last sentence should read "... 29 million <u>bank</u> m^3..."

Excavating Trench 'A'
by Hydraulic Shovel

1977 Bulk Sample Program

HAT CREEK PROJECT

VOLUME 1

TABLE OF CONTENTS Page

Page

4 GEOLOGY

Page

5 MINE PLANNING

Page

VOLUME 2 Page

8 MATERIALS-HANDLING

Page

Page

14 ECONOMICS AND COSTS ESTIMATES

Page

VOLUME 1

LIST OF TABLES

VOLUME 2

No. LIST OF TABLES Page

VOLUME 1

No. | LIST OF FIGURES | Page

VOLUME 2

LIST OF FIGURES

SECTION 1

INTRODUCTION

This mining report is the culmination of five years' work by a task force of B.C. Hydro's Thermal Division and its consultants to develop and to establish the feasibility of a base plan which, by adding a 500 MW unit in each of four successive years, would exploit the rich coal deposits of the Hat Creek Valley for the generation of electricity. A mass of data has been accumulated and analysed, and a point has now been reached when, both on practical and economic grounds, application to the regulatory authorities for necessary licences may be made with confidence.

While many options for the use of the Hat Creek coal deposits have been explored during the past five years, the work in 1979 has concentrated on finalizing the base plan. This has now been achieved. The plan, described in detail in the following sections, deals with the extraction of part of the coal in the No. 1 Deposit by means of hydraulic shovels, trucks, and conveyors, over the 35-year projected lifetime of a 2,000 MW powerplant. The mine mouth powerplant (which consists of 4 x 500 MW units) would be built on the top of the hill above Harry Lake. Any changes to the base plan are likely to be minor and confined mainly to advances in technology.

This report is based upon detailed consultants' reports, and incorporates the results of extensive studies conducted in 1979 by the Mining Department of B.C. Hydro.

In debating whether or not to go ahead with the Hat Creek Project, it may be worth reflecting on how fortunate are the people of British Columbia in possessing what appear to be the world's thickest deposits of thermal coal, located furthermore almost ideally from the point of view of access and mining. Using approximately only half of the proven reserves in the No. 1 Deposit would fuel the powerplant for 35 years, leaving the balance, plus the untouched and much larger No. 2 Deposit, for the benefit of future generations.

The energy crisis having forced a universal re-assessment of coal as an energy resource, coal-owning countries are everywhere engaged in constructing new mines for the purpose of generating power. As an example, at a new coal field in South Africa, four mines have been developed and are supplying fuel to three 3,600 MW powerplants. A fifth mine, developed in less than four years from the planning stage to full production, is now exporting substantial quantities of thermal coal. The power generated from this single field will amount to more than seven times the proposed capacity of the Hat Creek Project. Closer to home, Oregon and Washington are embarked on a 20-year program of constructing no less than eight thermal plants based on coal from a newly developed field in Washington. A lengthening list of new mine construction reflects the re-awakening of interest in coal as a source of power.

It has been adequately proved that an efficient technology exists to mine coal and burn it to produce electricity. This report shows how such technology can be tailored to cope with the complexities of the Hat Creek No. 1 Coal Deposit. Should the project be approved, it would result not only in British Columbia's first major coal-fired powerplant, but be the first step towards developing many possible alternative industrial uses for the coal, and a significant broadening of the base of British Columbia's whole economy.

2 SUMMARY

SECTION 2

SUMMARY

2.1 MINING STUDIES PERFORMED - APRIL 1974 TO DECEMBER 1978

Exploration Drilling

Extensive diamond core-drilling between 1974 and 1978 *identified two deposits, the smaller of which is estimated to contain in* excess of 700 million tonnes. Since 1974, 270 core-holes totalling 75,800 m in length have been drilled. 206 of these holes, on a 150 m by 150 m grid-pattern totalling 54,000 m, have been drilled in the No. 1 Deposit. A further 19,800 m of drilling was completed in pursuit of geotechnical, geohydrological, and other investigation.

The results of these drilling programs, which were conducted under the supervision of Dolmage Campbell and Associates Ltd. and the B.C. Hydro Mining Department, have provided the basis for successive geological interpretations and evaluations of the quality of the coal in the deposit by DCA, CMJV and, most recently, by BCH. Reserves in excess of 700 million tonnes have been established for the No. 1 Deposit. The No. 2 Deposit has been identified as a potentially much larger resource.

Geotechnical and Geohydrological Studies

An assessment and exploration program initiated and assigned to Golder Associates in 1976 has now established a safe overall pit slope angle of 16^o, which can in some areas rise to 25^o, depending on pit wall materials. The same studies have also established waste dump design parameters. A satisfactory level of confidence in data relating to mine design now exists.

A geohydrological program to determine whether pit slope stability can be improved by reducing groundwater pressure has indicated that limited depressurization can be achieved. Geotechnical monitoring will have to continue throughout the life of the mine.

Mining Studies

PD-NCB Consultants, commissioned in 1975 to perform conceptual design studies, recommended that future work should be concentrated on the No. 1 Deposit. The Cominco-Monenco Joint Venture were engaged in 1977 to undertake preliminary engineering design studies. After investigating alternatives, their report submitted in 1978 recommended a design for an open-pit mine to supply 350 million tonnes of coal averaging 17.0 MJ/kg, on a dry basis, over a period of 35 years, requiring the removal of 450 million m^3 of waste. The proposed open-pit would cover an area 3 km by 2.5 km and be 265 m deep, using a shovel-truck-conveyor mining system, with coal-crushing, blending, and stockpiling facilities at the mine mouth. Blended coal would be moved by conveyor to the powerplant 4 km away and 500 m above the valley floor. Waste would move by conveyor to disposal areas at Houth Meadows and Medicine Creek.

The Bulk Sample Program

In 1977, a bulk sample of 6,300 t was excavated from two trenches in the No. 1 Deposit for a burn test. This pilot-scale operation provided valuable data on the mining, handling, and storage of coal and waste materials. Equally valuable was the experience gained in using hydraulic shovels. This proved that the coal can be satisfactorily extracted without blasting, with the exception of a few isolated pockets of rock.

Coal Beneficiation

Bench tests and pilot-scale tests conducted in 1976 established the difficulty of washing Hat Creek coal. Further tests by Simon-Carves on samples from the trenches using modified procedures confirmed and explained the original findings. A pilot-scale test in 1977 involved a 73-t sample. This indicated that coal-washing (beneficiation) was practical, though not justified at present for Hat Creek coal on technical and economic grounds.

2.2 1979 STUDIES

During 1979, the previous mining studies were re-evaluated incorporating all the new data acquired in 1978. Major new studies were conducted in the areas of Coal Quality, Pit Design, Production Scheduling, Materials-handling, and Selective Mining. The results of these studies were integrated with those parts of the previous studies that were unchanged and a revised cost estimate and schedule was prepared.

The final results of this work program are presented in detail in the remainder of this report. Some of the key results are:

(1) 331 million tonnes of coal will be mined over the life of the powerplant, necessitating the removal and disposal of approximately 427 million m^3 of waste;

(2) The powerplant will be supplied with a blended fuel averaging 18.0 MJ/kg, 33.5% ash and 0.51% sulphur on a dry-coal basis, with a moisture content of 23.5%. This fuel will be supplied within a tolerance of ±1 MJ/kg on heating value;

(3) The improved coal quality results from the use of hydraulic shovels applying selective mining techniques;

(4) The pit has been redesigned and the production rescheduled, which has resulted in a major reduction in pre-production stripping from 20 million m^3 to under 7 million m^3;

(5) The Materials-handling System has been substantially redesigned and conveyor belt widths generally have been increased from 1,200 mm to 1,400 mm;

(6) Peak manpower levels have been reduced from 1,005 to 875;

(7) The coal quality characteristics have been evaluated by a specialist consultant and a boiler fuel specification produced;

(8) Summary of estimated mine costs (October 1979 Canadian dollars)

1. Capital cost to full production in Year 4 (costs to end of Year 3)	$248 million;
2. Pre-production operating costs to start of commercial production in Year 1	$55 million;

3. Additional capital costs during project life (primarily for equip ment replacement) $290 million;

4. Operating costs per tonne of coal during full production range from $4.71 to $5.81;

(9) Levellized fuel costs over the project life, uninflated and discounted at 3%, are $0.567/GJ ($7.80 per tonne of coal), excluding the cost of power consumed in the mining operation. This is equivalent to 6.19 mills/kW.h. Power costs are $0.47 per tonne based on 20-mill power.

2.3 CONSULTANTS EMPLOYED

The following consulting firms have performed assignments related to the Hat Creek Mining Studies:

(1) Geological Exploration 1974-1978

Dolmage Campbell and Associates (DCA)

(2) Mine Conceptual Design 1976-1977

Powell Duffryn-National Coal Board (PD-NCB) in association with Wright Engineers Limited and Golder Associates

(3) Geotechnics and Hydrology 1977-1978

Golder Associates

(4) Mine Feasibility Studies 1977-1978

Cominco-Monenco Joint Venture (CMJV) with sub-consultants: North American Mining Consultants Inc. (NAMCO); Simon-Carves of Canada Ltd.; MBB Mechanical Services

(5) Materials-handling and Low-grade Coal Beneficiation 1979

Simon-Carves of Canada Ltd.

(6) Coal Fuel Specification 1979

Paul Weir Company (WEIRCO)

(7) Geostatistics 1978-1979

Mineral Exploration Research Institute (IREM-MERI)

(8) Coal Deposit Computer Modelling 1978-1979

Mintec Inc.

3 A PROJECT DESCRIPTION

No.	LIST OF FIGURES	Page

SECTION 3

A PROJECT DESCRIPTION

3.1 THE PLAN

The Upper Hat Creek Valley of South Central British Columbia contains the thickest deposits of thermal coal so far discovered in the world. An estimate suggests that up to 15 billion tonnes could exist in the area, although only two deposits have so far been identified which permit coal to be extracted by open-pit mining. Of these, the No. 1 Deposit is estimated to contain over 700 million tonnes, the No. 2 Deposit over a billion tonnes.

The Hat Creek Project is a plan to extract some of the coal from the smaller No. 1 Deposit and to burn it for the purpose of generating electricity. This would create some 1,000 steady jobs at the mine, apart from 3,000 temporary construction jobs. Should the project be approved and licensed, it would broaden the traditional base of hydro-power generation in British Columbia by starting to use coal, a major alternative resource.

3.2 A HISTORY OF THE PROJECT

The existence of coal deposits in the Hat Creek Valley has been known for over a century, having first been reported in 1877 by G.M. Dawson, of the Geological Survey of Canada. Since then, various private titleholders have made sporadic attempts to mine the coal and to sell it. They all failed for lack of markets and the ability to operate on a sufficiently economic scale. More recently, substantial coal reserves were identified in 1944. In 1957, a subsidiary of the B.C. Electric Co. (a predecessor of B.C. Hydro) began a systematic exploration of the deposits. These explorations have continued on an expanding scale, culminating in a feasibility study which concluded that the project would be both technically practical and economically desirable. B.C. Hydro established a Thermal Division in 1974. Its engineers have written this report.

3.3 THE LOCATION OF MINE AND POWERPLANT

While the mine is to be located in the Upper Hat Creek Valley, about 200 km North-East of Vancouver, the location of the powerplant is optional. Eight alternatives were considered. The site chosen was largely dictated by environmental imperatives - a hillside, about 4 km from the open pit and at an elevation about 500 m higher than the valley floor. Good dispersal of emissions is ensured by a chimney rising an additional 244 m.

3.4 THE RESOURCE: COAL

The quality of coal in the No. 1 Deposit appears to vary over an unusually wide range, from less than 9.0 MJ/kg to 23.0 MJ/kg. The overall average is 17.7 MJ/kg, approximately one-half that of high quality bituminous coal found in the East Kootenays, the Rocky Mountain Belt of B.C., and in the Eastern United States. Variations in the quality of Hat Creek coal, added to the high moisture and ash content, are problems that have been provided for in the design of the power-plant. It has also been taken into account in studies leading to the choice of the mining method: a process of selective mining and blending, which will ensure production of a fuel averaging 18.0 MJ/kg, 33.5% ash and 0.51% sulphur on a dry-coal basis, with a moisture content of 23.5%. Geologically, 16 sub-zones have been identified in the Hat Creek Coal Formation. Two of these sub-zones are largely composed of waste, with the other 14 consisting of coal of varying quality.

3.5 THE PROJECT

Design studies have defined the major constraints and requirements of the project which features:

(1) A large open-pit mine, with adjoining waste disposal areas, at the North end of the Hat Creek Valley;

(2) A powerplant containing four coal-fired boilers, operating steam-driven turbine generators, located on high ground some 4 km East of the open pit;

(3) A combination of hydraulic shovels, trucks, and belt conveyors, to mine and move both coal and waste;

(4) A diversion of Hat Creek and Finney Creek around the open pit with the necessary headworks, spillways, canals, etc.;

(5) A cooling water reservoir supplied by a 21 km buried pipeline from a pumphouse on the Thompson River;

(6) Two large waste disposal areas, which would gradually be covered with topsoil and landscaped.

3.6 THE MINING METHOD

A plan has been drawn up whereby part of the No. 1 Deposit would supply coal for operating a 2,000 MW powerplant over a 35-year lifespan. The coal would be mined from an open pit developed to a depth of 235 m below the valley floor. There is enough coal above this elevation to meet the planned requirements of the powerplant. In Year 35, the open pit would measure approximately 4 km by 2½ km, with a circumference of around 16 km. The surface area of the hole would measure around 598 ha.

Berms (benches) about 18 m wide would step down to the pit bottom, with overall slopes at an angle varying from 16^{o} to 25^{o}, based on geotechnical calculations. It is proposed to remove 331 million tonnes of coal over 35 years, together with 427 million m^3 of waste materials, some of which would be stockpiled for construction needs.

A ramp would be cut towards the heart of the No. 1 Deposit for the main conveyors installed to transport coal and waste up to the surface. Some of the topsoil would be stockpiled for use during reclamation. Both coal and waste rock would be mined by using large hydraulic shovels and trucks. The trucks would haul both coal and waste to loading pockets at the conveyor where, after brushing to -200 mm, the material will be transported to the top of the ramp for subsequent distribution along another system of conveyors.

The coal would be mined according to a plan designed to provide a mixture of the right quality. Coal of poorer quality would be moved by conveyors to a dry beneficiation plant, where some of the impurities would be removed by a crushing and screening process which would raise the heating value to an acceptable level. Coal not requiring beneficiation would move direct to a coal preparation area, where it would be screened, crushed to -50 mm, and conveyed to the Coal Blending Area. Here slewing stackers using the Windrow Method would build up stockpiles of blended fuel ready to be reclaimed and transported by an overland conveyor to the powerplant.

The waste material would be moved by conveyors to either of two waste dumps, the larger in Houth Meadows, the smaller in the Medicine Creek area. Both dumps were chosen because their location, though conveniently adjacent to the open pit, would not interfere with future mining. Houth Meadows is expected to take all the waste excavated during the first 15 years, with both dumps being used from Year 16 on. Medicine Creek will also be used to dump the anticipated 10,000 t/d of both fly-ash from the electrostatic precipitators and bottom-ash from the furnace bottoms. Both waste rock and ash would be spread in the dumps by stackers, and all dump surfaces would ultimately be levelled, contoured, and landscaped when the mine closes.

3.7 THE POWERPLANT

The powerplant, with four 500 MW (net) units, would be located on high ground near Harry Lake, some 4 km East of the open pit. The ground level of the powerplant is 1,410 m above sea level, which is about 500 m higher than the ground level at the surface at Open Pit No. 1.

Each water tube boiler would be about 95 m high, with furnace dimension about 18 m square, followed by numerous surfaces containing steam and/or water, to which hot gases leaving the furnace transfer heat. At full load each boiler would consume about 407 t/h of typical Hat Creek coal to produce 1,750 t/h of high-pressure steam.

Electricity would be generated in the powerplant by four steam turbine-generators, each capable of generating 560 MW (gross) for a total net capacity of 2,000 MW.

At the turbine exhaust, a condenser condenses the steam to water after it has done its work. The water is then returned to the boiler to be converted into steam, which is a closed cycle. A condenser does, however, require large quantities of cold water flowing through it to condense the exhaust steam. In providing cooling for the condenser, the cooling water itself warms up, and the heat it has gained must be dissipated. As it would be harmful to the environment to discharge this heated water into the natural water system, the Hat Creek method of cooling provides for two cooling towers, each rising to a height of 135 m. The heated water leaving the condensers is piped into the cooling towers, where it is allowed to cascade down to the bottom, passing in droplet form over a latticework. Air flowing upwards through the tower is heated as the water is cooled, most of the heat transfer being latent heat from the portion of the water which evaporates. Make-up water must be added to replace this evaporative loss to the atmosphere. This is pumped from the plant reservoir, containing roughly a two-month supply. The reservoir is replenished from the Thompson River through a 21 km buried water pipeline.

3.8 POWERPLANT EMISSIONS AND WASTE

A vital factor in the powerplant design and operation is an acceptable environmental impact. Both air and water quality control systems have therefore been incorporated into the design.

Air quality measures include location of the plant high above the valley, a 244 m high multiflue stack, and cold electrostatic precipitators, capable of trapping 99.55% of the particulates. Space has been left for possibly adding, later, flue gas desulphurization. Hat Creek performance coal contains only 0.51% sulphur on a dry basis. When abnormal atmospheric conditions are predicted which may cause ambient SO_2 levels to increase, a MCS (Meteorological Control System) will be applied. This will involve either switching to low-sulphur coal or reducing the load. Oxides of nitrogen emissions will be controlled by appropriate design and operation of the boilers.

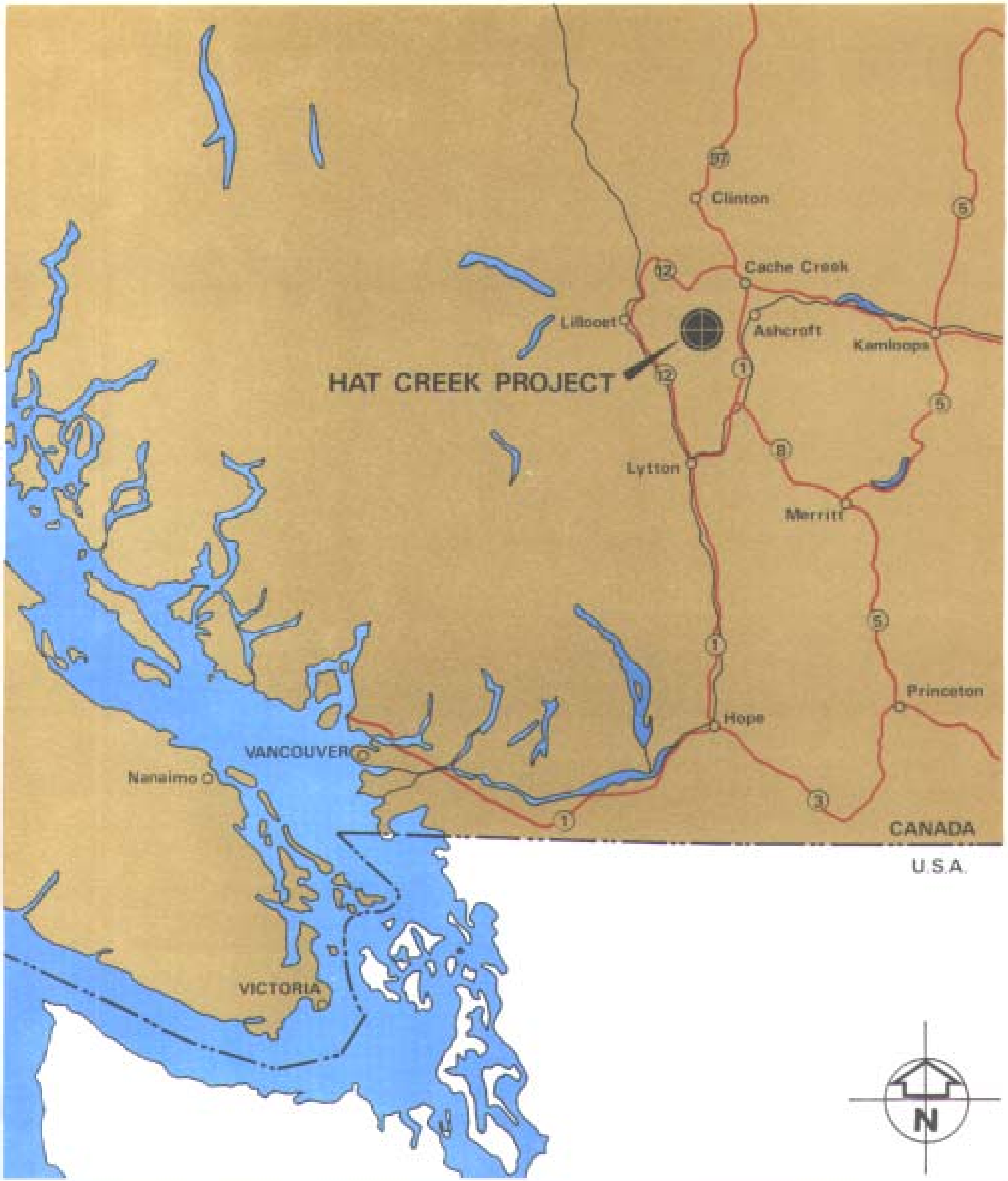
10,000 t of ash will be produced daily. Fly-ash will be wetted and conveyed, with bottom-ash, to a "dry" disposal area in the Medicine Creek Valley below the dam for the plant reservoir. Bottom-ash will be continuously removed from below the boilers. The ash disposal area would be progressively covered with topsoil and landscaped over the lifetime of the project.

3.9 TWO KEY AREAS: DRAINAGE AND RECLAMATION

Both drainage and reclamation of disturbed land are related and inter-acting. With several difficult landslide areas along the West side, a comprehensive mine drainage plan is a pre-condition of successful mine development. The drainage plan developed for Hat Creek is designed to meet the difficult ground conditions revealed by exploration. It includes an inter-locking system of diversions, dikes, ditches, de-watering wells, and the provision of lagoons to trap sediments and leachates. Prior to construction, an area of ponds and lakes would be drained of water which might mobilize the already unstable ground in the slide areas.

In terms of environmental protection, the drainage plan ensures that water-borne contaminants will be trapped and disposed of; only water purified to an acceptable degree will be allowed to re-enter the natural water courses. Flows will also be handled in such a way as to re-establish wetland habitats for wildlife.

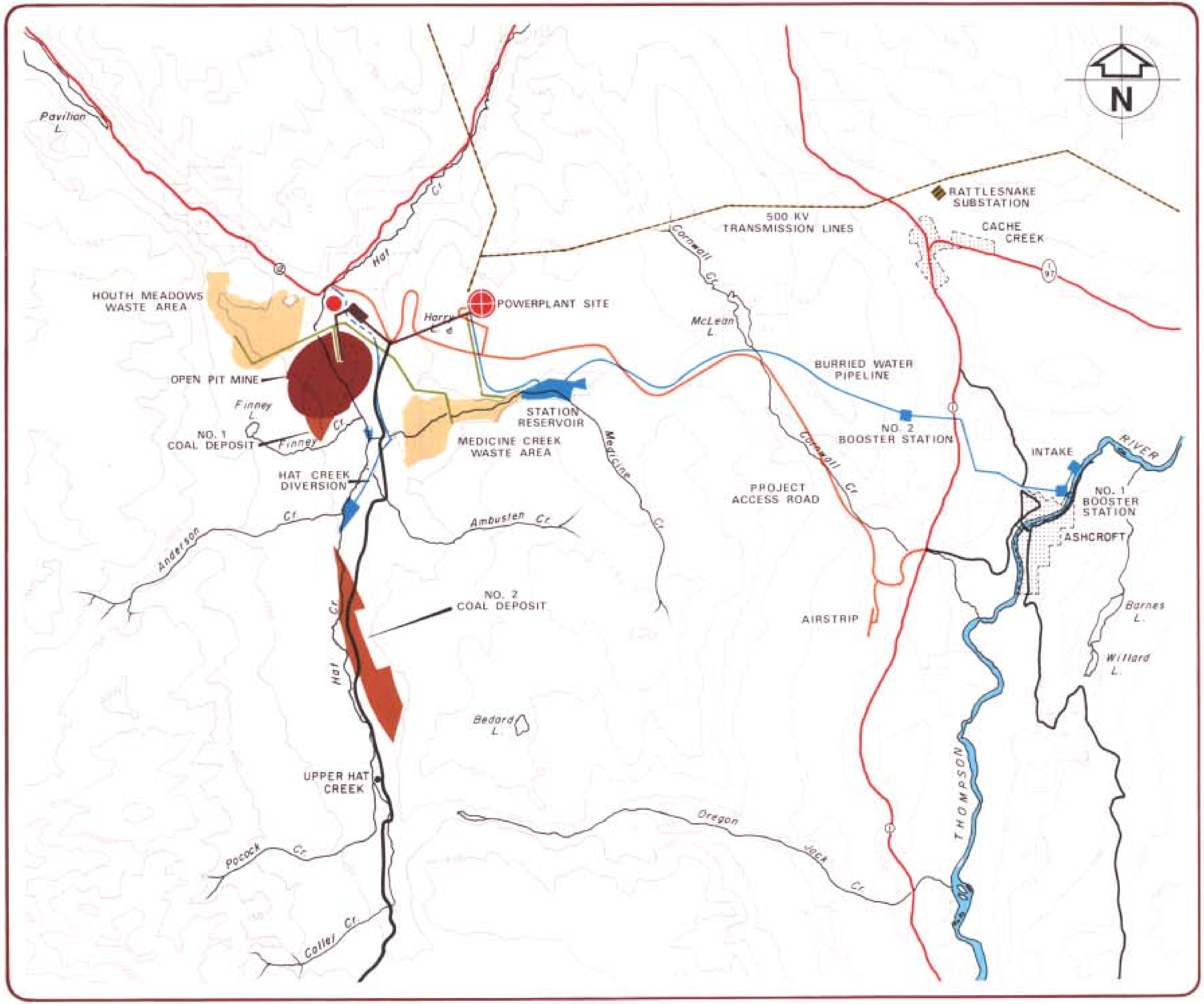
The guiding rule governing land reclamation would be to reclaim progressively those areas which permit restoration concurrently with operation of the mine (e.g. the ash dump in Medicine Creek), and to budget for extensive reclamation once the mine closes. 96% of the land disturbed during the lifetime of the mine (except the open pit) will be levelled, contoured, covered with topsoil, and seeded or re-planted with shrubs and trees, the objective being to restore it as closely as possible to its former condition. Most of the remaining 4% would be accounted for in the need to retain access roads, reservoirs, drainage ditches and the like for the purpose of continued monitoring of water quality, etc. It is estimated that this reclamation program will cost $40 million over the lifetime of the mine.



HAT CREEK PROJECT

FIGURE 3-1

PROJECT LOCATION



HAT CREEK PROJECT

FIGURE 3.2
PROJECT COMPLEX MAP

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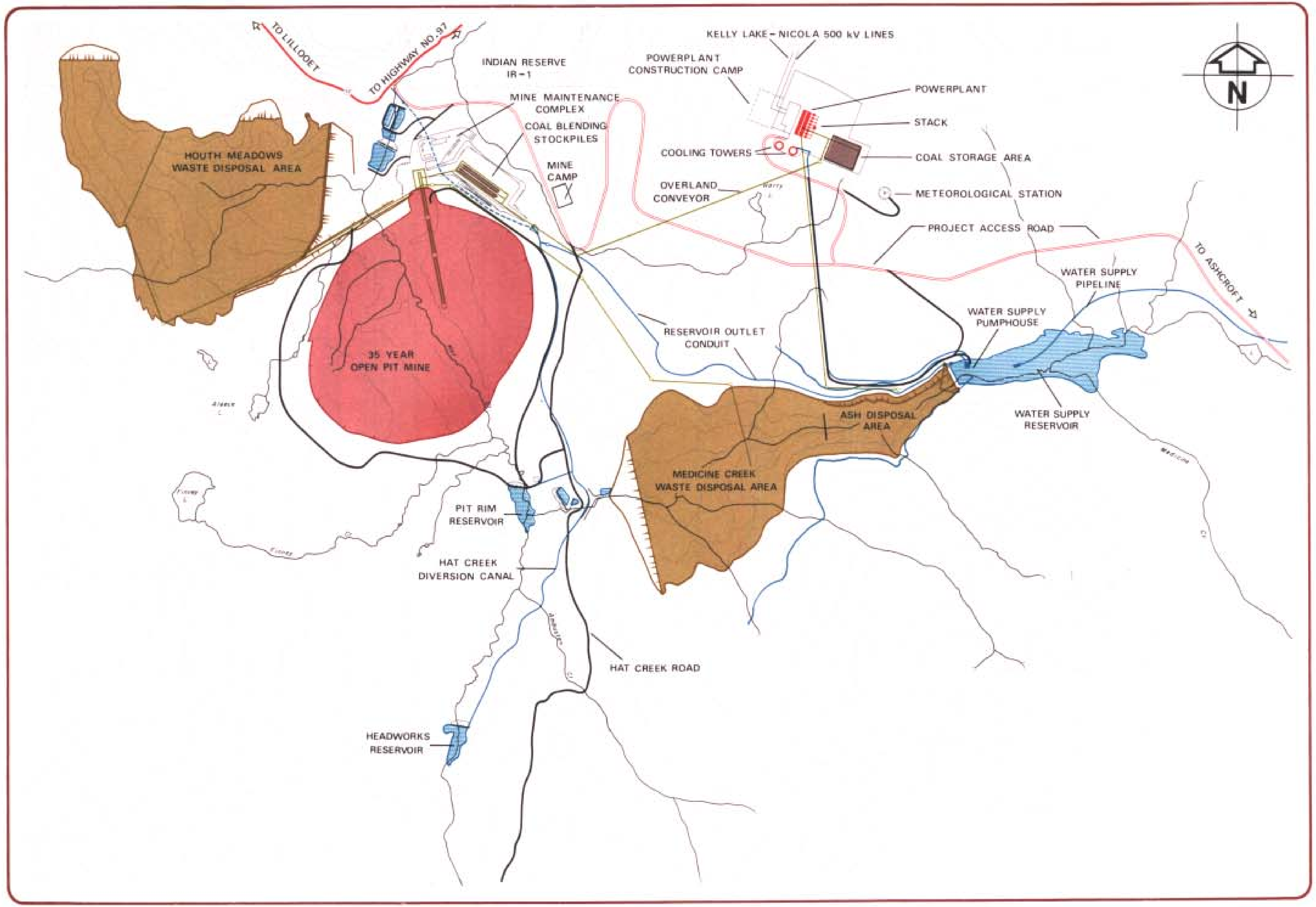
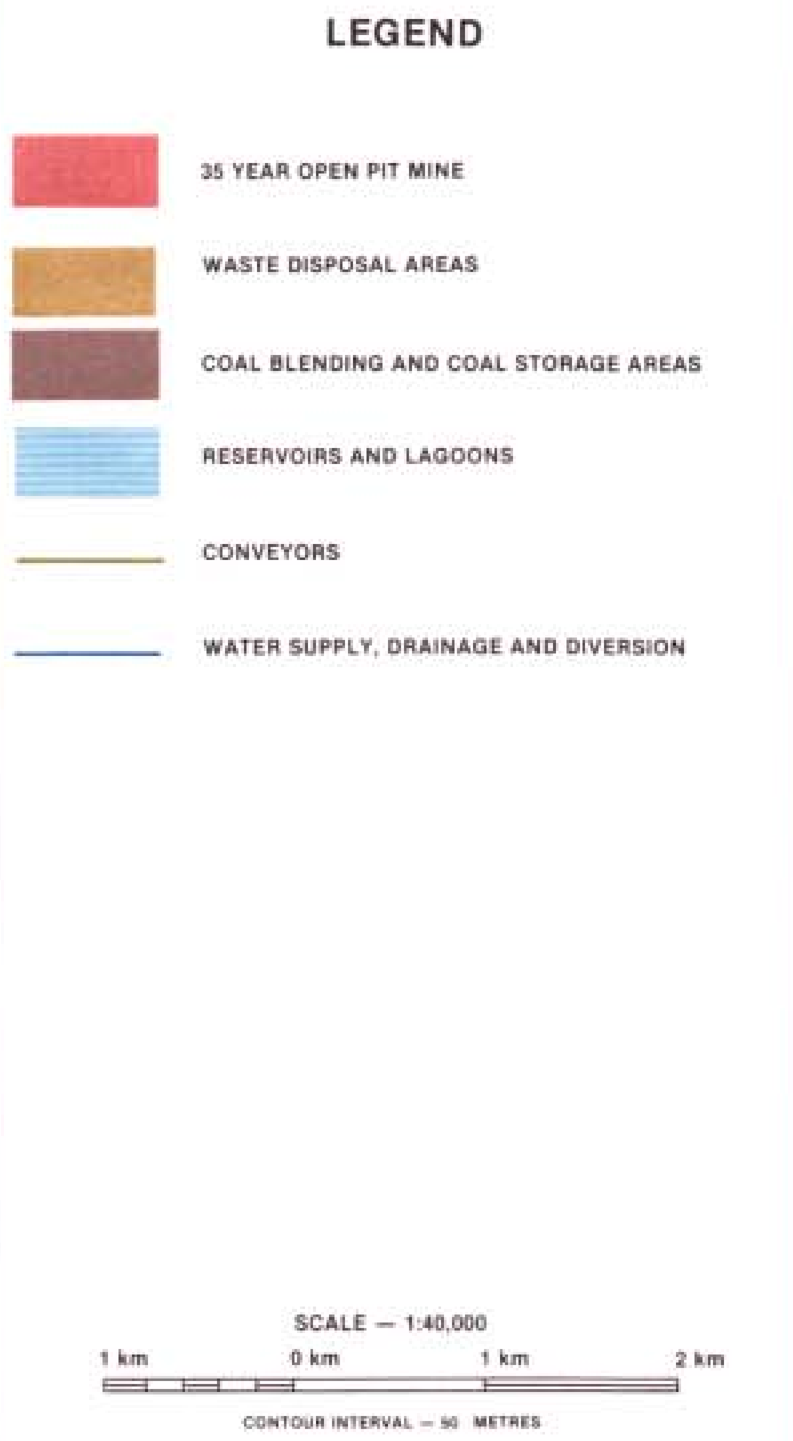
SOURCE: British Columbia Hydro and Power Authority (3)





HAT CREEK PROJECT

FIGURE 3.3

DETAILED SITE LAYOUT MAP

00192 1/2 2

SOURCE: British Columbia Hydro and Power Authority (4)

HAT CREEK PROJECT

FIGURE 3-4

PROJECT OVERVIEW FROM SOUTH EAST

SOURCE: Toby, Russell, Buckwell and Partners Architects (2)

00142 1/2
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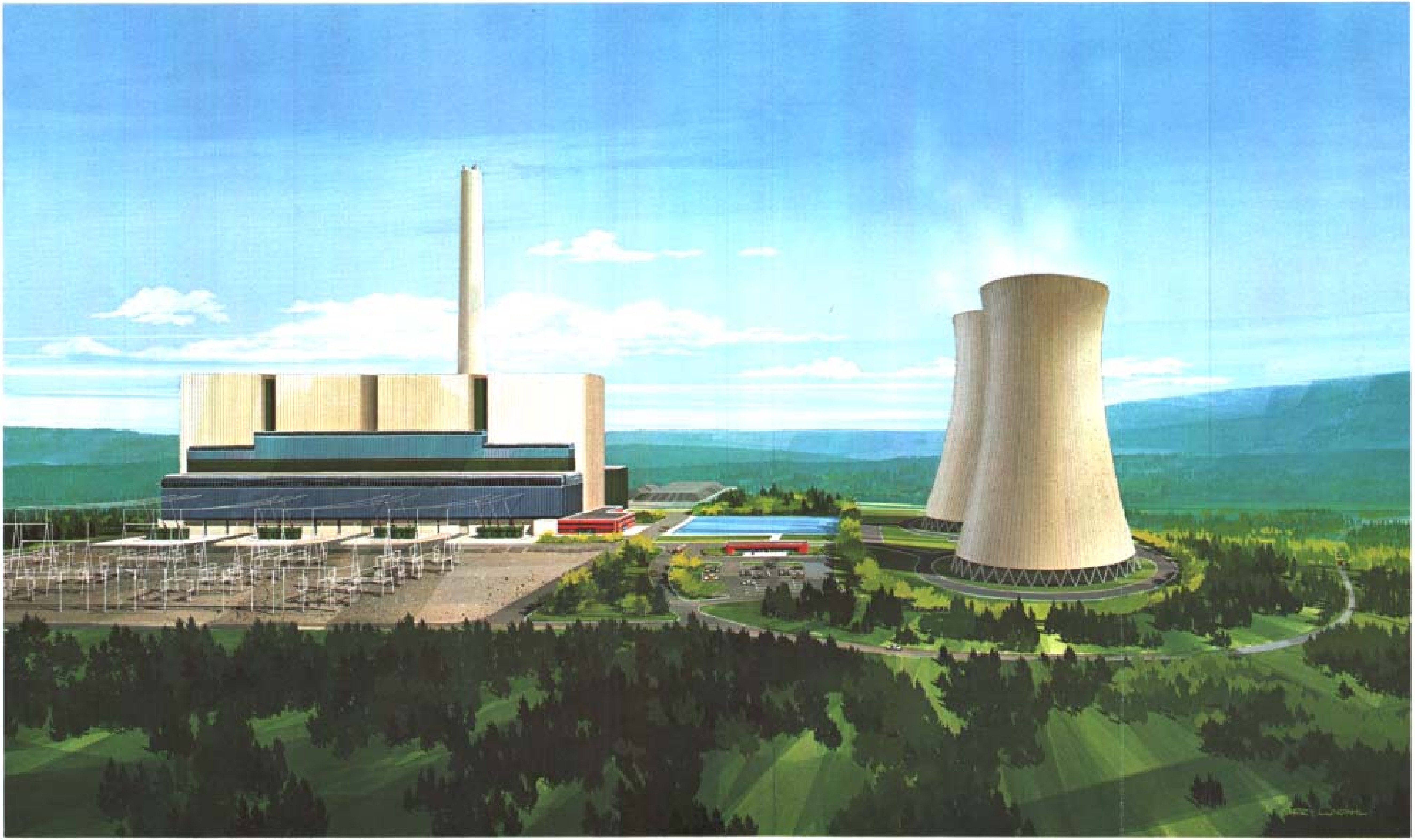


HAT CREEK PROJECT

FIGURE 3-5

POWERPLANT FROM WEST

SOURCE: Toby, Russell, Buckwell and Partners Architects (2)

00142 1/2
4

4 GEOLOGY

No.	LIST OF TABLES	Page

LIST OF FIGURES

LIST OF FIGURES (Continued) Page

SECTION 4

GEOLOGY

4.1 INTRODUCTION

This report summarizes all the geological, geophysical, and coal quality data for the No. 1 Deposit, based on a 152.4 m grid. Statistical studies for the various parameters show a high level of confidence, from which it is concluded that the geological data are adequate for mine planning.

To determine chemical properties of the coal deposits, proximate, ultimate, and ash analyses were made on the core samples at Commercial Testing Laboratories and General Testing in Vancouver, and Loring Laboratories in Calgary. In order to improve technical control and expedite analytical work, a field laboratory was set up for the 1977/78 exploration program to handle routine proximate analysis, thermal value determination, sulphur, and screen analysis. All sampling and analytical procedures followed American Society for Testing and Materials' (ASTM) standards.

Samples were also provided for washability studies at the laboratories of Energy, Mines and Resources in Edmonton, Birtley Engineering (Canada) Limited, and Warnock Hersey Professional Services Ltd., in Calgary. Warnock Hersey also conducted wet attrition tests to simulate size degradation in a washing plant and wet screen analyses of the low-grade coal for any possible beneficiation.

4.2 EXPLORATION AND DEVELOPMENT

In the earlier stages of exploration, between 1925 and 1959, 22 diamond-drill holes aggregating 4,375.8 m were drilled. This indicated the potential of a large low-rank coal deposit in the Hat Creek Basin.

In 1974, B.C. Hydro initiated a detailed exploration program to define the limits, structure, and coal quality of the Hat Creek Basin.

Golder & Associates were retained as consultants for the geotechnical studies, including slope stability, foundation, and geohydrological investigation which formed an integral part of the overall program.

Till 1977, the geological drilling and exploration program was conducted by Dolmage Campbell and Associates. Subsequently, B.C. Hydro took over the responsibility of running the program. In the total program, 206 exploration holes (54,037 m), 151 geotechnical holes (7,996.7 m), and 117 holes (2,117.7 m) for surficial material investigation, bulk sampling, and other studies, aggregating 73,860.3 m, were drilled. (Table 4-1)

Under the same program, 64 holes (21,800 m) were drilled in the No. 2 Deposit South of the No. 1 Deposit. Though the reserves indicated were larger than those of the No. 1 Deposit, the mining and economic conditions were not as favourable. Consequently, no further drilling was considered at this time.

Regional surface geophysical surveys, especially gravity and magnetometer, have helped in delimiting the coal deposit and identifying the distribution of the denser materials - i.e. burnt zone rocks and volcanic rocks.

Aerial photographic surveys were carried out to provide topographic maps and control for exploration work. Elevation control was established by running third-order levels from the geodetic bench mark near the junction of Highways 12 and 97. Additional survey bench marks established by McElhanney in 1976 served as ground control in the area.

After drilling was completed, the drill sites were cleaned, levelled, and restored to the natural ground contours before seeding with a mixture developed for use in the Hat Creek region.

4.2.1 Geophysical and Geological Logging

All holes were geophysically logged (Gamma Ray and Density) on a scale of 1:250. Geolographs provide data on the rate of penetration versus bit pressure and bit rpm versus pump pressure. Gamma ray and density log peaks were used to identify marker horizons and varying lithologies throughout most of the deposit, thus providing a means of sub-zone correlation between drill holes. Gamma ray log peaks essentially reflect claystone interbeds (partings) with relatively high radioactive K-ion content. The corresponding density log reflects the variation in density of the rock and coal or coaly material.

Five ranges of the API (American Petroleum Institute) values were established to represent coals of varying ash content and waste bands. These were plotted on cross sections to aid in the interpretation of the lithofacies distribution and structure of the deposit. Correlation of the data led to the concept of 16 sub-zones within the four major zones recognized earlier.

Cores obtained from drilling have been geologically logged; the lithological and structural characteristics, mineralization, etc., have also been recorded. All the cores have been indexed and preserved in core sheds at the site.

4.2.2 Coal Sampling

Systematic analytical work was conducted, applying a 6 m maximum sampling interval for proximates, thermal values, and sulphur determination; and 12 m - 18 m maximum for mineral-ash analyses, fusibility, grindability, and other tests. As a rule, the sampling intervals were required to correspond to the natural boundary of the homogeneous coal as reflected by the geophysical logs. The cores were split in half along their length and bagged for chemical tests. The other half were stored for future reference. Since 1977, all the samples were run at B.C. Hydro's laboratory located at the site. Check samples were regularly sent out to commercial laboratories.

4.3 GEOLOGY

4.3.1 Stratigraphy

The tertiary sediments in the Upper Hat Creek Valley were deposited in a Northerly-trending topographic depression in the South-West part of the Intermontane Belt of the Canadian Cordillera. The mountains bordering the valley range in age from Permian to Cretaceous. The valley floor is underlain by tills and glacio-fluvial deposits subsequent to the Pleistocene glaciation. Table 4-2 summarizes the general stratigraphy of the region.

The coal-bearing section belongs to the Hat Creek Formation of the Eocene Epoch deposited 36 to 42 million years ago. It is underlain by the Coldwater Formation consisting of detrital sediments and overlain by poorly consolidated bentonitic claystone and siltstone beds of the Medicine Creek Formation. These beds were subjected to glaciation and subsequently overlain by glacio-fluvial material.

Based on lithology, coal quality, and geophysics, the Hat Creek Coal Formation was sub-divided into 16 sub-zones. Two of these sub-zones, A-6 and C-1, are essentially waste and coaly shale units, while the remaining 14 represent coal of varying quality. Table 4-3 illustrates a scheme for the development of the stratigraphic sub-division of the Hat Creek Coal Formation.

A typical sub-division is illustrated in Figure 4-1.

4.3.2 Structure

The regional structure of the Hat Creek Coal Basin is a North-trending graben flanked on both sides by gravity faults. Transverse faults have offset the graben in places.

The primary structure in the No. 1 Deposit consists of two synclines separated by an anticline, plunging at an average of 15^o to 17^o towards the South-South-West. It is truncated on the South and East by steeply dipping boundary faults (Figure 4-2). Repetition of stratigraphic sections has been observed in some of the drill cores. Such overturning is due to local reverse-faulting, which is probably also responsible for the anomalous thickness of detrital sediments encountered in the Western sector. These compressive forces do not appear to be strong enough to cause a major regional uplift. Undoubtedly, the general facies change in this direction has significantly contributed to the thickening of the waste material zone (Figures 4-3, 4-4A, and 4-4B).

4.3.3 Burn Zone

The "Burn Zone" is characterized by pink to yellowish-brown coloration on North and South walls of Trench A, in outcrops North of Trench A, and in several of the cores. The red coloration is due to the formation of iron-oxide by baking of ferrous oxide and hydroxide of the clay. The well-preserved structure of the original sediments and the vesicular nature of the burnt material suggest the effect of burning of the coal. The interlayered and enclosing claystones were baked in this process. The coals were ignited by spontaneous combustion or forest fires, though the volcanic activity in the adjoining area could also have been partially responsible.

4.3.4 Trench Geology

In 1977, three trenches were excavated in the Northern half of the deposit. Of the three trenches, Trench A and B were excavated to provide bulk coal samples for testing burning characteristics at the Battle River Powerplant in Alberta. Trench C provided information on the stability of the claystone highwall.

Trench A: This exposed B-zone coal and the contact with underlying Cl claystone at its West end and the collapsed burn zone material in contact with coals at the East end.

Bedding plane shearing, contorted folds, and faults have been observed. Large sections of petrified wood with pyritic inclusions were observed at the contact of coal and Cl claystone.

Trench B: This exposed the D-zone coal, representing the earliest phase of thick coal deposition. It was marked by an abundance of petrified wood up to 12-15 m long. The coal was hard, compact, and massive, with a thin film of siderite and a cluster of very fine pyrite crystals along the fracture planes.

Trench C: Trench C excavation showed the sliding of the older Coldwater Formation over the younger glacial till. The failure of some of the faces indicates material weakness due to water seepage and swelling of the bentonitic claystone.

4.4 DEPOSITIONAL ENVIRONMENT OF THE HAT CREEK COAL FORMATION

Some 40 million years ago, peat deposition began in a generally broad North-trending marsh with little or no circulating water. The favourable climatic conditions, aided by the slowly sinking basin throughout the period of D-zone deposition, accounts for the immense thickness of the virtually uninterrupted coal mass. When the equilibrium was disturbed by rapid sinking, the basin was cyclically flooded by fresh water, leading to the deposition of numerous partings in the coal measures following D-zone deposition. The Western and the South-Western margins of the peat basin received fluctuating amounts of coarse sediment, resulting in rapid lithofacies change from coal to coarse sandstone, particularly in rock member sub-zones A-6 and C-1 which thicken significantly towards the South and West. In the centre and North-East of the peat basin, the rates of subsidence and deposition were about equal, and the effect of the silty sediment from the Western stream was minimal, allowing the continued accumulation of plant debris to proceed uninterrupted.

The Interior Plateau region was affected by volcanic activity contemporaneous with the peat deposition. The widespread occurrence of ash beds in the coal measures reflects these episodes of volcanic eruptions.

4.5 COAL QUALITY

Systematic analytical work was conducted on all cores, applying a 6 m maximum sampling interval for proximates, heating values, and sulphur, and a 12 to 18 m maximum for mineral-ash analysis, fusibility, grindability, and other tests.

In 1977, 7,000 t of sample coal was transported to Battle River Powerplant in Alberta for technological evaluation of its burning characteristics. This program demonstrated that a typical Hat Creek coal can be handled, pulverized, and burned in a commercial powerplant.

Washability tests were performed on the above sample. Earlier studies on bulk-auger samples had indicated an imbalance in size consist due to excessive size degradation in the washing process affecting the actual recovery values. Subsequent wet attrition tests, at Warnock Hersey in Calgary, explained this anomaly.

4.5.1 Ash and Heating Value

The dry-ash vs. heating value MAF (Moisture Ash Free) regression analysis of the three holes, DH 135, 136, and 274, in the central part of the basin indicates a linear relationship for samples from the A, B, and C-zones with less than 60% ash (db). The plot for D-zone from the same holes shows an almost identical trend. This is indicative that the coals from various zones have the same rank. To establish a practical ash vs. heating value (db) regression line (Figure 4-5), and the analytical values (Table 4-4) for all the coals within the deposit with the exclusion of those below the cut-off grade, were included in the regression analysis.

The ultimate analysis is required to calculate the net heating value of the coal and to establish the emission levels of oxides of sulphur and nitrogen.

The average values for Hat Creek coal are:

C = 46.2

H = 3.6

O = 15.4

N = 0.9

Cl = 0.03

S = 0.51

4.5.2 Moisture Determination

One of the most critical parameters in coal analysis is the determination of "in-situ" moisture.

In the exploration stage, where heavy reliance is imposed on drill cores, it is not possible to get cores in their natural state because of the drilling-water contamination. To improve this situation, "equilibrium moisture" as per ASTM (1412-56) was determined. This tended to be higher than true "in-situ" moisture, as coal in nature is more compact and not always saturated to the optimum level that the ASTM calls for.

Tests run from 1957 to 1976 produced an average equilibrium moisture of 24.2%.

The 1978 5A Drilling Program incorporated a careful moisture analysis program. The sampling procedure involved the following steps:

(1) Taking 10 cm samples every 15 m in coal;

(2) Taking the sample immediately after it came out of the core barrel;

(3) Wiping the surface moisture off with a rag;

(4) Sealing the sample in plastic wrap and tape;

(5) Resealing the sample in a plastic tube with the air squeezed out and the end heat-sealed.

The results for 121 samples showed an average total in-situ moisture of 21.86% (with a standard deviation of 4.14% and a standard error of the mean of 0.38%), average ash (db) 28.18%.

Moisture in coal is present in two forms: surface and bonded. The surface (or air-dried) moisture is readily lost when exposed to the atmosphere. The mean value obtained for 2,600 samples tested for air-dried moisture was 12.97% with a standard deviation of 5.73% and a standard error of the mean of 0.11%.

The residual or bonded moisture is determined by heating an air-dried sample for an hour at 110°C. Normally the coal will re-absorb this moisture when exposed to the atmosphere. The mean value of over 4,000 residual moisture tests was 9.06% with a standard deviation of 4.75% and a standard error of the mean of 0.07%.

Studies conducted by the Paul Weir Company have predicted a mean total moisture content for run-of-mine coal of 23.5%.

4.5.3 Sulphur Distribution

Initial studies on sulphur distribution in the No. 1 Deposit showed an average value of 0.51%, of which approximately 71% was organic, 25% pyritic and 4% sulphate.

Table 4-5 shows the distribution of the forms of sulphur by zone and for the whole deposit.

Recent studies indicate that the distribution is not as erratic as was thought earlier. In many sections within the sub-zones, continuity in sulphur distribution is observed. There are distinct bands in the sub-zones that contain a high sulphur concentration. High sulphur concentration has been identified in the top 3 m of A1 sub-zone coal and at the bottom of the B2 sub-zone. The identification of such sections will have a direct impact in controlling the sulphur content of the run-of-mine coal.

Some of the other broad conclusions are:

(1) The Western sector of the deposit shows higher sulphur than the Eastern sector;

(2) A-zone contains the highest average total sulphur, while B-zone contains the highest local concentrations.

Sulphur is discussed further in Section 4.7.2 on Geostatistics.

4.5.4 Mineral Analysis of Ash, Ash Fusibility and Grindability

The major constituents of the coal-ash average 52.6% SiO_2 and 28.3% Al_2O_3 may be of interest for alumina extration. The analyses of ash from the four zones show no appreciable difference, indicating the source material for the ash remained unchanged throughout the coal deposition.

The ash deformation temperature is indicative of its physical behaviour at combustion temperatures. The range from initial deformation to fluid temperature suggests the fouling conditions of the boiler.

The average initial deformation temperature, taken over the entire deposit, is in excess of 1,400°C, the limit of most of the laboratory furnaces.

The Hardgrove Grindability Index for D-zone is lower than the A, B, and C-zone coal. The normal range of HG Index falls between 38 and 50.

4.5.5 Specific Gravity

The specific gravity of coal was determined on small pieces of coal cores by the water displacement method after the sample had been fully saturated with water. As there was no significant difference between the specific gravities of coal from different zones for a given ash value, one common regression curve was developed:

$$\text{Specific Gravity} = 1.21104 + 0.00738 \times \text{Ash\%}$$

The average of 1,584 waste samples gave a specific gravity of 1.93. For calculation purposes a specific gravity of 2.00 was considered as more conservative.

The burn zone material averaged 2.16.

These values were used in reserve estimation.

4.6 COAL RESERVES

4.6.1 Introduction

The coal reserves for the Hat Creek No. 1 Deposit were calculated using a computer model. The selection of the modelling technique was controlled by the necessity to accurately reflect the complex structure, and to handle the variability of the coal density and quality. Other important criteria were: the ability to produce adequate displays for verifying and using the model; the ease of making changes for the addition of new data or for correcting errors; and the flexibility to adapt to changing requirements.

The technique selected was to construct a cross-sectional model using the Variable Block Model (VBM) method developed by Mintec Inc. Using this method makes it possible to produce a model that accurately duplicates the geologist's interpretation on each section with assigned quality values for each block.

4.6.2 Development of the Variable Block Model

4.6.2.1 Developing Reserve Blocks

The geological zones and structural features were digitized from cross-sections using an electronic digitizer. Cross-sections were then plotted by the computer on the same scale as the originals for checking.

On each cross-section the sub-zones were sub-divided by faults and further sub-divided equally into smaller blocks less than 200 m in horizontal length.

The top and bottom surfaces of each block coincide with the sub-zone boundaries, which produces a block of variable thickness conforming to the geological interpretation. Each block is projected halfway to the adjoining cross-sections: 76.2 m North and South.

When the block definition process is completed the data is stored in the "Geometry File".

4.6.2.2 Quality Assignment to Blocks

Composite sample values were calculated for each sub-zone in each drill hole. The individual samples were weighted by their length and specific gravity. The composite values were computed in two different ways. The first method combines all the samples, both coal and waste, for a given sub-zone and drill hole, which effectively assigns the whole intersection to either coal or waste at a given cut-off grade. This method represents non-selective mining. In the second method, the coal and waste samples were accumulated separately provided that they formed part of a band greater than 2 m in thickness, which reflects selective mining capability. Bands less than 2 m thick were combined with the adjacent samples. The split between coal and waste was defined by an assigned cut-off grade. Using the second method generated additional data for storage: coal thickness, waste thickness and the number of coal/waste contacts.

Quality values were calculated for each block using the inverse square of the distance method applied to the distance between the block centre and the mid-point of the composite sample used. The search distance used was 175 m North-South and 500 m East-West. If the closest composite contained no coal, then none was assumed to exist within the block. In the interpolation of blocks using the selective mining method the volumes of coal and waste in the block were estimated in proportion to the ratio of coal to waste thickness.

Blocks outside the search distance were classified as "undefined" and no quality values were assigned. Undefined materials were assumed to be waste in the A6 and C1 sub-zones and to be coal in the remaining sub-zones. The undefined coal is considered to be in the category of "Possible Reserves".

The specific gravity of coal was calculated from the formula:

$$S.G. = 1.211 + 0.00738\ (\%\ \text{dry-ash}).$$

Burn zone material was assigned a specific gravity of 2.16, and other waste 2.00 (see Section 5.2.5.2).

These factors were used in developing the composite sample values and in reserve calculations. In the "undefined" coal blocks calculations were based on the average specific gravity for the sub-zone.

Block values can be calculated for either the selective or non-selective mining cases and for different cut-off grades. Each set of block values is stored in its own "Quality File". In this study four "Quality Files" were prepared: for both mining cases each at two different cut-off grades - 9.3 MJ/kg and 6.98 MJ/kg.

4.6.2.3 Application of the Variable Block Model

The "Geometry" and "Quality" files can then be used for calculating the reserves within a designed pit or for the total deposit.

4.6.3 Reserves

1. Selective Mining

The proven and probable coal reserves of the Hat Creek No. 1 Deposit have been computed to be 739,523 million tonnes with a heating value of 17.71 MJ/kg, ash content 34.82% and sulphur content of 0.51%. The possible reserves are an additional 45 million tonnes.

These figures are for the proposed mining method of selective mining with removal of 2-m partings and a cut-off value of 9.3 MJ/kg. Table 4-6 and Table 4-7 show the distribution of the reserves by sub-zones and by 100-m bench elevations.

2. Non-selective Mining

If no waste parting removal is considered, then the reserves of the No. 1 Deposit based on a cut-off value of 9.3 MJ/kg would be (as shown in Table 4-8) 746,058 million tonnes coal at 16.72 MJ/kg, 37.73% ash, and 0.46% sulphur.

Table 4-9 illustrates what the coal reserves would be if the cut-off value was lowered to 6.98 MJ/kg.

4.7 GEOSTATISTICS

4.7.1 Preliminary Studies

The objectives of a geostatistical study is to measure the degree of continuity in a parameter (e.g. heating value, sulphur) throughout the deposit. With a knowledge of the degree of continuity, block values may be developed and an estimate made of the error of estimation.

Preliminary studies were assigned to Mineral Exploration Research Institute (IREM-MERI) to investigate the spatial distribution of heating value and sulphur.

An initial study of 14 sub-zones showed good continuity of heating value in the coal zones. The Inverse Square Distance Method (ISD) approximates the good continuity which was found to exist in the coal zones.

4.7.2 Sulphur

Initial studies of sulphur variation indicated poor continuity. However, many additional sulphur values were determined and incorporated in a geostatistical study of the total sulphur distribution in the deposit. Variograms were developed for each sub-zone and reviewed with IREM-MERI. With the additional data, good variograms, which indicate continuity and predictability, were obtained for 10 of the 16 sub-zones. The remaining six sub-zones showed random sulphur distribution.

Figure 4-6 presents a sample variogram.

The results of the variogram calculations are summarized in Table 4-10.

The parameters shown in Table 4-11 were used to produce estimates of the sulphur content of all the blocks contained within each sub-zone by kriging. The kriged block values were input to the Variable Block Model for use in reserve and pit evaluation calculations.

Table 4-12 shows a sample of the results obtained from kriging the block sulphur values in a portion of the A5 sub-zone. Two important conclusions are drawn from this table:

(1) The standard error of the individual blocks does not substantially deviate from the average value of 0.081;

(2) A large number of intersections were found to krige each block.

This indicates that in the A5 sub-zone, where sufficient data has been gathered, a confidence interval of 10% can be expected for the block mean at a 68% (1 S.D.) precision level. Individual blocks will vary up or down from this figure.

Additional tests indicated a 12% confidence interval for the two B sub-zones at a 68% precision level and a 20% confidence interval for D1, D2, and D4. The impact of the lower precision in D-zone is small because of the low average sulphur content. It must be emphasized that the previous precision figures do not apply to the six sub-zones that exhibited random behaviour. The distribution of these six sub-zones are predicted by classical statistics and shown on Table 4-10.

The precision figures were calculated for 75 m x 75 m blocks. During the mining phase, the confidence interval will be improved by:

(1) Drilling to test the quality distribution ahead of mining on a smaller spacing than the present 150 m x 150 m grid, to increase the number of samples and hence the confidence interval;

(2) Coal from several locations is mixed in the blending pile, which further reduces the sulphur variation.

4.7.3 Research Project

A research project was undertaken by IREM-MERI to investigate the applicability of a three-dimensional method to estimate heating value in 75 m x 75 m x 15 m bench blocks. Following three months of theoretical research, a new method to estimate grades in sedimentary deposits was developed. A series of computer programs have been developed to produce a model of the deposit using the new method. Careful checking and verification of the results is still required before the system is ready for application.

TABLE 4-1

SUMMARY OF DRILLING
UPPER HAT CREEK VALLEY
1925 - 1978

			No. 1 Deposit		No. 2 Deposit	
			No. of Holes	Meters	No. of Holes	Meters
1.	Exploration:	Pre-1974	22	4,375.8	64	21,799.9
		1974-1978	206	54,037.0		
2.	Geotechnical:	(slope stability foundation incl.)	74	9,714.9		
		(Geohydrological and offsite)	77	7,996.7		
3.	Miscellaneous:	Surficial Material Investigation,				
		Washability Sampling, etc. BAH) AH) P)	117	2,117.7		
	TOTAL		474	78,236.1	64	21,799.9

DH - Diamond Drilling
RH - Rotary
BAH - Bucket Auger HOle
AH - Auger Hole
P - Percussion

Table 4–2

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.), andesite, vesicular basalt.
	Unconformity (?)				
	Miocene or Middle Eocene ?		Kamloops Group: Finney Lake Formation	Not Determined	Lahar, sandstone, conglomerate.
			Unconformity		
	Late Eocene		Kamloops Group: Medicine Creek Formation	600+	Bentonitic claystone and siltstone.
			Paraconformity		
	Late Eocene to Middle Eocene	* 36 - 42	Kamloops Group: Hat Creek Coal Formation	550	Mainly coal with intercalated siltstone, claystone, sandstone and conglomerate.
			Kamloops Group: Coldwater Formation	375	Siltstone, claystone, sandstone, conglomerate, minor coal.
			Fault Contact or Nonconformity		
	Middle Eocene	43.6-49.9	Kamloops Group	Not Determined	Rhyolite, dacite, andesite, 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, rhyolite; tuff breccias, agglomerate.
	Erosional Unconformity (Duffell & McTaggart 1952)				
		98	Mount Martley Stock	Not Determined	Granodiorite, tonallite.
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.

* Based on palynology by Rouse 1977

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

Table 4-3

DEVELOPMENT OF STRATIGRAPHIC SUBDIVISION IN HAT CREEK COAL FORMATION

STAGE I	STAGE II	STAGE III	STAGE IV
A	A_1	A_{1-1} A_{1-2} A_{1-3} A_{1-4}	A1 A2 A3 A4 A5
	A_2(waste zone)	A_{2-1}	A6
B	B_1	B_{1-1} B_{1-2}	B1 B2
C	C_1(waste zone)	C_{1-1}	C1
	C_2	C_{2-1} C_{2-2}	C2 C3 C4
D	D_1	D_{1-1} D_{1-2} D_{1-3} D_{1-4}	D1 D2 D3 D4
Recognition of four broad zones in the No. 1 Deposit.	Identification of two waste zones - A_2 and C_1.	A_1 - divided into four subzones separated by three waste partings. B_1 - divided into two subzones. C_2 - divided into two subzones separated by a lenticular waste parting. D_1 - divided into four subzones of varying quality.	For uniformity and convenience each subzone was assigned its own suffix. Thus A_{2-1} and C_{1-1} the principle waste zones are represented by A6 and C1 respectively. Four additional subzones were introduced: A5, C2, C3 and C4.

TABLE 4-4

SUMMARY OF PROXIMATE AND ASH ANALYSES

EXCLUDING SAMPLES WITH HHV < 9304 KJ/KG & ASH > 70.00%

PROXIMATE, MOISTURE AND OTHER SUMMARY

	HHV (KJ/KG)	% ASH	% F.C.	% V.M.	% S	% MOISTURES AS RECVD.	AIR DRY	RES-IDUAL	EQUIL.	% CO2	%ALK. AS NA2O	WATER SOLUBLE % NA2O	% K2O
MAXIMUM	27398	62.18	72.83	46.61	5.54	36.92	31.56	22.35	35.60	15.60	1.57	.35	.60
MINIMUM	9317	7.96	7.56	.63	.03	2.26	.44	.22	16.76	.02	.08	.15	.01
RANGE	18081	54.22	65.27	45.98	5.51	34.66	31.12	22.13	18.84	15.58	1.49	.20	.59
WEIGHTED MEAN	18443	32.56	33.96	34.37	.55	22.54	12.93	8.90	23.83	1.42	.51	.26	.07
SAMPLE COUNTS	4028	4028	1375	1375	4026	1793	1792	4027	34	1445	951	18	19
SAMPLE CORE LENGTHS	15384	15384	7101	7101	15374	9276	9275	15383	239	6935	4418	54	58
ARITHMETIC MEAN	18037	33.76	33.54	33.90	.57	22.44	12.96	7.94	23.82	1.48	.51	.25	.05
SAMPLE COUNTS	4028	4028	1375	1375	4026	1793	1792	4027	34	1445	951	18	19
SAMPLE CORE LENGTHS	15384	15384	7101	7101	15374	9276	9275	15383	239	6935	4418	54	58
STANDARD DEVIATION	4456	12.94	8.79	5.35	.37	4.51	5.33	4.15	4.70	2.00	.24	.05	.13
COEFF. OF VARIATION %	24.70	38.32	26.21	15.79	66.21	20.12	41.18	52.28	19.74	35.12	47.95	21.87	25.75

MINERAL SUMMARY - %DRY ASH

	% SIO2	% AL2O3	% TIO2	% FE2O3	% CAO	% MGO	% NA2O	% K2O	% MN3O4	% V2O5	% P2O5	% SO3	% UNDET +ERR
MAXIMUM	77.16	40.19	1.85	56.00	47.08	8.07	5.42	1.80	1.94	.49	6.14	7.64	7.57
MINIMUM	17.06	9.26	.04	.10	.33	.00	.17	.00	.00	.00	.00	.04	-1.56
RANGE	60.10	30.93	1.81	55.90	46.75	8.07	5.25	1.80	1.94	.49	6.14	7.60	9.13
WEIGHTED MEAN	52.39	27.53	.94	8.40	3.55	1.57	1.40	.49	.17	.06	.42	2.08	.99
SAMPLE COUNTS	913	913	913	913	913	913	951	951	913	913	913	913	913
SAMPLE CORE LENGTHS	4159	4159	4159	4159	4159	4159	4418	4418	4159	4159	4159	4159	4159
ARITHMETIC MEAN	52.29	27.96	.91	8.34	3.46	1.57	1.35	.51	.16	.05	.38	1.98	.96
SAMPLE COUNTS	913	913	913	913	913	913	951	951	913	913	913	913	913
SAMPLE CORE LENGTHS	4159	4159	4159	4159	4159	4159	4418	4418	4159	4159	4159	4159	4159
STANDARD DEVIATION	7.29	5.10	.28	6.35	3.72	.76	.79	.30	.22	.04	.61	1.22	1.04
COEFF. OF VARIATION %	13.95	18.25	31.49	76.15	7.43	48.52	59.04	58.42	35.18	71.02	58.68	61.72	7.91

TABLE 4-5

SULPHUR FORMS

	Zone A	Zone B	Zone C	Zone D	Deposit Total
Pyritic Sulphur %	0.22	0.20	0.11	0.04	0.13
Organic Sulphur %	0.50	0.44	0.31	0.24	0.36
Sulphur as Sulphates %	0.02	0.03	0.01	0.02	0.02
Total	0.74	0.67	0.43	0.30	0.51

TABLE 4-6

RESERVE ESTIMATION BY SUB-ZONES WITH 2 m MINIMUM THICKNESS

* HHV CUT-OFF 9.30 * NO DILUTION * 2-METRE MIN. THICKNESS *

DATE : 27-Mar-79

ZONE	COAL TONNES	ASH%	HHV MJ/KG	SUL%	TOTAL VOLUME	COAL VOLUME	WASTE TONNES	UNDEF TONNES COAL	UNDEF TONNES WASTE	UNDEF VOLUME COAL	UNDEF VOLUME WASTE
BURN	0.	0.00	0.00	0.00	6769.	0.	14620.	0.	0.	0.	0.
A1	27223.	31.18	18.74	0.75	28365.	18905.	18921.	0.	0.	0.	0.
A2	41408.	39.60	15.88	0.77	40524.	27566.	25915.	0.	0.	0.	0.
A3	35944.	45.50	13.96	0.65	41833.	23244.	37178.	0.	0.	0.	0.
A4	49558.	40.75	15.58	0.66	57099.	32794.	48611.	0.	0.	0.	0.
A5	58665.	44.42	14.47	0.74	56168.	38139.	36056.	0.	0.	0.	0.
A6	7041.	50.48	12.32	0.63	65940.	4450.	122745.	0.	235.	0.	117.
B1	72681.	38.06	16.55	0.65	56301.	48816.	14317.	488.	0.	327.	0.
B2	68561.	37.78	16.66	0.71	63751.	46075.	33836.	1129.	0.	758.	0.
C1	10245.	48.83	12.89	0.54	160095.	6527.	286629.	0.	20507.	0.	10253.
C2	19842.	47.06	13.37	0.51	24326.	12740.	22515.	512.	0.	328.	0.
C3	20058.	46.09	13.77	0.36	23116.	12940.	17272.	2388.	0.	1540.	0.
C4	32405.	45.01	13.90	0.35	31660.	21013.	18457.	2188.	0.	1418.	0.
D1	70005.	31.35	18.82	0.29	56075.	48594.	4150.	7799.	0.	5407.	0.
D2	89306.	25.18	21.09	0.27	70872.	64010.	0.	9585.	0.	6862.	0.
D3	70476.	19.70	23.08	0.29	59822.	51984.	389.	10367.	0.	7643.	0.
D4	66106.	24.84	21.50	0.38	55313.	47436.	668.	10518.	0.	7543.	0.
TOTAL	739523.	34.82	17.71	0.51	898027.	505233.	702279.	44973.	20742.	31825.	10371.

NOTE: 1. TONNAGES ARE THOUSANDS OF METRIC TONNES
2. VOLUMES ARE THOUSANDS OF CUBIC METRES

TABLE 4-7

RESERVE ESTIMATION BY BENCHES WITH 2 m MINIMUM THICKNESS

* HHV CUT-OFF 9.30 * NO DILUTION * 2-METRE MIN. THICKNESS *

DATE : 27-Mar-79

SUMMARY FOR ALL BENCHES :

	BENCH	COAL TONNES	ASH%	HHV MJ/KG	SUL%	TOTAL VOLUME	COAL VOLUME	WASTE TONNES	UNDEF TONNES COAL	UNDEF TONNES WASTE	UNDEF VOLUME COAL	UNDEF VOLUME WASTE
1 (	1200)	0.	0.00	0.00	0.00	0.	0.	0.	0.	0.	0.	0.
2 (	1100)	235.	35.08	17.80	0.42	1489.	161.	2657.	0.	0.	0.	0.
3 (	1000)	40344.	40.41	15.64	0.56	79369.	26791.	105050.	341.	0.	244.	0.
4 (	900)	183099.	34.81	17.56	0.54	194776.	125031.	135066.	3476.	227.	2443.	114.
5 (	800)	209334.	33.47	18.15	0.51	206531.	143973.	122767.	1632.	8.	1177.	4.
6 (	700)	139151.	34.87	17.76	0.53	156375.	95061.	120642.	1373.	0.	994.	0.
7 (	600)	90910.	35.82	17.50	0.50	118810.	61816.	110798.	2116.	134.	1528.	67.
8 (	500)	53480.	35.75	17.57	0.41	80907.	36400.	77968.	5791.	2821.	4113.	1410.
9 (	400)	21455.	30.64	19.52	0.33	44104.	14982.	26946.	12713.	13578.	8859.	6789.
10(	300)	1514.	37.50	17.16	0.34	15666.	1019.	386.	17530.	3974.	12467.	1987.
11(	200)	0.	0.00	0.00	0.00	0.	0.	0.	0.	0.	0.	0.
	TOTAL	739523.	34.82	17.71	0.51	898027.	505233.	702279.	44973.	20742.	31825.	10371.

TABLE 4-8

RESERVE ESTIMATION WITH NON-SELECTIVE MINING AT 9.3 MJ/kg CUT-OFF

* HHV CUT-OFF 9.30 * NO DILUTION * NO MINIMUM THICKNESS *

DATE : 30-Mar-79

ZONE	COAL TONNES	ASH%	HHV MJ/KG	SUL%	TOTAL VOLUME	COAL VOLUME	WASTE TONNES	UNDEF TONNES COAL	UNDEF TONNES WASTE	UNDEF VOLUME COAL	UNDEF VOLUME WASTE
BURN	0.	0.00	0.00	0.00	6769.	0.	14620.	0.	0.	0.	0.
A1	43219.	48.17	13.04	0.53	28365.	27637.	1455.	0.	0.	0.	0.
A2	38078.	46.08	13.78	0.63	40524.	24600.	31848.	0.	0.	0.	0.
A3	32392.	54.60	10.94	0.53	41833.	20076.	43515.	0.	0.	0.	0.
A4	46830.	49.40	12.85	0.54	57099.	29744.	54710.	0.	0.	0.	0.
A5	60364.	49.93	12.68	0.64	56168.	38239.	35858.	0.	0.	0.	0.
A6	1839.	50.21	12.46	0.44	65940.	1164.	129317.	0.	235.	0.	117.
B1	69475.	36.91	16.83	0.64	56301.	46917.	18116.	485.	0.	327.	0.
B2	66861.	38.89	16.25	0.67	63751.	44716.	36553.	1135.	0.	758.	0.
C1	9043.	51.45	12.33	0.49	160095.	5692.	288300.	0.	20507.	0.	10253.
C2	23876.	49.10	12.65	0.52	24326.	15185.	17624.	517.	0.	328.	0.
C3	20766.	48.58	12.75	0.34	23116.	13244.	16665.	2416.	0.	1540.	0.
C4	32922.	46.72	13.24	0.34	31660.	21182.	18120.	2206.	0.	1418.	0.
D1	73542.	32.99	18.27	0.28	56075.	50669.	0.	7864.	0.	5407.	0.
D2	89306.	25.18	21.09	0.27	70872.	64010.	0.	9585.	0.	6862.	0.
D3	70852.	20.02	22.92	0.29	59822.	52179.	0.	10385.	0.	7643.	0.
D4	66693.	25.20	21.35	0.38	55313.	47769.	0.	10537.	0.	7543.	0.
TOTAL	746058.	37.73	16.72	0.46	898027.	503022.	706701.	45130.	20742.	31825.	10371.

NOTE: 1. TONNAGES ARE THOUSANDS OF METRIC TONNES
2. VOLUMES ARE THOUSANDS OF CUBIC METRES

TABLE 4-9

RESERVE ESTIMATION WITH NON-SELECTIVE MINING AT 6.98 MJ/kg CUT-OFF

* HHV CUT-OFF 6.98(3000BTUS) * NO DILUTION * NO MINIMUM THICKNESS *

DATE : 30-Mar-79

ZONE	COAL TONNES	ASH%	HHV MJ/KG	SUL%	TOTAL VOLUME	COAL VOLUME	WASTE TONNES	UNDEF TONNES COAL	UNDEF TONNES WASTE	UNDEF VOLUME COAL	UNDEF VOLUME WASTE
BURN	0.	0.00	0.00	0.00	6769.	0.	14620.	0.	0.	0.	0.
A1	44284.	47.83	12.94	0.53	28365.	28365.	0.	0.	0.	0.	0.
A2	46623.	49.68	12.68	0.59	40524.	29638.	21772.	0.	0.	0.	0.
A3	51223.	57.51	9.98	0.48	41833.	31339.	20988.	0.	0.	0.	0.
A4	62286.	53.28	11.63	0.49	57099.	38885.	36429.	0.	0.	0.	0.
A5	74965.	52.68	11.79	0.60	56168.	46911.	18513.	0.	0.	0.	0.
A6	3060.	58.15	10.65	0.36	65940.	1870.	127903.	0.	235.	0.	117.
B1	82809.	41.13	15.48	0.60	56301.	54868.	2212.	495.	0.	327.	0.
B2	76557.	42.01	15.24	0.66	63751.	50489.	25007.	1153.	0.	758.	0.
C1	14528.	55.61	10.80	0.52	160095.	8972.	281740.	0.	20507.	0.	10253.
C2	31374.	52.45	11.56	0.48	24326.	19658.	8679.	525.	0.	328.	0.
C3	28250.	52.04	11.63	0.32	23116.	17737.	7679.	2456.	0.	1540.	0.
C4	40558.	49.54	12.31	0.32	31660.	25765.	8953.	2236.	0.	1418.	0.
D1	73542.	32.99	18.27	0.28	56075.	50669.	0.	7864.	0.	5407.	0.
D2	89306.	25.18	21.09	0.27	70872.	64010.	0.	9585.	0.	6862.	0.
D3	70852.	20.02	22.92	0.29	59822.	52179.	0.	10385.	0.	7643.	0.
D4	66693.	25.20	21.35	0.38	55313.	47769.	0.	10537.	0.	7543.	0.
TOTAL	856909.	41.03	15.62	0.45	898027.	569124.	574497.	45235.	20742.	31825.	10371.

NOTE: 1. TONNAGES ARE THOUSANDS OF METRIC TONNES
2. VOLUMES ARE THOUSANDS OF CUBIC METRES

TABLE 4-10

TOTAL SULPHUR DISTRIBUTION IN SUB-ZONES
OF NO. 1 DEPOSIT

Sub-zone	Number of Inter-sections	Mean Sulphur %	Standard Deviation	Standard Error of the Mean
A1	32	0.723	0.193	0.034
A2	38	0.804	0.174	0.028
A3	42	0.634	0.137	0.021
A4	48	0.624	0.165	0.024
A5	54	0.739	0.187	0.025
*A6	-	0.540	0.169	0.027
B1	53	0.640	0.210	0.029
B2	57	0.664	0.174	0.023
*C1	-	0.450	0.300	0.051
*C2	55	0.486	0.209	0.028
*C3	56	0.356	0.213	0.028
*C4	67	0.369	0.266	0.032
D1	74	0.323	0.192	0.022
D2	77	0.260	0.096	0.011
*D3	84	0.298	0.0987	0.011
D4	86	0.388	0.102	0.011

* These sub-zones exhibit random distribution in the variograms.

TABLE 4-11

KRIGING PARAMETERS

Zone	Co	Sill	Range	Angle of Anisotropy	Anisotropic Ratio
A1	0.0100	0.0376	300	-	-
A2	0.0025	0.0300	390	90	2.5
A3	0.0032	0.0120	400	90	2
A4	0.0050	0.0265	600	90	3
A5	0.0110	0.0348	600	-	-
A6	-	-	-	-	-
B1	0.0260	0.0415	500	-	-
B2	0.0100	0.0257	500	90	2
C1	-	-	-	-	-
*C2	0.0437	0.0437	50	-	-
*C3	0.0454	0.0454	50	-	-
*C4	0.0780	0.0780	50	-	-
D1	0.0060	0.0300	540	-	-
D2	0.0008	0.0074	400	-	-
*D3	0.0060	0.0060	50	-	-
D4	0.0020	0.0100	200	-	-

* C2, C3, C4, D3 - exhibit random distributions in the variogram construction so they were kriged with a short range (50 m) and a pure nugget effect, i.e. Co=SILL.

A6, C1 - each block was assigned the zone average from Table 4-10.

TABLE 4-12

SULPHUR DISTRIBUTION IN SUB-ZONE A5

Mean of 25 blocks = 0.886

Average std. error = 0.081

Block Mean S%	0.947	0.907	0.900	0.947	0.951
Block Std. Error S%	0.085	0.083	0.080	0.095	0.114
No. of Intersections	13	13	12	13	14
	0.0934	0.897	0.904	0.958	0.985
	0.078	0.079	0.072	0.085	0.102
	14	14	15	15	19
	0.870	0.835	0.857	0.943	1.027
	0.073	0.072	0.073	0.077	0.085
	16	17	17	19	19
	0.809	0.773	0.798	0.899	0.997
	0.072	0.069	0.071	0.076	0.083
	19	20	20	23	22
	0.768	0.741	0.759	0.831	0.923
	0.078	0.070	0.071	0.073	0.083
	20	20	23	25	23

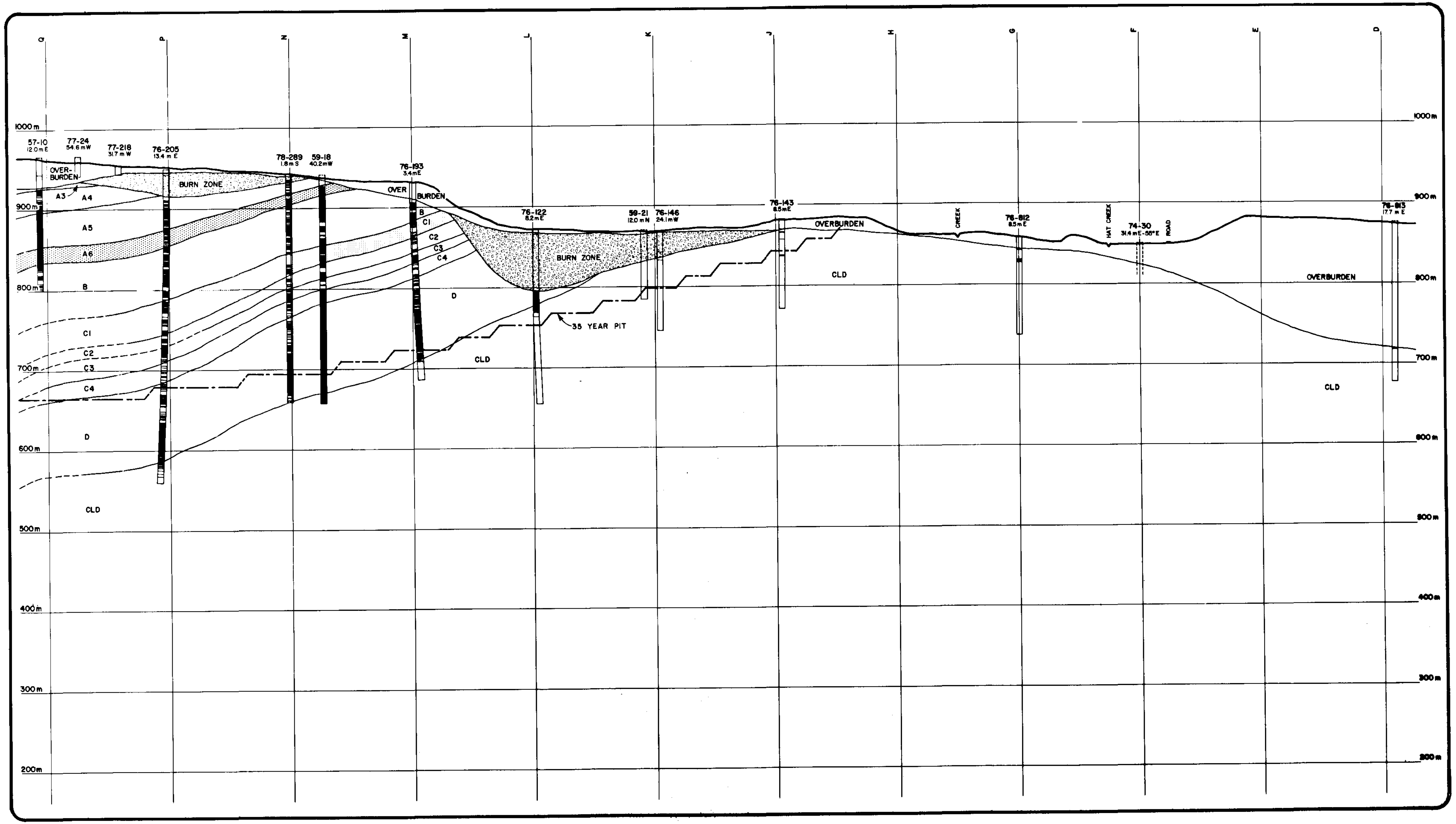
HAT CREEK PROJECT

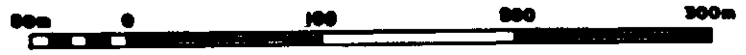
FIGURE 4-1

Geophysical Log DH 274 showing 16 sub-zones

00142 1/2 5

SOURCE: British Columbia Hydro and Power Authority





LEGEND

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

HAT CREEK PROJECT

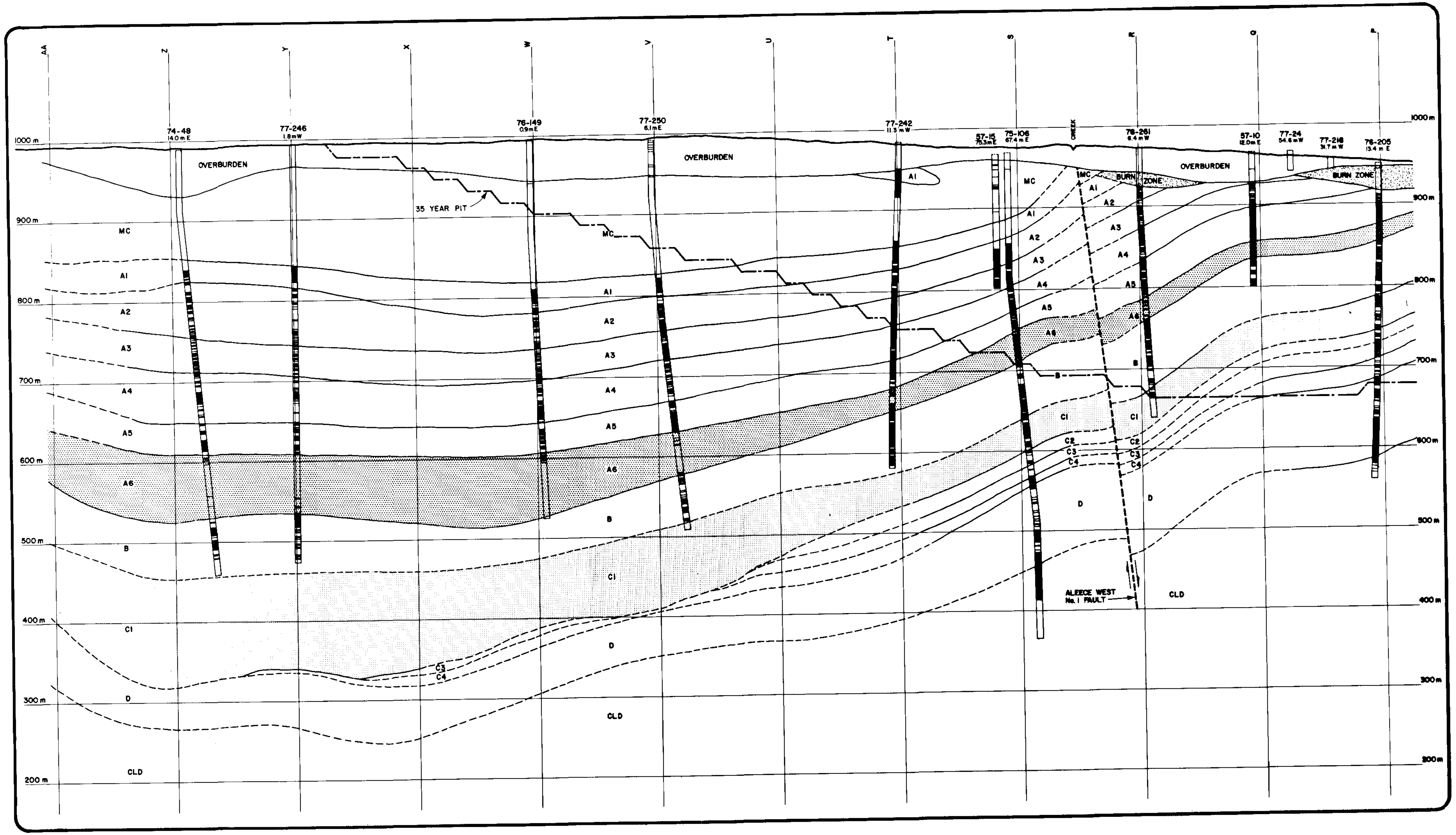
FIGURE 4·4A

GEOLOGICAL CROSS SECTION 18 NORTH

SECTION DRAWN LOOKING WEST

00142 1/2 6

SOURCE: British Columbia Hydro and Power Authority



LEGEND

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

50m 0 100 200 300m

HAT CREEK PROJECT

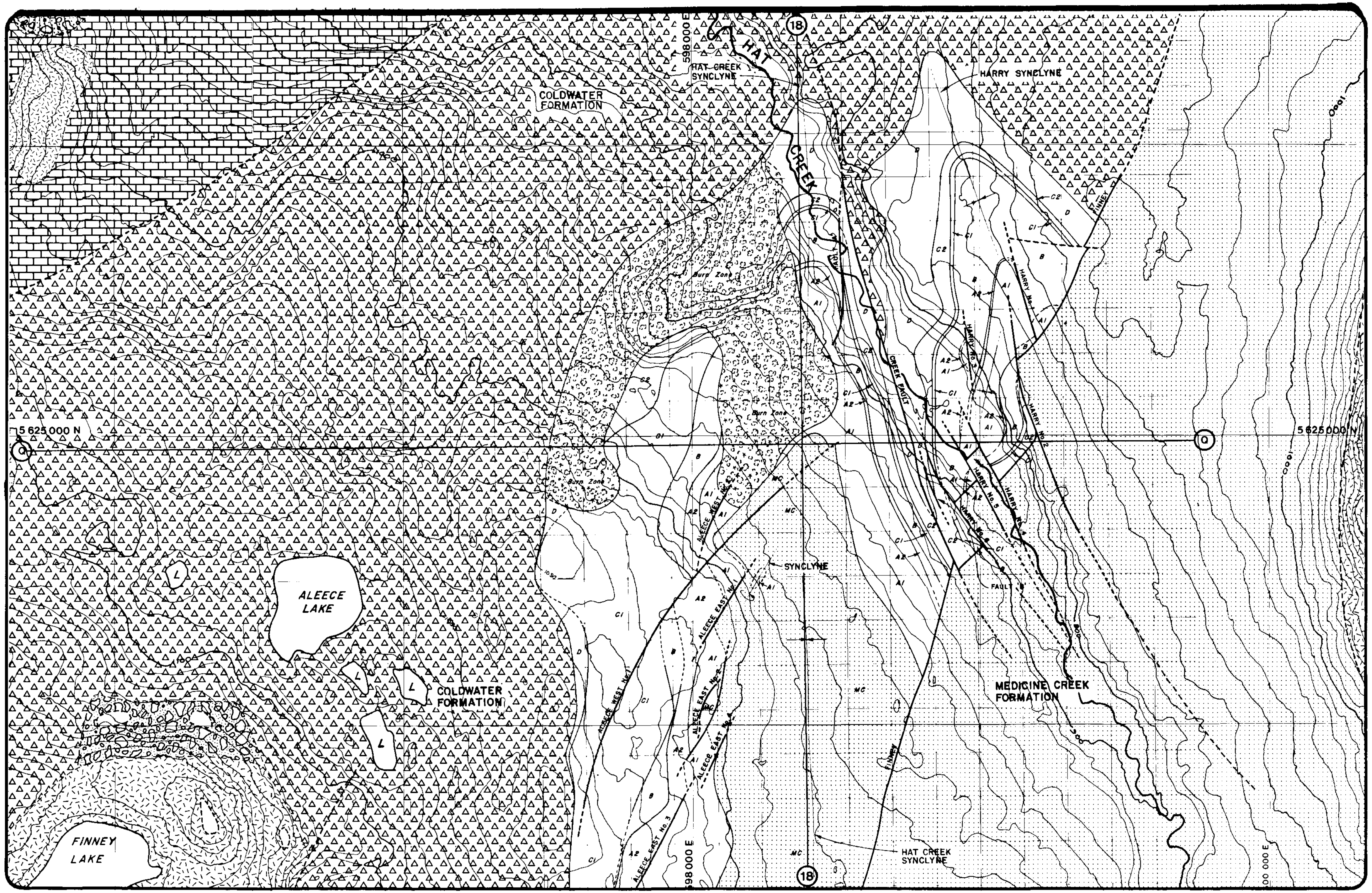
FIGURE 4-4B

GEOLOGICAL CROSS SECTION 18 SOUTH

SECTION DRAWN LOOKING WEST

00142 1/2 7

SOURCE: British Columbia Hydro and Power Authority



LEGEND

MIOCENE

PLATEAU BASALTS
Basalt, olivine basalt (13.2 m.y.), andesite, vesicular basalt

MIOCENE OR MIDDLE EOCENE

FINNEY LAKE FORMATION
Lahar, sandstone, conglomerate

EOCENE

MEDICINE CREEK FORMATION
Claystone, siltstone

HAT CREEK FORMATION
Coal, carbonaceous shale, claystone, siltstone, sandstone, conglomerate

SUBZONE & THICKNESS

A1	110 - 225 m	C1	0 - 170 m
A2	0 - 90 m	C2	15 - 55 m
B	50 - 70 m	D	80 - 100 m

COLDWATER FORMATION
Claystone, siltstone, shale, sandstone, conglomerate

KAMLOOPS VOLCANICS
Rhyolite, dacite, andesite, basalt and equivalent pyroclastics

CONIACIAN TO APTIAN

SPENCES BRIDGE GROUP
Andesite, dacite, basalt, rhyolite tuff, breccias, agglomerate

MOUNT MARTLEY STOCK
Granodiorite, tonalite

KAZANIAN TO VISEAN

CACHE CREEK GROUP
MARBLE CANYON FORMATION
Marble, limestone, argillite

GREENSTONE
Greenstone, chert, argillite, minor limestone and quartzite, chlorite schist, quartz-mica, schist

OUTCROPS

60 BEDDING OR LAYERING

CONTACTS (Confirmed, inferred)

FAULTS (Confirmed, inferred)

L LAKE

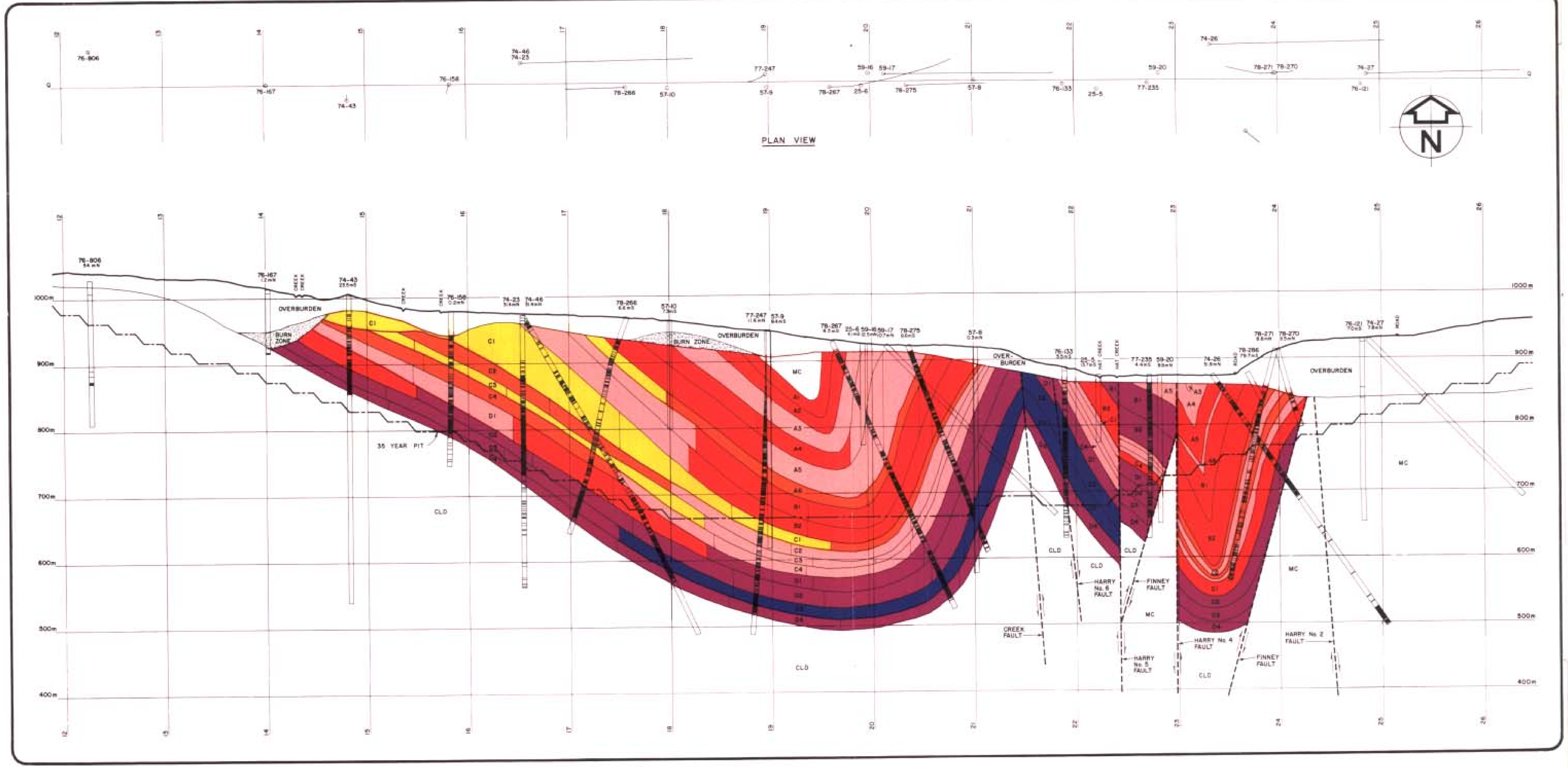
BURNT COAL ZONE

FIGURE 4-2

REGIONAL BEDROCK GEOLOGY NO. 1 DEPOSIT

00142 ½ (8)

SOURCE: British Columbia Hydro and Power Authority



LEGEND

SPECIFIC ENERGY RANGES
MJ/kg (DRY BASIS)

< 9.3	MC MEDICINE CREEK FORMATION
9.3 - 13.0	CLD COLDWATER FORMATION
13.0 - 16.5	BURN ZONE
16.5 - 20.0	A6 A6 SUB-ZONE
20.0 - 24.0	C1 C1 SUB-ZONE
> 24.0	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

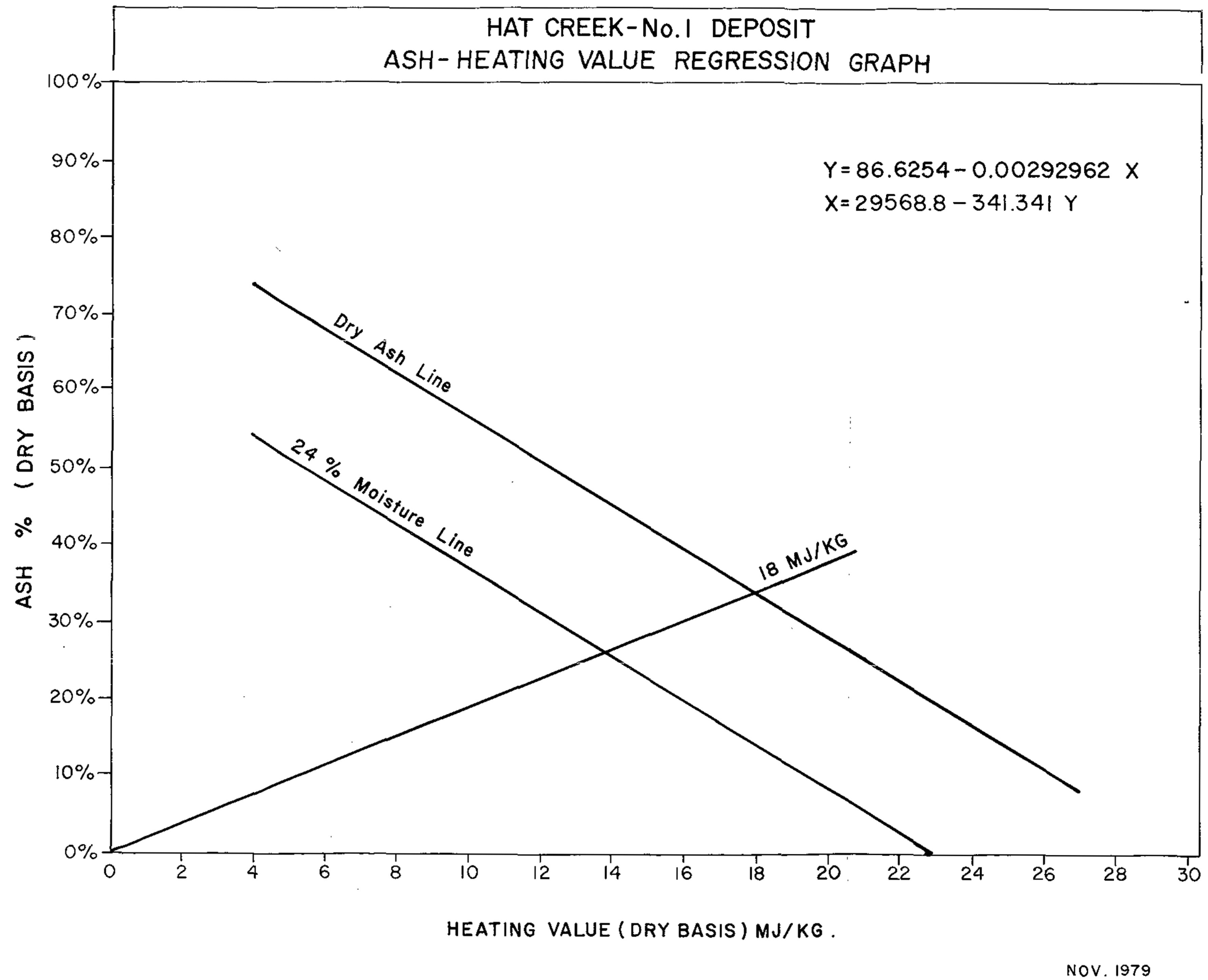
FIGURE 4·3

GEOLOGICAL CROSS SECTION SECTION Q

SECTION DRAWN LOOKING NORTH

00142 1/2 9

SOURCE: British Columbia Hydro and Power Authority

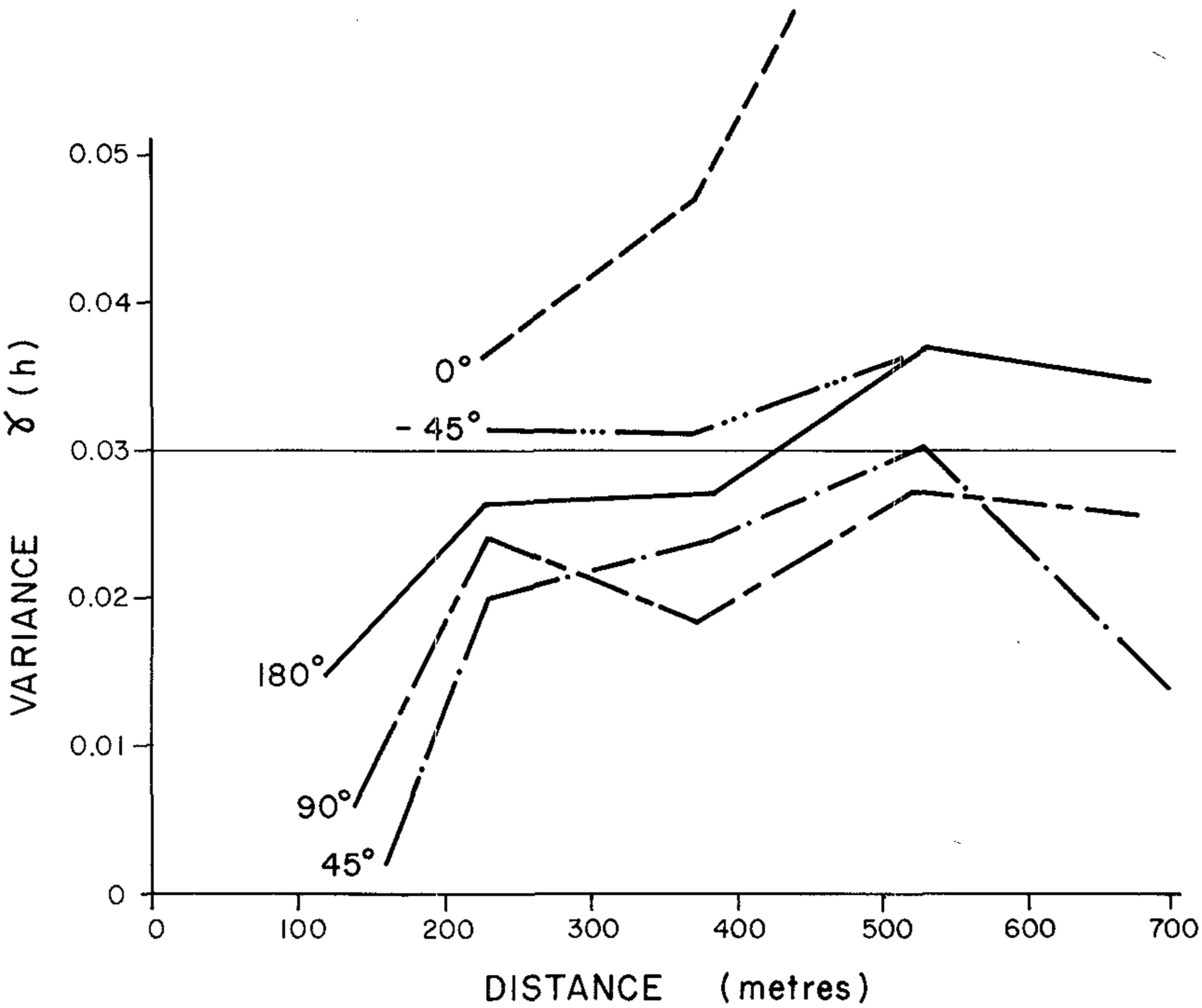


HAT CREEK PROJECT

FIGURE 4-5

Regression Curve Ash-Heating Value (Ash < 60%)

SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 4-6

Sample Semi-Variogram, Sulphur — Sub Zone A2

SOURCE: British Columbia Hydro and Power Authority

5 MINE PLANNING

Page

No.	LIST OF TABLES	Page

LIST OF FIGURES

LIST OF FIGURES (Continued) Page

SECTION 5

MINE PLANNING

5.1 INTRODUCTION

The objective of this study is to develop a mining plan that is both technically practicable and economically sound. Its purpose is to provide a reliable supply of coal of consistent quality to meet the forecast requirements of the powerplant over the estimated 35-year project life.

Conceptual design studies completed in 1976 by Powell Duffryn - National Coal Board (PD-NCB) evaluated the potential mining methods and economics of mining both the No. 1 and the No. 2 deposits. From these studies, the recommendation was accepted that the No. 1 Deposit was the more economic for development and that open-pit mining was the most appropriate method. This section describes the basis and the methods of planning used, and presents the pit design and production schedules developed.

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.

Because the time frame for this plan extends beyond 40 years it is important that options for future development are not foreclosed. Thus a major constraint in planning the mine is to ensure that planned activities do not jeopardize the possibility of ultimately mining the total reserve in the No. 1 Deposit or impede development of the No. 2 Deposit. To meet this constraint, the pit has been developed in a logical, sequential manner to produce 35 years' coal supply. The pit is developed with working slopes a few degrees flatter than the designed final pit slope. As the pit limits are reached, the slopes are steepened to conform to the design. Should it become necessary to extend the life of the pit, the degree of difficulty entailed would be directly related to the lead time associated with the change of plan. A decision made to extend mining before final pit slopes are reached would permit a smooth continuation of the operation. A last minute decision would result in the need for flattening pit slopes all the way to the surface before significant tonnages of coal could be produced.

Should the total resource of the No. 1 Deposit ultimately be mined, the pit would be over 200 m deeper than the presently planned pit. The technical and economic feasibility of mining to this greater depth has not been established. Further studies, both mining and geo-technical, would be required for this purpose.

In locating permanent facilities and waste dumps, care was taken to ensure that they were placed beyond the projected ultimate pit limits. The exceptions to this are the locations of the Hat Creek Diversion Canal, the headworks dam, and the pit rim dam. In these cases, it was shown to be more economic to relocate the facilities when necessary.

A prerequisite to any significant development of the coal deposits is the diversion of Hat Creek. The Hydro Electric Design Division of B.C. Hydro has prepared a Preliminary Engineering Design Report for the diversion of both Hat Creek and Finney Creek. The results of this work have been incorporated in this report.

The planned diversion of Hat Creek consists of a head-works dam to control the flow and channel it into a diversion canal, which carries the water around the East side of the pit before returning it through a buried conduit to the creek downstream of the mine facilities. The diversion system is designed to handle the 1,000-year return flood. An emergency spillway is incorporated into the headworks structure to prevent the overtopping of the dam with the overflow water channelled to the mine.

5.2 DESIGN CRITERIA

5.2.1 Powerplant Requirements

Based on the planned powerplant operating regime, annual coal consumption was determined from pre-production to the end of Year 35. These fuel requirements were established for the following functions:

(1) Commissioning of boiler units in the pre-production year and the first three years of operation;

(2) Establishing a two-week dead stockpile at the powerplant and a one-week live blending pile at the mine;

(3) Annual commercial power generation based on forecast capacity factors.

5.2.1.1 Powerplant Needs at Target Quality

The powerplant needs based on target quality of 18 MJ/kg dry basis and 23.5% moisture are as follows:

Year	Boiler Units	Net Capacity (MW)	Average Capacity Factor (%)	Million Tonnes at 18 MJ/kg Dry Basis and 23.5% Moisture
Pre-Production				1.11
1	1	500	69	3.15
2	2	1,000	60	4.79
3	3	1,500	60	7.35
4	4	2,000	61	9.45
5	4	2,000	65	10.60
6-15	4	2,000	70	10.86/year
16-25	4	2,000	65	10.09/year
26-35	4	2,000	55	8.53/year

A further potential coal demand that the mine must be capable of satisfying could occur if the powerplant is required to operate continuously for a period of up to six months at maximum continuous rating on all four units.

5.2.1.2 Allowable Coal Quality Variations

A live stockpile of 300,000 t of coal (one week's coal supply at maximum rating on all four units) would be used to blend the run-of-mine coal and minimize the quality variations.

The quality of coal delivered to the powerplant may vary between 17 MJ/kg and 19 MJ/kg, with a sulphur content between 0.46% and 0.56% on a dry coal basis.

5.2.2 Material Delivery Points and Mine Facilities Location

The delivery points for coal and waste, and two locations for the construction of the mine facilities complex, are as follows:

Coal

The coal delivery point, determined in consultation with the powerplant engineering staff, is the receiving conveyor at the powerplant. The responsibility of the mine for coal-handling terminates at this location.

Low-grade Coal

Low-grade coal will be delivered to a dry beneficiation plant. Provision must be made to combine beneficiated coal with the run-of-mine coal and to remove rejects to the waste dumps.

Waste

Mine waste must be contained in waste dumps close to the mine. Weak waste materials must be retained by engineered embankments. Dumps must not overlie any coal or be located where they will restrict any possible pit expansion. Houth Meadows and Medicine Creek have been identified as suitable areas for waste dumps. Small areas around the No. 1 Deposit and close to the proposed dumps will be used as temporary topsoil storage areas.

Mine Facilities Complex

Potential locations for constructing the mine facilities complex are:

(1) The North-Eastern end of the Upper Hat Creek Valley South of Indian Reserve IR-1 and bounded by Harry Creek and Hat Creek;

(2) The area located North-East of the confluence of Hat Creek and Medicine Creek, and between the No. 1 and the No. 2 Deposit.

The mine facilities complex and any other permanent structures should be 300 m minimum distance from the rim of the ultimate pit and not overlie any coal.

5.2.3 Geotechnical Constraints

5.2.3.1 Introduction

A geotechnical assessment program was initiated and assigned to Golder Associates in 1976. Extensive field investigations took place along with the exploration drilling programs over three years, with special drilling programs directed to geotechnical objectives. The major purpose of the work has been to establish safe working slopes for the open-pit mine in the No. 1 Deposit.

The stability of these slopes is controlled by the strength of the materials and the groundwater conditions in the area.

The reports by Golder Associates culminate in a final report: "Geotechnical Study 1977-78" dated December, 1978. There are six volumes presenting the detailed findings of all the work, with 16 appendices supporting the main text.

5.2.3.2 The Nature of the Materials in the No. 1 Deposit

The unconsolidated overburden is mostly strong granular glacio-fluvial sands, gravels, and till.

The slide material is very weak, consisting of loose, mixed debris, mostly soft and bentonitic.

The bedrock, soft clays, and siltstones exhibit varying low strengths and are weak when compared with hard rock formations.

The coal has greater strength than the above, but is still weak.

Overall, the materials represent saturated weak rocks that were originally deposited in a lacustrine environment and are softened when wet.

5.2.3.3 Geotechnical Conclusions

Pit Slope Stability

The following design slope angles recommended by Golder Associates for the 1978 Mining Feasibility Report by CMJV have been accepted for this Mining Report. Figure 5-3 presents Golder Associates' schematic diagram for these angles around the pit.

Surficial deposits (other than slide debris)	25°
Slide debris	16°
Coal	25°
Coldwater rocks (other than coal)	20°

The results of laboratory strength tests carried out on the Coldwater rocks show a wide spread in values, but do not indicate significant variations between different sectors of the pit. Therefore, there is no justification at this stage for varying the slope angles within the different Coldwater rock materials. As more data is accumulated in the future during the detailed design phase and early excavation, further refinement of slope angles can be anticipated.

In arriving at these steeper recommended angles, the following assumptions have been made:

(1) That pit slope depressurization by negative pore pressure generation would be moderately successful;

(2) 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;

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

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

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

During the current study it became apparent that depressurization would be more difficult to achieve than anticipated and that, except in restricted areas, conventional means (pumping wells, adits, horizontal drains) would not be appropriate. However, the current design is markedly different from the PD-NCB pit, on which all the original work was done (see Golder Associates' Report No. 6). The pit involves flatter interim pit slopes than final slopes and a progressively expanding pit which generally does not excavate slopes to final depth until the last 10 years. The geotechnical consequences of this design are favourable, since the materials in the slopes would only be stressed at low levels during the earlier years of mining (see Figure 5-1). Much experience could be gained within the deposit while slopes of modest height were cut at flat angles. Moreover, the in-situ groundwater studies and the laboratory testing program have indicated that depressurization by the development of negative pore pressures on excavation should be a significant factor in maintaining slope stability. (Figure 5-8)

The major conclusions on slope stability for the Mining Report are, therefore, that the final slopes can be excavated at the slope angles stated above, but with the following reservations:

(1) That it would be possible to achieve slope stabilization by pumping or gravity drainage only in very limited areas of the pit;

(2) That whilst slope stabilization by the development of negative pore pressures is likely to be effective in many areas of the pit, it would also be marginal in some places; these areas are difficult to predict in advance;

(3) The approach to mine planning currently being used permits valuable experience to be gained with the slopes whilst negative pore pressures are still operative in the earlier years.

5.2.3.4 The 35-Year Pit Design

Flatter interim pit slope angles in the coal benches during the opening up or development of the pit have been incorporated (see Figure 5-1).

The overall slope during any interim pit phase will always be less than the recommended final slope angles.

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°.

5.2.3.5 Handling Overburden Surficial Deposits

The sand, gravel, and glacial silts on the Eastern perimeter are 92 m to 122 m thick and will be required for construction and fill purposes early on. The materials are dense in situ and will be stable at much steeper slopes than the bedrock clays. However, there is a water table contact with the top of the bedrock that may present drainage problems.

The slide masses on the Western and South-Western perimeters present a stability problem. Movement of these slide masses could be re-activated along pre-existing 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.

A drainage program will be initiated and maintained to reduce this potential threat. Also, the slide front around the perimeter of the pit will need clearing back and a "creep-monitoring" system set up.

The active slide on the North-West perimeter will be stabilized by surficial drainage, diverting Hat Creek, and putting in a fill ramp at the toe of the slide across the valley 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. About 30 million m^3 of this material will have to be excavated in the 35-year pit, and it is known to be very sticky and difficult to handle when wet in Springtime. It may be impractical to maintain benches for more than two years in this unconsolidated overburden on the Western side. Rather, the ground could be evenly sloped to 16° from bedrock to surface perimeters.

5.2.3.6 Bench Strengths

For economic efficiency, a standard bench height of 15 m has been considered to be practical and safe. Local conditions may dictate using lesser bench heights.

Instability of some benches would be time-dependent, where failures could depend on the dissipation of pore pressures. Much of this activity is expected to develop within weeks or months of the digging (page 78, Golder).

The clay-rich rocks, being dispersive, are highly susceptible to erosion by water, especially when brecciated.

Much clean-up work should be expected on a regular basis, because of the highly dispersive nature of the lower claystone on the Western side of the pit. Mine operations will have to carefully plan the approach and access for a return to areas where the benches have been left standing for a number of years.

5.2.3.7 Other Geotechnics

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 (page 73, Golder). 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. Waste Dumps

Because of the large proportion of the weak bentonitic clay, conventional mine waste dumps are not feasible. It is necessary to store the material behind engineered embankments. No major geotechnical problems are envisaged for waste dump or embankment stability, either in Houth Meadows or Medicine Creek, provided material quality selection and the recommended designs are adhered to.

Embankments would be constructed of clean granular fill from the stripping of the glacio-fluvial sands and gravels; the materials could be placed by spreader.

The conglomeratic unit of the Coldwater Formation below the coal would provide a sufficiently strong buttress between the Houth Meadows Waste Dump and the pit to inhibit instability during the pit operation.

5.2.3.8 Field-Test Knowledge and Experience (Bulk Sample Program)

The bulk sample excavations were undertaken in 1977 in disturbed, weathered materials above the water table. Much information has been obtained from this work program defining the strength and nature of the materials in both coal and waste zones. Equipment performance of motor scrapers, hydraulic shovel excavators, rear dump trucks, and bulldozer ripping, coal-crushing, waste dump stability, road-making, revegetation of dumps, drainage conditions, and climatic effects of freezing-thawing on bench faces causing detrition - were all studied and yielded basic information from which conclusions have been drawn for mine planning.

The strength and nature of the deep-seated coal and clay beds has been geotechnically evaluated by testing drill core samples from exploration drilling programs covering the entire No. 1 Deposit and its adjacent perimeter area. The results of uniaxial compressive strength-testing of the rocks are presented graphically in Figure 5-2 by Golder Associates.

5.2.3.9 Mining Methods Assumptions

Selective mining by careful removal of the clay partings within the coal beds has been planned. Drilling and blasting the benches is neither required nor desirable; hydraulic excavators can do the digging efficiently and provide the selectivity of materials for loading in trucks. (Golder's Tables 5-1 and 5-2 indicate the test results of the various materials and "diggability" under "Geotechnical Comments".)

The changes that will necessarily be introduced into the geometry of pit slopes as mining proceeds can only be determined as actual experience in excavation of the various materials is obtained.

Adoption of a flexible mine plan and selection of equipment initally must allow for changes in mining methods and pit design later on.

5.2.3.10 Ultimate Slopes

The eventual dissipation or equilibration 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 if it is eventually mined by open-pit methods.

It is anticipated that after a period of mining, the pit will have grown to a size that will require realignment or replacement by other means, such as a tunnel or conduit of some 1,400 m of the Hat Creek Diversion Canal. Subsequent realignment of the canal to suit the ultimate pit slope is considered to be the most economical arrangement, but mining of the total resource may preclude this due to the surface ground slope.

The alternative scheme for the long-term diversion of Hat Creek is to put it in a tunnel around the Eastern side of the pit. The timing of the construction of this tunnel will depend on what happens with mining and slope stability near the canal. The surface ground between the pit excavation and the canal will be constantly monitored for both effectiveness of depressurization during mining and also for signs of movement or "creep". Such movement could lead to cracking or rupture of the canal, causing seepage into the Eastern side pit walls and consequent instability. Action will be taken to relocate the canal when necessary.

5.2.4 Hydrology

5.2.4.1 The Hydrology Program

Its purpose is:

- to define the groundwater-pressure regime;
- to assess the feasibility of depressurizing the proposed mine slopes by drainage and pumping;
- to evaluate the permeability of the materials, their dewatering characteristics, and recharge;
- to test depressurization by electro-osmosis.

5.2.4.2 Hydrological Relationship to Geotechnical Constraints

Slope depressurization is necessary if the final pit slope angles to be used for pit design are to be steeper than those calculated for undrained slopes.

The permeability of the materials to be excavated in mining to depth controls the capability of drainage, which in turn would determine the handling characteristics of the materials to be handled. It also controls the ability to depressurize the ground in situ.

Knowledge of what groundwater flows exist provides the basis for predicting slope stability and the possible hazards of activated slides. Depressurization by dewatering and unloading is necessary to achieve improved pit slope angles. The quantities and qualities of water to be intercepted by the pit excavation as it is deepened establishes the design basis for the mine drainage scheme.

5.2.4.3 Work Programs and Methods Used to Determine the Hydrology

1. Piezometers

During three years of exploration drilling programs, piezometers have been installed in over 200 holes. Many of these holes have multiple standpipe piezometers. Records have been accumulated from reading the water levels in these holes, and 184 working piezometer holes are still being recorded monthly. The opportunity was taken to install instruments in holes being drilled for coal exploration within the pit area, in holes being drilled for geotechnical purposes in the pit slopes, in the slide area, and in the waste dump areas of Houth Meadows and Medicine Creek. Full piezometric coverage of the site in depth and area has been obtained as shown on Figure 5-4.

Sixteen of the holes have had more sensitive pneumatic packer-type piezometers installed in the standpipes to give pressure-fluctuation readouts. Some of these were used for quick response in the pumping tests. Piezometer hydrographs have been prepared from the piezometric data and evaluated.

2. Pump Testing Program

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 5-3 gives a summary of results of these field tests on bedrock units.

5.2.4.4 Conclusions

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 5-5 shows the variations.

5.2.4.5 The Hydrogeological Picture of the Hat Creek Valley

From the work performed by Golder Associates, a reasonably clear model for the Hat Creek Coal Basin has emerged. The model can basically be divided into three hydrogeological units: the surficial deposits, the coal, and the sediments above and below the coal.

The surficials are highly variable, changing from predominantly slide debris and till on the West to gravels and fine sands on the East. There is a wide range within the hydrogeological parameters in this unit, with the alluvium in the valley bottom giving relatively high hydraulic conductivities. They constitute the major water-bearing units in the Hat Creek Valley.

The coal parameters are also variable and are not easily characterized. Falling head tests suggest that the B and D-zones are generally four orders of magnitude more permeable than the A and C-zones, possibly because of their generally lower ash content and greater development of structure. Although the single pump test (W-77-1) in the D-zone coal did not suggest good drainability, it has been assumed that these materials will be more drainable than the non-carbonaceous Coldwater sediments. A pump test (W-78-2) in the cleaner part of the A-zone coal has shown that this unit can be relatively easy to drain, at least in some areas.

The remaining Coldwater sediments (claystone/siltstone/conglomerate) have very low hydraulic conductivities and low consolidation coefficients.

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. The flow systems are shown in Figure 5-6.

The Western bench slopes would not be well drained and groundwater discharge in the form of springs and seeps are common, particularly below the 970 m contour. This South-West perimeter of the pit frontage, with its overlying masses of inactive slide material, could become unstable again due to mining excavations.

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 has to 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.

5.2.4.6 Controls and Preventive Measures

1. The Mine Drainage Plan

Described in "Section 6.3.2.1, The Open Pit", this report deals with the diversion of Hat Creek and Finney Creek perimeter drainage, in-pit drainage, and dewatering wells.

In Section 6.3.2.2 the whole drainage scheme of the South-West slide area is described.

A more detailed document of the whole drainage system has been prepared by CMJV Consultants, which incorporates the Golder Associates' recommendations and findings. (Ref: "Hat Creek Project - Mine Drainage Report", CMJV, October 1979)

2. Pressure Control by Electro-Osmosis

In "difficult-to-drain" situations this method can be used to increase the factor of safety against failure by driving the water away from a face to a point where it can be pumped - e.g. a well.

An electric current is fed into the ground between two electrodes. The potential difference set up between the electrodes in ground of low hydraulic conductivity creates seepage pressures due to electro-osmotic flow, which directs water away from the anode to the cathode. The cathode can be constructed in the form of a well which can be pumped.

A test was carried out at Hat Creek at pump test hole #W 77-2.

Reductions in pressure of over 14 m head were achieved at the anode over a period of 20 days, and it was concluded that the technique could have some application at the site. The technique is mostly suited for stabilization of limited areas, because of the time and cost of the installations needed.

5.2.4.7 Evaluation of Piezometer Hydrographs

Hydrographs of 227 piezometers installed in 137 boreholes drilled in 1976-78 have been studied and are presented in Appendix 12 of Golder Associates' Report. The hydrographs are based on monthly readings in both standpipe and pneumatic piezometers. The following conclusions may be drawn from this analysis:

(1) Standpipe piezometers installed in claystone units of low hydraulic conductivity are slow to respond. Basic time lags range up to six months;

(2) The pneumatic piezometers are significantly more responsive; however, a reading resolution of ±0.5 m with current read-out sensitivity reduces their capability to detect seasonal changes;

(3) Most piezometers showed a slight rise (0.3 to 2 m) during the Fall and early Winter, and some shallow piezometers in more permeable rock zones showed a similar rise during the Spring melt in April to May;

(4) Once the piezometers stabilized, the observed seasonal changes in piezometric levels appear to be less than 3 m for all but a few installations;

(5) Piezometers in the more permeable surficial materials, with the exception of those close to watercourses, showed similar responses to those observed in the bedrock zones.

A longer period of recording will be necessary before a more definitive rainfall-recharge relationship can be determined. However, these hydrographs show that there are two periods during the year when groundwater recharge does take place, and, as expected, the seasonal changes in piezometric evaluations are very small.

5.2.5 Material Characteristics

5.2.5.1 General Description

The open pit will be directly concerned with the following four major types of materials:

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

A large number of identified rock types was consolidated into 10 principal categories of materials:

(1) Clean coal;

(2) Silty coal and shaley coal;

(3) Carbonaceous shale and carbonaceous claystone;

(4) Shale and claystone;

(5) Silty claystone and silty shale;

(6) Coaly shale and coaly siltstone;

(7) Carbonaceous siltstone;

(8) Siltstone;

(9) Sandstone;

(10) Conglomerate.

The strengths and geotechnical characteristics of these materials are dealt with in Section 5.2.3, along with concerns for slope stability and design slope angles. See also Tables 5-1 and 5-2.

In general, the open-pit mining of the Hat Creek No. 1 Deposit will be in relatively weak and soft rocks and overburden. 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 hard rock. The coal itself varies from hard to soft types, depending on how much clay is in it.

The other major factor inherent in the materials being mined is the moisture content of the materials. From the drilling programs, bulk sample excavations, and geological theory of deposition of the coal beds, it is known that all the materials will be saturated and almost non-drainable. Bench faces may develop a skin dryness, but this will probably only penetrate to a maximum of one metre after a year of exposure.

Climatic changes over Winter freezing and Spring thawing will affect material characteristics because of their high moisture content.

The bentonitic clay seems prevalent in a lot of the upper-zone interbed partings, especially in the West and South-West areas of the pit. This clay absorbs moisture, swells when wet, and becomes extremely sticky and slippery. Waste materials will react according to how much bentonite (montmorillonite) they contain.

The wet low-grade coal is generally mushy and weak in strength. This will cause problems in mining the A and C zones' benches.

5.2.5.2 Specific Gravity

In the course of the exploration drilling programs, specific gravity tests were conducted in 5,622 samples, using a variety of methods. This testing covered a large number of materials of both coal and waste.

The specific gravity test results, together with the ash and moisture determinations for the samples, were input to a computer data file. The data were retrieved from the file summarized by various classifications. For each case, cumulative frequency distribution curves were plotted and standard statistical parameters calculated: mean, standard deviation, standard error, and range. Scatter diagrams were produced in each case for ash vs. specific gravity, ash vs. moisture content, and specific gravity vs. moisture.

Examination of the scatter diagrams produced the following conclusions:

(1) For coal and coaly materials, there is a distinct ash-specific gravity relationship;

(2) There is no apparent difference in this relationship in the different coal zones;

(3) In the higher ash range, there is some indication of a curvilinear relationship; however, with the scatter of the available data, this could not be confirmed;

(4) There are no apparent relationships between moisture content and ash, nor between moisture content and specific gravity.

Since the distribution diagrams for coal demonstrated the same trend and overlapped, the plot with the least scatter that adequately represented the range (303 samples), was selected to establish the regression relationship:

Specific Gravity (coal) = 1.21104 + 0.00738 x Dry Ash%

(Correlation coefficient = 0.90510)

For comparative purposes, a second relationship was determined for 120 samples of shaly coal. This relationship produces very similar results to the first equation over most of the range, with a maximum difference of 2% at the extremes, which increases the confidence in the selected equation.

The specific gravity of the many types of waste materials does not lend itself to analysis and correlation. Based upon inspection of the data, the following were selected for use in the study:

Surficials and Waste Rock:	Specific Gravity = 2.00
Burn Zone:	Specific Gravity = 2.16

5.2.5.3 Swell Factors

The swell factors of three primary materials were studied and the results are as follows:

	As Mined	Dumped in Stockpiles
Coal	35%	35%
Waste above bedrock		
- Granular surficials	20%	15%
- Cohesive surficials	30%	25%
Bedrock waste	30%	25%

Lacking site-specific measurements to derive swell factors for large-scale materials-handling activities, each planned waste dump was arbitrarily limited to approximately 75% of its recommended capacity. This would allow a safety margin should swell factors during actual operation be greater than those used in the study.

5.2.5.4 Material Cutting Resistance

Uni-axial compression tests and tri-axial shear tests were carried out to determine the cutting resistance of the various surficial and bedrock materials.

The average test results are shown on Tables 5-1 and 5-2. The same tables indicate the moisture content by type of materials, which exerts a major influence on the characteristics of mined materials and related equipment productivity.

5.2.5.5 Bearing Capacity of Materials

For the mine buildings and fixed structures generally, the in-situ strengths of both surficial materials and bedrock are expected to exceed the minimum specification of 5 kg/cm^2 for foundation support.

A study was made to determine the ability of roads to support large mobile equipment working at high production rates. Roads on granular surficial materials were considered to require minimal preparation, construction activities consisting of filling excavations or other hollows with adjacent materials to attain a uniform gradient, and providing for drainage. Normal road topping would be applied to the graded surface. Specific road-building technology is only considered necessary in the North-West slide area.

Roads on waste rock and in-situ coal are considered capable of supporting the traffic of 154-t trucks, provided an adequate sub-base is constructed. As the effective moisture in most of the bedrock materials is below the derived values for plastic limits, geotechnical conclusions indicate that heavy traffic is likely to compact rather than to liquify the materials.

The design of haul roads crossing the active slide area must take into account two problems: soil creep, and localized "boils" in the bentonite clays. The first problem requires construction of a higher standard sub-base and more frequent upkeep, resulting in higher localized road maintenance costs. The recommended solution to bentonite "boils" is simply to identify them prior to road building, and to avoid them.

5.2.6 Dilution and Mining Loss

No allowance is made for dilution and mining loss in this preliminary engineering study.

5.2.6.1 Dilution

In most mining studies it would be appropriate to make an allowance for accidental inclusion of waste materials mined with fuel-grade coal.

The mining approach recommended for the Hat Creek Coal Deposit stipulates that waste partings shall be selectively removed during mining when the thickness of these partings exceeds two metres. The quantity of diluents in the run-of-mine coal would therefore be a function of the surface area of the coal/waste interfaces and the attitude of these interfaces.

The sampling procedures carried out on Hat Creek drill cores have included significant quantities of waste material in the samples of good quality coal. The coal quality values used in mine planning evaluations have already been reduced due to this factor. In actual mining operations much of this included waste would be rejected. For this reason it was decided not to include any further allowance for the dilution of fuel-grade coal.

5.2.6.2 Mining Loss

Mining losses of the coal reserves could occur from the following day-by-day operating situations:

(1) Coal lost when waste is removed at coal/waste interfaces;

(2) Errors in dispatching coal to waste dumps;

(3) Degrading of coal during ground sloughs to such an extent that it would be dispatched to the waste dumps;

(4) Losses from dusting of fine coal and spillages during transportation.

When estimates are made of these potential losses of coal, they are found to constitute less than half of one per cent of the total coal mined. This parameter was therefore considered insignificant and not included in the preliminary engineering design.

5.2.7 Selective Mining

5.2.7.1 Definition

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. In spite of its erratic history, it is still possible to identify seven separate occurrences of flooding within the C-zone. 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 m to 10 m, 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 m, are more readily mined with the coal. However, while this simplifies the mining process, it reduces the quality of the coal fed to the boilers, which are subjected to additional wear and produce larger quantities of ash to be disposed of.

The separation of these smaller partings from the coal would improve the boiler-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 ½ m to 5 m. 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.

The results of these studies were reviewed from a practical and economic viewpoint. The two main conclusions drawn from this review were:

(1) The mining method employed would govern the degree of selective mining that could be effected;

(2) The cost of separating small waste bands (½-1 m) would be high and reduce equipment productivity significantly.

5.2.7.2 Selective Mining Methods

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 m^3 bucket was able to segregate partings 1 m 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 m^3 buckets at other mining operations indicates that the wrist-like digging action of these machines will permit selective mining of partings 1.5 m to 2 m thick without reducing equipment productivity. The hydraulic shovels have also proved effective in digging hard, rocky materials that cable shovels are unable to cope with unless the materials are blasted. The digging action of the widely used mining cable shovels severely limits their effectiveness in selective mining. 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 m 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 m, depending on their position and attitude.

During operation, careful control must be exercised to ensure the success of selective mining. Closely spaced sample holes will be drilled ahead of mining, to permit local correlation of coal quality for short-term mine planning. This will be supplemented by detailed geological mapping of the exposed coal faces. Reject bands will be marked and face maps supplied to the shovel operators and their supervisors. These maps, together with the marked differences in the physical characteristics between the coal and waste, are expected to ensure the feasibility of selective mining. The results obtained will be monitored by a quality control group and by the product sampling and monitoring of the crushed product en route to the blending pile.

5.2.7.3 Selective Mining Evaluation

Several comparative evaluations have been made of the results obtained by selective and non-selective mining. Similar results were obtained in each case.

The results for a trial 35-year pit applying a 9.3 MJ/kg cut-off grade are:

	2 m Selective Mining	Non-selective Mining
Coal-tonnes (Mt)	347	365
HHV - MJ/kg	18.06	17.12
Ash-content - %	33.47	36.20

These results show that with selective mining:

(1) The total heat content supplied to the boilers is a fraction of a per cent higher;

(2) The HHV is 5.5% higher;

(3) The total tonnes of ash fed to the boilers is reduced from 132 million tonnes to 116 million tonnes.

From these facts it is concluded that selective mining is beneficial because: it provides for good resource utilization; improves boiler operating efficiency; and will improve boiler reliability due to the significant decline in the quantity of ash handled. These benefits can be obtained without a significant increase in mining costs.

Recent developments in 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.

5.3 MINING METHODS

5.3.1 Review of Alternatives

The following six alternative mining systems were identified:

(1) Shovel/truck;

(2) Shovel/truck/conveyor;

(3) Shovel/conveyor;

(4) Bucketwheel excavator/conveyor;

(5) Continuous excavator/truck and/or conveyor;

(6) Dragline/truck and/or conveyor.

From this list two systems were determined to be the most practical: The Bucketwheel Excavator/Conveyor System and the Shovel/Truck Conveyor System.

North American Mining Consultants (NAMCO) were retained to assess the feasibility of the Bucketwheel Excavator and Conveyor System for developing the deposit, while Cominco-Monenco Joint Venture (CMJV) carried out similar studies with the Shovel/Truck/Conveyor System.

In order to deliver a consistent fuel quality (heating value and sulphur) to the powerplant, the pit must be deepened rapidly during pre-production and the first 10 years of production. As a result, coal and waste mining will be carried out simultaneously on a number of working benches. The economic advantages of employing the Bucketwheel Excavator System in this type of operation are therefore not realized, and this system only becomes a practical alternative when most of the pit expansion occurs laterally.

Because of the minimal affect on the project cost, it was decided not to consider a change in the mining system from the Shovel/Truck/Conveyor System to the Bucketwheel Excavator System during the life of the project. It was also felt that this evaluation could better be made after some experience had been acquired with the recommended Shovel/Truck/Conveyor System. Since the recommended system has in-pit conveyors, and the operating life of the major mining equipment is 10 years or less, it should be possible to have a smooth transition to a Bucketwheel Excavator/Conveyor System if such a change were found to be advantageous.

5.3.2 The Shovel/Truck/Conveyor System

As described in Section 5.4 ("Pit Design and Production Scheduling"), a series of incremental pits and a 35-year pit were developed by computer using the Dipper System, based mainly on economics. From these computer-generated data, and incorporating the design criteria described in Section 5.2, practical, operational pit plans were designed.

The selected scheme is a Shovel/Truck System in combination with an in-pit conveyor system. It includes a coal screening and crushing plant at the Northern end of the pit, and a coal stockpiling and blending facility from which blended coal is reclaimed and transported by overland conveyor to the powerplant. The low-grade coal (with a heating value ranging from 7.0 to 9.3 MJ/kg) is treated in a dry beneficiation plant with a capacity of 1,000 t/h. Beneficiation plant rejects are mixed with the mine waste in the Waste-handling System, while upgraded coal is conveyed to the blending facility.

Mine waste is transported by conveyor belts to Houth Meadows and Medicine Creek waste dumps and deposited by spreaders. Houth Meadows will be started in Year -1 by trucks and developed by spreaders in Year 1. Medicine Creek will be started by trucks in Year 12 and developed by spreaders in Year 15. Neither of the dumps will have been built to maximum capacity at the end of Year 35.

The mine service facilities are located at the Northern end of the mine and South of Indian Reserve IR-1.

All the foregoing are shown in Figure 3-3 (Detailed Site Layout Map).

The 35-Year Pit

Figure 5-17 shows the 35-year pit. It covers an area of about 5.4 km^2. The pit bottom is at elevation 662.5 m.

Significant features in the pit include:

1. Northern Exit

The mine plan developed shows multiple road access to the various benches. The in-pit conveyor and the principal roads exit to the North end of the pit.

Studies conducted to bring waste to Medicine Creek from a Southern exit showed that a causeway from the pit to the dump would interfere with the access road to the Upper Hat Creek Valley and, more importantly, with the Hat Creek Diversion. Long, large-diameter culverts under this causeway would need to be installed to make this scheme possible.

It was confirmed that the in-pit conveyor should exit to the North. The natural saddle of footwall waste between the two synclines provides an ideal location for the conveyor which would not entail additional mining of waste. An in-pit conveyor belt exiting South would require more waste to be mined to allow for an acceptable slope.

2. In-Pit Conveyors

A four-line, 1,500 m in-pit conveyor-belt system extends from 895 m elevation at the surface to 702 m elevation. A study of the number of mining benches and the corresponding hauling distances to the various delivery points confirms that the In-pit Conveyor System is essential for a more efficient hauling operation and the reduction of haulage costs.

3. Dump Stations

Three dump stations are located adjacent to the in-pit conveyor to which coal, low-grade coal, and waste material are delivered. The locations of the dump station were governed by the material distribution by bench and their corresponding average hauling distances.

Dump Station No. 1, at 887.5 m elevation, will handle material mainly from 1,045 m to 865 m benches inclusive; Dump Station No. 2, at 827.5 m elevation, material from 850 m to 775 m benches inclusive; Dump Station No. 3, at 722.5 m elevation, material from 760 m to 670 m benches inclusive.

These dump stations will be developed as mining progresses in depth and when hauling to existing pockets is neither practical nor economic. Based on computer-generated incremental pits and a study comparing haulage costs to the dump stations, the following schedule of installation was developed:

Dump Station No. 1 - Operational Year -1

Dump Station No. 2 - Operational Year 8

Dump Station No. 3 - Operational Year 20

The dump station design and road network complement each other. Material can be delivered and dumped either from the Eastern or Western sections of the pit. This feature simplifies hauling operations and reduces hauling costs.

4. Mine Roads

Mine roads vary in width from 25 m in coal, sand, and gravel to 40 m in the Medicine Creek and Coldwater formations. A 60 m-wide berm is provided adjacent to the active slide. This wide berm provides ample room for periodic clearing operations should soil creep occur.

The road network provides access at a minimum of two locations to each bench, usually on opposite sides of the pit. This operational feature will be important for two reasons: (1) it reduces hauling distances to the dump stations; and (2) it will provide better assurance of continuous mining should localized wall failures occur. The road network is designed to allow pit expansion after 35 years.

Three major berms are located at elevations 902.5 m, 827.5 m, and 722.5 m to coincide with the dump station elevations (902.5 m berm is one bench higher than Dump Station No. 1). During mining operations, access to the mining benches will be from these berms, which are essentially extensions of the dump stations.

5. The Pit Bottom

The pit bottom at elevation 662.5 m measures 700 m x 450 m at the widest dimensions and has an area of about 263,000 m^3. A secondary pit bottom, one kilometre long and 100 m wide, is at elevation 677.5 m. Both of these bench bottoms are totally in coal which has a wide range of heating value. Some eight million tonnes of coal can be mined by deepening the pit bottom without additional waste removal. This coal provides assurance that the designed pit can meet the powerplant requirements over the life of the project.

Mine Development

Mining is initiated on six benches west of Hat Creek and bounded by co-ordinates 5625200 N in the North, 5624700 N in the South and 598400 E in the West. The pre-production pit is connected to Houth Meadows Dump by a 2.5 km temporary surface road at 880 m elevation. Prior to the installation of the conveyor system all construction materials will be used for road construction. Unsuitable materials are hauled by truck to Houth Meadows and dumped to 880 m elevation.

Excavation for Dump Station No. 1 will be started during pre-production in order to have the station operational in Year -1. The reasons for starting the first dump station early are threefold:

(1) To reduce haulage distances from the pit to the dumps;

(2) To assure the supply of approximately one million tonnes of coal to commission the powerplant in Year -1 (truck haulage to the power-plant for this quantity is impractical);

(3) To have a source of sand and gravel for construction in and around the mine areas. Approximately three million bank cubic metres of sand and gravel will be mined from Dump Station No. 1 during pre-production.

A temporary 1.5 km surface road at elevation 887.5 m connects Dump Station No. 1 with the pre-production pit.

Figures 5-12 to 5-17 show pit development in various stages.

The mining sequence adopted shows that, during the early years, production is concentrated along the Eastern limb of the main syncline which has a wide range of calorific value. Mining of the thick sand and gravel beds overlying the North-East sector of the deposit is limited at this time. By developing the pit this way during the early years, the average heating value is maintained and a low stripping ratio is achieved.

In later years, as the mine develops in depth, the lower quality coal in the Western limb is exposed on the upper benches. By this time, sufficient sand and gravel will have been removed to allow mining of the higher grade coal in the Eastern syncline. This mining strategy ensures that both the average coal quality and the stripping ratio will be maintained at reasonable levels.

The pre-production pit starts almost at the centre of the deposit and expands progressively towards the final wall. This development sequence will provide ample time to observe pit walls and prepare adjustments in pit design if required. The road network in the incremental pits is designed to provide enough flexibility to accommodate a revision of the pit design.

In the incremental pits, the coal benches were laid out so that coal could be mined from them at any time without having to mine the bench above. This ensures that a wide variety of coal quality will be available for blending. In sections located in waste, three to four benches were grouped together with the uppermost bench minable. Each succeeding bench becomes minable as the bench above it is mined out. This scheme was adopted to reduce waste stripping. Figure 5-3 ("Pit Slopes") shows the systems described.

Temporary roads between benches are limited. The intention is to construct and use the final haul roads as soon as it is practicable.

The excavation and installation of Dump Stations No. 2 and No. 3 is governed by the mining schedule of the various benches. This results in material being hauled from the two benches above and the three benches below the dump pocket elevation. The haul roads are designed accordingly.

5.4 PIT DESIGN AND PRODUCTION SCHEDULING

Pit design and production scheduling were performed, making extensive use of computer software developed by Mintec Inc., supported by manual mine planning techniques. This section describes the methods employed to perform the work starting from the Variable Block Model (VBM) developed earlier (described in Section 4.6) to the completion of the production schedule.

5.4.1 Planning Data

A set of cross-sections and bench plans for the coal deposit were produced to provide a clear picture of the structure and the spatial distribution of coal quality.

The preparation of the cross-sections from the Variable Block Model was straightforward. Each cross-section in the model was computer-plotted showing the geological sub-zones (see Figure 5-9) and the reserve blocks together with the tonnage and heating value for each block.

The preparation of the bench plans was more complex, because the VBM was constructed on cross-sections. The plans were ultimately produced by manually adjusting the computer plots. The adjustments required were primarily in areas of structural complexity and where sub-zones terminated between sections. The bench plans were produced for the mid-points of 27 benches at 15 m intervals. Each sub-zone block was annotated with an identification number, its coal tonnage, heating value, and waste quantity. These plans and sections were colour coded by heating value range for easier use in mine planning (see Figure 5-10).

5.4.2 The Dipper System

The Dipper System is designed to assist the mining engineer to develop mine plans and production schedules quickly. This permits the evaluation of many alternative mining sequences in the time it takes to develop a single plan manually and results in a more practical and economic mine plan.

The Dipper System is designed to operate using a rectangular block model of the deposit. The blocks used for the evaluation of the Hat Creek Coal Deposit are 50 m square in plan and 15 m high. A block of coal this size represents approximately 55,000 t. Smaller blocks can be used to refine the pit design and production schedule, where warranted, by closely spaced data, at the expense of increased computer time. The model defines the mining area using 196,000 blocks. For each block the waste volume, coal tonnes, and heating value were calculated from the Variable Block Model. These calculations are made every 10 m, and the resulting composite values accurately reflect the geological interpretation and quality data for each block. The surface topography was digitized and input to the Dipper Model.

To permit the evaluation of alternatives, a value function is required. A gross value is assigned to each block based upon its total heat content. This gross value is reduced to a net value by the deduction of variable assigned overhead and mining costs for use in pit design.

The mining geometry in Dipper is simulated by a series of inverted, truncated cones. Each cone is defined by the base radius, which is equivalent to half the minimum mining width, and the slope, which can be varied in up to nine specified directions to reflect varying pit slopes. The centre of each cone coincides with the centre of a block. Any block whose centre is within the cone generated is included in the volume mined.

The design of the pit is controlled by the requirement to meet certain criteria. Typical parameters that can be varied in applying the Dipper System include:

(1) Mining cost;

(2) Minimum average heating value for each cone;

(3) Maximum stripping ratio for each cone;

(4) Required coal tonnage in a pit increment.

When these criteria have been specified, the pit limits are determined by evaluating the cones within the boundaries defined by the engineer. The parameters of all blocks contained by a cone are accumulated and the results tested against the criteria. If the criteria are met, the cone is mined, and the process is repeated for another cone until the required tonnage is mined or no further cones meet the criteria.

Data displays available include:

(1) Printer plotted symbol maps of the deposit by section and bench;

(2) Symbol maps showing the pit limits on each bench;

(3) Tabulated summaries of reserves.

5.4.3 Pit Design

The Dipper System's pit design capabilities were tested by developing a sequence of incremental pits to produce 347 million tonnes at an average heating value of 18.0 MJ/kg. The final pit bottom had moved about 200 m South compared with earlier manually designed pits; the stripping ratio was significantly reduced in the early years, with only a small improvement in the overall stripping ratio. The Dipper results were checked against cross-sections, bench plans, and previous designs in order to evaluate the differences. After checking, it was concluded that the results of the test were reasonable and that the system should be adopted for the pit design work.

Further tests were performed in order to remove concerns about the validity of the costs assigned and also to try to improve the coal quality in the first five years of operation. The cost parameters were varied in a series of runs, and it was found that the relative economics provided a sound basis for the design of a sequence of "best" pits. The coal quality improvement tests demonstrated that the objective could be achieved, but would result in an extended period of unacceptably low quality fuel later. This was a valuable exercise in demonstrating the speed and flexibility of the Dipper System.

In applying the system to the design of the overall pit slope angles were established in four directions: East 20^{o}, South and West 19^{o}, and North 15^{o} (to allow for the conveyor ramp - see Figure 5-11). These overall slopes were determined from manually designed pits, which reflected the geotechnical constraints and incorporated mine haul roads. In the initial runs the minimum average heating value for each cone was set at 17.0 MJ/kg and the maximum stripping ratio at 2.0. In subsequent runs these parameters were varied to force desired improvements in the plan.

The required coal tonnage in a pit increment was set at approximately one year's production for the first 10 years, and in five-year segments thereafter. In designing the interim pits, a flatter working slope (16^{o} except to the North) was used.

The pit is designed one increment at a time until a final pit is reached which provides sufficient tonnage of an acceptable quality. When a satisfactory final pit was established, a pit design was prepared manually to incorporate roads, crusher stations, and conveyorways. The interim pits were then re-worked to modify the quality or stripping ratio. In this fine tuning process, the pit design can also be forced to excavate material in a particular area to permit installation of required facilities.

The results for the 16 incremental pits developed are presented in Table 5-4. In arriving at this final series of pits, a total of 92 increments were examined to ensure the production of a consistent quality of fuel and to reduce the fluctuations in the stripping ratio.

5.4.4 Production Scheduling

At this stage of a project production scheduling would not normally be carried beyond the stage reached with the completion of the sequence of interim pits. However, in the case of the Hat Creek Project it was considered necessary to ensure that the larger, five-year increments did not include extended periods where only unacceptable quality fuel was available.

Working within the incremental design pits, production scheduling selects the coal to be mined in a given time period. This is accomplished by examining the pit bench by bench from the top down, removing the coal until the production requirements are met, and identifying the waste that must be removed to permit mining that coal. This process is repeated for succeeding years until all the coal in that pit increment is mined. Scheduling then continues from the next increment and progresses until the pit is mined out.

This preliminary production schedule showed a wide fluctuation in the quantities of waste removal for each year. To ensure a practical mining operation that makes efficient use of the equipment available, these fluctuations must be smoothed out. This smoothing was achieved by establishing the annual waste production capacity and forcing advanced waste removal in low stripping years. This procedure was effective, and a practical production schedule was produced that maintained an acceptable quality of fuel and balanced material quantities over the life of the project.

Initially, the production schedules were developed based on an annual coal tonnage requirement at an average quality. The resulting schedule showed that the total heat content of the coal produced in a given year deviated from the powerplant requirements. To overcome this problem the production was rescheduled to deliver the required total heat content.

The Adjusted Production Schedule (Table 5-5) shows the final production schedule that was produced by this process. A final manual adjustment was made to this schedule to incorporate waste removed outside the pit limits for the development of facilities (Table 5-6).

5.5 WASTE DUMPS AND EMBANKMENTS

5.5.1 General

The total amount of waste material mined from the pit over its 35-year lifespan would be 426.8 million bank m^3. Two areas have been selected where the waste could be safely and economically dumped: (1) Houth Meadows, at the North-West rim of the pit, with a maximum capacity of 542 million m^3 or about 439 million bank m^3; (2) Medicine Creek, about one kilometre South-East of the pit, with a capacity of 257 million m^3, with the crest at 1,130 m elevation. The potential exists for Medicine Creek to be raised to 1,200 m elevation which would increase the capacity by another 310 million m^3 for mine waste and ash.

The selection was based on proximity, capacity, geotechnical characteristics, and topographical and geological features which render both dumps capable of meeting the most stringent requirements. Another significant factor was the possibility of expanding the 35-year pit to mine out the No. 1 Deposit and starting to mine the No. 2 Deposit to the South.

Comprehensive studies were undertaken by Golder Associates, geotechnical consultants, and their recommendations incorporated into the design of the dumps (see Section 5.5.2). B.C. Hydro's own geotechnical engineers have reviewed the consultants' work, and have issued a report "Memorandum on Proposed Waste Disposal Embankment Studies", dated October 1979. Section 7 of their report, Conclusions and Recommendations for Final Design Studies, is shown in Section 5.5.6.

5.5.2 Geotechnical Constraints and Parameters

5.5.2.1 Material Parameters

Tests have led to establishing two general categories of waste:

(1) Unstable and very weak bentonitic claystones 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.

5.5.2.2 Parameters of Waste Dumps and Embankments

Geotechnical tests and studies were concerned with three main issues related to dump stability:

(1) The stability of retained waste;

(2) The stability of retaining embankments and their foundations;

(3) The gross interaction of waste dumps and pit slope excavations.

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 with 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 is far enough from the 35-year pit but would be within 600 m from the pit rim of an ultimate, or total resource, pit. Investigations were conducted on the basis of the total resource pit from the CMJV report rather than the 35-year 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.

5.5.3 Construction and Development

Although both Houth Meadows and Medicine Creek dumps at maximum capacity can accommodate the total 35-year mine waste, it is recommended that neither dump should be built to capacity until more data is available. Material characteristics relating to swell factors are uncertain and can only be ascertained during actual operations. Room for additional waste will also be required for any expansion of the pit. Neither the Southern end nor the bottom of the No. 1 Deposit will have been mined out after Year 35.

Of the two dumps, Houth Meadows will be the first to be constructed. It will be developed at a full rate from Year 1 to Year 14 by two conveyor-spreader systems, each working in 35-m lifts. From Year 12 to Year 14, haulage trucks will lay the foundations of the Medicine Creek Dump in preparation for one of the conveyor-spreader systems which will be transferred from Houth Meadows. From Year 15 onwards, both Houth Meadows and Medicine Creek dumps will be constructed concurrently. Figure 5-18 shows the different stages in the development of each dump. This sequence of dump development is geared not only to the most efficient exploitation of the No. 1 Deposit during the 35-year project life, but takes into account the possible expansion of the No. 1 Deposit and/or future mining of the No. 2 Deposit. It also allows ample time to study the effects of accumulating large amounts of waste in the dumps.

The development sequence for each 35-m lift is divided into three phases:

5.5.3.1 Construction of Access Roads and Initial Conveyor Pads

Conveyor pads will be constructed at the far end of the dump from the retaining embankment at an elevation 20 m above the existing dump surface in that area. Access roads and conveyor pads will be constructed on contour using sidehill cuts to the extent practical. Conveyor pads will be 40 m wide, which is sufficient for the installation of the shiftable conveyors and initial operation of the spreader.

The access roads and conveyor pads will be built with glacial till, sand, or gravel. Road construction equipment: front-end loaders, 32-t trucks, dozers, graders, and compactors, will be used for this job. This equipment will also be used for filling areas inaccessible to the spreaders.

5.5.3.2 Dumping General Waste

The spreader will start dumping waste from the initial conveyor pad. 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.

This system is illustrated in Figure 8-7.

5.5.3.3 Construction of Embankments

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 face of the embankment must be trimmed to the designed 2.5:1 slope ready for revegetation. The shiftable conveyor system is dismantled and re-erected on a new conveyor pad constructed at the planned elevation of the next lift.

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.

Tables 5-7 and 5-8 show the capacity by lift of the Houth Meadows and Medicine Creek dumps and the construction schedules.

5.5.4 The Houth Meadows Waste Dump

Development of the Houth Meadows Dump will start in about Year -1 after the causeway for the Main Transfer Conveyor has been built. Prior to the construction of the dump, the base will be prepared by laying free-draining sand and gravel material for drainage and constructing a leachate collection facility at the toe of the embankment.

Waste from the pre-production pit, and sand and gravel from Dump Station No. 1, will be hauled by trucks. These will be used to build the dump to the 880 m elevation. In the meantime, the road construction equipment will be constructing the first transfer and shiftable conveyor pads at the 900 m elevation. Conveyor-spreader System No. 1 will be installed at this elevation so that waste can be dumped to the first 35-m lift (between the 880 m and the 915 m elevation) in Year 1.

The second transfer and shiftable conveyor pads at the 935 m elevation will be built after the 880-915 m lift has advanced far enough to allow space for construction. Conveyor-spreader System No. 2 will be installed at the 935 m elevation and dumping of waste to the second lift (between the 915 m and the 950 m elevation) will commence in Year 2. Both of the conveyor spreader systems will then work concurrently, in parallel.

Following the two bottom 35-m lifts, the schedule for the succeeding lifts is:

(1) Construct transfer and shiftable conveyor pad at 970 m elevation in Year 5; relocate Conveyor-spreader System No. 1 from the 900 m elevation; commence waste dumping in Year 6. Upon completion of this lift in about Year 14, the conveyor and spreader will be transferred to Medicine Creek;

(2) Construct a transfer and shiftable conveyor pad at the 1,005 m elevation in Year 8; relocate Conveyor-spreader System No. 2 from the 935 m elevation and commence waste dumping in Year 9. The 985-1,020 m lift will be completed in about Year 22;

(3) Construct a transfer and shiftable conveyor pad at the 1,040 m elevation in Year 22; relocate Conveyor-spreader System No. 2 from the 1,005 m elevation, commence waste dumping in Year 23 and carry on the operation until Year 35. A total of 305 million m^3 will be dumped in Houth Meadows from Years -2 to 35. A further 134 million m^3 could be placed in this area if required.

Houth Meadows is designed with the ultimate embankment crest at the 1,005 m elevation. The major embankment runs from the hill by the Hat Creek road - Lillooet Highway junction to the NE-SW-trending conglomerate ridge. Three minor embankments are located running in an East-West direction and are required to prevent waste from flowing on to the Lillooet Highway. As recommended by the geotechnical consultants, the dumps are designed with a 2.5 horizontal to 1 vertical on the outside face and 1:1 on the inside face of the embankment. Figure 5-21 shows the waste dumps slopes.

The retained waste dump is designed sloping at a 5% grade from the crest of the embankment at the 1,005 m elevation to the Westernmost limits at the 1,150 m elevation. The surface area of the dump covers approximately 580 ha at maximum capacity.

Surface water in the dump area will be collected by a suitable drainage system around the perimeter and surface runoff will ultimately be collected in the settling ponds. Figure 5-19 is a detailed drawing of the Houth Meadows Waste Dump.

5.5.5 The Medicine Creek Waste Dump

Development of Medicine Creek Waste Dump will commence in Year 12, three years before the installation of the Conveyor-spreader System. Contractors will prepare the base of the dump by laying free-draining sand and gravel material for drainage, and will build the narrow portion of the dump up to the 1,040 m elevation by trucks. Approximately 9.4 million bank m^3 will be hauled by the contractors over a temporary road. By the end of Year 14, construction work should have been completed. The dump will then be built using Conveyor-spreader System No. 1, which will be transferred from the Houth Meadows Dump.

The dump development sequence is as follows:

(1) Truck construction: base of dump to the 1,040 m elevation from Year 12 to Year 14, by contractor, using haulage trucks;

(2) Construct transfer and shiftable conveyor pads at the 1,060 m elevation in Year 14; relocate Conveyor-spreader System No. 1 from Houth Meadows Dump in Year 15; dumping of waste 1,040-1,075 m lift from Year 15 to Year 18;

(3) Construct transfer and shiftable conveyor pads at the 1,095 m elevation in Year 17; relocate Conveyor-spreader System No. 1 from the 1,060 m elevation in Year 18; build 1,075-1,110 m lift from Year 18 to Year 26;

(4) Construct transfer and shiftable conveyor pads at the 1,130 m elevation in Year 25; relocate Conveyor-spreader System No. 1 from the 1,095 m elevation in Year 26; build 1,110-1,145 m lift from Year 26 to Year 35.

A total of 113 million bank m^3 of waste will be dumped in Medicine Creek from Year 15 to Year 35. About 29 million m^3 capacity remains below the 1,130 m crest.

From Year -1 to Year 14, while dumping of waste will be in Houth Meadows, ash from the powerplant will be deposited at Upper Medicine Creek (downstream of the water reservoir dam). Ash deposition will progress downstream while dumping of waste will progress upstream. At about Year 20 or Year 21, the two disposal systems will meet. At this time, waste material will be dumped at a slope of 2.5 horizontal to 1 vertical at the interface between the waste and the ash. By doing so, ash will overlay the waste as both are built up. Figure 5-20 is a detailed drawing of the Medicine Creek Dump.

Following the geotechnical consultants' recommendations, the retaining embankment is designed at 2.5 horizontal to 1 vertical at the outside face and 1:1 in the inside face. The retained waste slopes at a 5% grade from the crest of the embankment to the interface with the ash, after which the latter slopes at 1% up to the water reservoir dam (see section detail Figure 5-20). The Northern side of the waste dump forms a V-cut with the hillside to permit access to the reservoir overflow outlet conduit which carries any overflow from the reservoir down to the Hat Creek Diversion Canal.

Canals around the perimeter of the dump will be installed to collect surface runoff. Runoff from the dump surface will be diverted to the settling ponds West of the embankment.

5.5.6 Conclusions and Recommendations Relating to Waste Disposal Embankment Studies

The geotechnical consultants' studies and the recommended design basis for the waste disposal embankments were reviewed by the B.C. Hydro Hydro-electric Generation Projects Division. They presented the following conclusions and recommendations in their design memorandum:

5.5.6.1 Conclusions

It is concluded that the studies are complete and adequate for the preliminary design stage. The design for the retained waste material disposal and the stability of the retaining embankment and its foundation have an acceptable factor of safety for static conditions. The analysis for interaction with total resource pit slope is reasonable.

5.5.6.2 Recommendations

For final design studies it is recommended that:

(1) An exploration program be carried out at the proposed Medicine Creek retaining embankment to confirm either that siltstone and claystone do not exist in the foundation, or that they do not affect the stability of the retaining embankment;

(2) The stability of waste dump and pit slope of the Houth Meadows Dump be studied further, if the total resource pit scheme is to be adopted;

(3) Tests be carried out to assess the proposed method for compaction (i.e. by impact of gravels falling from conveyor belts) of the embankment fills;

(4) The embankment and waste mass be analyzed for seismic stability and that the sands in embankment foundation be evaluated for liquefaction potential.

TABLE 5-1

DESCRIPTION OF SURFICIAL MATERIALS

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.		

TABLE 5-2

DESCRIPTION OF ROCK MATERIALS

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

TABLE 5-3

HYDRAULIC CONDUCTIVITY RESULTS

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)	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.

TABLE 5-4

INCREMENTAL DESIGN PIT QUANTITIES

** 2M SELECTIVITY -- PIT X2P/LAM **

SUMMARY OF MINING INCREMENTS

--PIT-NAME----	CUMULATIVE			INCREMENT			
	ORE TONS	HHV	S.R.	ORE TONS	HHV	S.R.	CUTOFF
R2PL1.DAT	1537.	18.45	2.16	1537.	18.45	2.16	9.30
R2PA2.DAT	4672.	18.99	1.43	3135.	19.25	1.07	9.30
R2PA3.DAT	9990.	18.79	1.27	5318.	18.62	1.12	9.30
R2PA4.DAT	18210.	18.63	1.29	8220.	18.44	1.32	9.30
R2PA5.DAT	29772.	18.62	1.33	11562.	18.60	1.40	9.30
R2PL6.DAT	43545.	18.53	1.30	13773.	18.35	1.22	9.30
R2PL7.DAT	58407.	18.46	1.27	14862.	18.24	1.17	9.30
R2PL8.DAT	73116.	18.30	1.28	14709.	17.67	1.32	9.30
R2PL9.DAT	84175.	18.23	1.30	11059.	17.74	1.42	9.30
R2PL0.DAT	95624.	18.15	1.23	11449.	17.59	0.75	9.30
R2PM1.DAT	109441.	18.13	1.19	13817.	17.98	0.93	9.30
R2PM2.DAT	163209.	17.91	1.23	53768.	17.45	1.29	9.30
R2PM3.DAT	213231.	17.99	1.12	50022.	18.26	0.78	9.30
R2PM4.DAT	254804.	18.01	1.14	41573.	18.12	1.21	9.30
R2PM5.DAT	307069.	17.97	1.18	52265.	17.78	1.38	9.30
R2PM6.DAT	335646.	18.09	1.25	28577.	19.30	1.99	9.30

TABLE 5-5

ADJUSTED PRODUCTION SCHEDULE

1 ** 2M SELECTIVITY -- PIT X2PL/M ** PRODUCTION BASED ON TONS * HHV **

YEAR	YEARLY SCHEDULE				CUMULATIVE SCHEDULE			
	MILL FEED	GRADE	WASTE	S.R.	MILL FEED	GRADE	WASTE	S.R.
1	0.	0.000	0.	0.000 *	0.	0.000	0.	0.000 *
2	1139.	17.597	3697.	3.246 *	1139.	17.597	3697.	3.246 *
3	2950.	19.292	4375.	1.483 *	4089.	18.820	8073.	1.974 *
4	4759.	18.155	7474.	1.570 *	8848.	18.462	15546.	1.757 *
5	7371.	17.994	10720.	1.454 *	16220.	18.249	26266.	1.619 *
6	9249.	18.455	13678.	1.479 *	25469.	18.324	39944.	1.568 *
7	10684.	17.914	16082.	1.505 *	36153.	18.203	56026.	1.550 *
8	10452.	18.762	14973.	1.433 *	46605.	18.328	70999.	1.523 *
9	10458.	18.750	15301.	1.463 •	57063.	18.406	86300.	1.512 *
10	11555.	16.970	16827.	1.456 *	68618.	18.164	103127.	1.503 *
11	10842.	18.087	18016.	1.662 *	79460.	18.153	121142.	1.525 *
12	11172.	17.553	14752.	1.320 •	90631.	18.079	135894.	1.499 *
13	11535.	17.000	20503.	1.777 *	102166.	17.957	156397.	1.531 *
14	10602.	18.496	17171.	1.620 *	112768.	18.008	173568.	1.539 *
15	11517.	17.026	18848.	1.637 *	124286.	17.917	192416.	1.548 *
16	11387.	17.221	6312.	0.554 •	135673.	17.859	198728.	1.465 *
17	11081.	17.696	15212.	1.373 *	146753.	17.846	213940.	1.458 *
18	10047.	18.126	10371.	1.032 *	156800.	17.864	224311.	1.431 *
19	10215.	17.827	14222.	1.392 *	167015.	17.862	238532.	1.428 *
20	10557.	17.250	14166.	1.342 *	177572.	17.826	252699.	1.423 *
21	10212.	17.833	9839.	0.964 *	187784.	17.826	262538.	1.398 •
22	9961.	18.283	10559.	1.060 *	197745.	17.849	273097.	1.381 *
23	9813.	18.557	12892.	1.314 •	207558.	17.883	285988.	1.378 *
24	9841.	18.506	16344.	1.661 *	217399.	17.911	302332.	1.391 *
25	9914.	18.368	12140.	1.224 *	227313.	17.931	314472.	1.383 •
26	10184.	17.883	9859.	0.968 *	237496.	17.929	324331.	1.366 *
27	10068.	18.089	9240.	0.918 •	247564.	17.935	333571.	1.347 *
28	8284.	18.591	7335.	0.886 *	255848.	17.956	340907.	1.332 *
29	8478.	18.164	14509.	1.711 *	264326.	17.963	355416.	1.345 *
30	8395.	18.345	8059.	0.960 *	272721.	17.975	363475.	1.333 *
31	8561.	17.988	11103.	1.297 *	281283.	17.975	374577.	1.332 *
32	8584.	17.941	12145.	1.415 *	289867.	17.974	386722.	1.334 *
33	8653.	17.797	6972.	0.806 *	298520.	17.969	393694.	1.319 *
34	8508.	18.101	10222.	1.201 *	307028.	17.973	403915.	1.316 *
35	8247.	18.675	9846.	1.194 *	315275.	17.991	413761.	1.312 *
36	8053.	19.123	1871.	0.232 *	323328.	18.019	415632.	1.285 *
37	7622.	20.205	1960.	0.257 *	330950.	18.070	417592.	1.262 *

YEAR 1 IS PREPRODUCTION AND IS NOT INC. IN CUMULATIVE

MATERIALS MINED Quantities in (10^6)		PRE-PRODUCTION YEARS				PRODUCTION YEARS											
		−4	−3	−2	−1	1	2	3	4	5	6	7	8	9	10	11	12
COAL MINED (Tonnes)	Annual			0	1·14	2·95	4·76	7·35	9·23	10·46	10·60	10·52	11·49	10·69	11·37	11·17	10·86
	Cumul.			0	1·14	4·09	8·85	16·20	25·43	35·88	46·48	56·99	68·49	79·18	90·55	101·71	112·57
MJ / Kg (Dry Basis)	Annual			0	17·6	19·3	18·2	18·1	18·5	18·3	18·5	18·6	17·1	18·3	17·3	17·6	18·1
	Cumul.			0	17·6	18·8	18·5	18·3	18·4	18·3	18·4	18·4	18·2	18·2	18·1	18·0	18·0
ASH CONTENT (%, Dry Basis)	Annual					33·52	33·52	33·52	33·52	33·52	32·80	32·80	32·80	32·80	32·80	35·31	35·31
SULPHUR CONTENT (%, Dry Basis)	Annual				0·55	0·52	0·55	0·56	0·53	0·56	0·53	0·53	0·53	0·53	0·53	0·54	0·54
HEAT UNITS DELIVERED IN MJ x 10^9 (Based on 23·5 % Moisture and Coal Cut-Off at 9·3 MJ/Kg)	Annual				15·35	43·56	66·27	101·77	130·63	146·43	150·01	149·69	150·31	149·65	150·48	150·39	150·37
	Cumul.				15·35	58·91	125·18	226·95	257·58	504·01	654·03	803·72	954·03	1103·68	1254·16	1404·55	1554·92
COAL Fuel above cut-off of 9·3 mj/kg (bank cubic metres)	Annual				0·76	1·97	3·19	4·93	6·19	7·02	7·11	7·06	7·71	7·17	7·63	7·49	7·28
	Cumul.				0·76	2·74	5·93	10·87	17·06	24·08	31·19	38·24	45·95	53·14	60·77	68·26	75·55
WASTE Above Bedrock	Annual			2·25	2·78	3·29	5·60	8·04	11·72	12·43	10·43	10·50	10·63	11·16	11·18	11·13	10·75
WASTE Bedrock	Annual			0·75	0·92	1·09	1·87	2·68	3·90	4·14	6·39	6·44	6·51	6·84	6·85	7·12	6·87
SUB – TOTAL WASTE (bank cubic metres)	Annual			3·00	3·70	4·38	7·47	10·72	15·62	16·57	16·82	16·94	17·14	18·00	18·03	18·25	17·62
	Cumul.			3·00	6·70	11·08	18·55	29·27	44·89	61·46	78·28	95·22	112·36	130·36	148·39	166·64	184·26
TOTAL MATERIAL MINED (bank cubic metres)	Annual				4·46	6·35	10·66	15·65	21·81	23·59	23·93	24·00	24·85	25·17	25·66	25·74	24·90
	Cumul.				7·46	13·82	24·48	40·14	61·95	85·54	109·47	133·46	158·32	183·50	209·16	234·90	259·81
STRIP RATIO bank cubic metres waste / tonnes of coal mined	Annual				3·25	1·48	1·57	1·46	1·69	1·58	1·59	1·61	1·49	1·68	1·59	1·63	1·62
	Cumul.				5·88	2·71	2·10	1·81	1·77	1·71	1·68	1·67	1·64	1·65	1·64	1·64	1·64

MATERIALS MINED Quantities in (10^6)		PRODUCTION YEARS											
		13	14	15	16	17	18	19	20	21	22	23	24
COAL MINED (Tonnes)	Annual	11·64	11·40	11·12	10·06	10·06	10·67	10·15	9·90	9·66	9·57	10·40	10·03
	Cumul.	124·21	135·61	146·73	156·79	166·85	177·52	187·67	197·56	207·22	216·79	227·19	237·21
MJ / Kg (Dry Basis)	Annual	17·0	17·2	17·6	18·0	18·1	17·1	17·9	18·4	18·9	19·0	17·5	18·2
	Cumul.	17·9	17·9	17·8	17·9	17·9	17·8	17·8	17·9	17·9	18·0	17·9	18·0
ASH CONTENT (%, Dry Basis)	Annual	35·31	35·31	35·31	33·90	33·90	33·90	33·90	33·90	32·77	32·77	32·77	32·77
SULPHUR CONTENT (%, Dry Basis)	Annual	0·54	0·54	0·54	0·50	0·50	0·50	0·50	0·50	0·48	0·48	0·48	0·48
HEAT UNITS DELIVERED IN MJ x 10^9 (Based on 23·5 % Moisture and Coal Cut-Off at 9·3 MJ/Kg)	Annual	149·60	150·00	149·72	138·53	139·30	139·58	138·99	139·35	139·67	139·10	139·23	139·65
	Cumul.	1704·52	1854·52	2004·24	2142·77	2282·07	2421·65	2560·64	2699·99	2839·66	2978·76	3117·99	3257·64
COAL Fuel above cut-off of 9·3 mj/kg (bank cubic metres)	Annual	7·81	7·65	7·46	6·75	6·75	7·16	6·81	6·64	6·48	6·42	6·98	6·73
	Cumul.	83·36	91·01	98·47	105·23	111·98	119·14	125·95	132·59	139·07	145·50	152·48	159·20
WASTE Above Bedrock	Annual	10·62	9·84	9·09	8·80	8·07	7·35	6·70	6·57	2·16	2·16	2·10	2·10
WASTE Bedrock	Annual	6·79	6·29	5·81	5·40	4·95	4·50	4·10	4·02	8·12	8·12	7·89	7·92
SUB – TOTAL WASTE (bank cubic metres)	Annual	17·41	16·13	14·90	14·20	13·02	11·85	10·80	10·59	10·28	10·28	9·99	10·02
	Cumul.	201·67	217·80	232·70	246·90	259·92	271·77	282·57	293·16	303·44	313·72	323·71	333·73
TOTAL MATERIAL MINED (bank cubic metres)	Annual	25·22	23·78	22·36	20·95	19·77	19·01	17·61	17·23	16·76	16·70	16·97	16·75
	Cumul.	285·03	308·81	331·17	352·13	371·90	390·91	408·52	425·75	442·51	459·22	476·19	492·93
STRIP RATIO bank cubic metres waste / tonnes of coal mined	Annual	1·50	1·41	1·34	1·41	1·29	1·11	1·06	1·07	1·06	1·07	0·96	1·00
	Cumul.	1·62	1·61	1·59	1·57	1·56	1·53	1·59	1·48	1·46	1·45	1·42	1·41

MATERIALS MINED Quantities in (10^6)		PRODUCTION YEARS											TOTAL
		25	26	27	28	29	30	31	32	33	34	35	
COAL MINED (Tonnes)	Annual	9·83	8·01	9·19	8·39	8·55	8·58	8·64	8·61	8·25	8·05	7·62	
	Cumul.	247·05	255·05	264·24	272·63	281·18	289·77	298·41	307·02	315·28	323·33	330·95	330·95
MJ / Kg (Dry Basis)	Annual	18·5	19·2	17·0	18·4	18·0	17·9	17·8	17·9	18·7	19·1	20·2	
	Cumul.	18·0	18·0	18·0	18·0	18·0	18·0	18·0	18·0	18·0	18·0	18·0	
ASH CONTENT (%, Dry Basis)	Annual	32·77	32·40	32·40	32·40	32·40	32·40	32·40	32·40	32·40	32·40	32·40	33·47
SULPHUR CONTENT (%, Dry Basis)	Annual	0·48	0·48	0·48	0·48	0·48	0·48	0·48	0·48	0·48	0·48	0·48	0·51
HEAT UNITS DELIVERED IN MJ x 10^9 (Based on 23·5 % Moisture and Coal Cut-Off at 9·3 MJ/Kg)	Annual	139·12	117·65	118·11	118·10	117·73	117·49	117·65	117·90	118·02	117·62	117·75	
	Cumul.	3396·76	3514·41	3632·52	3750·62	3868·35	3985·84	4103·49	4221·39	4339·41	4457·03	4574·78	4574·78
COAL Fuel above cut-off of 9·3 mj/kg (bank cubic metres)	Annual	6·60	5·38	6·17	5·63	5·74	5·76	5·80	5·78	5·54	5·40	5·11	
	Cumul.	165·81	171·17	177·30	182·97	188·71	194·48	200·28	206·05	211·60	217·00	222·11	222·11
WASTE Above Bedrock	Annual	2·10	6·00	6·01	6·00	6·01	6·01	6·01	6·01	5·99	1·43	1·20	250·22
WASTE Bedrock	Annual	7·90	3·83	3·84	3·84	3·84	3·85	3·85	3·84	3·83	0·92	0·76	176·58
SUB – TOTAL WASTE (bank cubic metres)	Annual	10·00	9·83	9·85	9·84	9·85	9·86	9·86	9·85	9·82	2·35	1·96	
	Cumul.	343·73	353·56	363·41	373·25	383·10	392·96	402·82	412·67	422·49	424·84	426·80	426·80
TOTAL MATERIAL MINED (bank cubic metres)	Annual	16·60	15·21	16·02	15·47	9·58	15·62	15·66	15·63	15·81	7·75	7·07	
	Cumul.	509·54	524·73	540·71	556·22	571·81	587·44	603·10	618·72	634·09	641·84	648·91	648·91
STRIP RATIO bank cubic metres waste / tonnes of coal mined	Annual	1·02	1·23	1·07	1·17	1·15	1·15	1·14	1·14	1·19	0·29	0·26	
	Cumul.	1·39	1·39	1·38	1·37	1·36	1·36	1·35	1·34	1·34	1·31	1·29	

HAT CREEK PROJECT

Table No. 5·6

Schedule of Annual Production

00142 ½ 10

SOURCE: British Columbia Hydro and Power Authority

TABLE 5-7

HOUTH MEADOWS WASTE DUMP
(Capacity by Lift - million bank m^3)
Embankment Crest at 1,005 m Elevation

	Retained Waste		Embankment		Total		Year	
Elevation	35-m Lift	Cum.	35-m Lift	Cum.	35-m Lift.	Cum.	Start	Complete
floor- 880	0.63	0.63	4.97	4.97	5.60	5.60	-2	2
880- 915	14.95	15.58	17.52	22.49	32.47	38.07	1	6
915- 950	35.67	51.25	14.20	36.69	49.87	87.94	2	9
950- 985	58.86	110.11	9.15	45.84	68.01	155.95	6	14
985-1,020	83.15	193.26	5.29	51.13	88.44	244.39	9	22
1,020-1,055	82.19	275.45	4.77	55.90	86.96	331.35	23	(35)
1,055-1,090	62.35	337.80	4.34	60.24	66.69	398.04	-	
1,090-1,125	30.90	368.70	2.72	62.96	33.62	431.66	-	
1,125-1,160	7.70	375.40	0.67	63.63	7.37	439.03	-	

Note: Embankment quantities also include material for the three secondary embankments North of Houth Meadows.

Available capacity after Year 35 = 134.03 million bank m^3

TABLE 5-8

MEDICINE CREEK WASTE DUMP

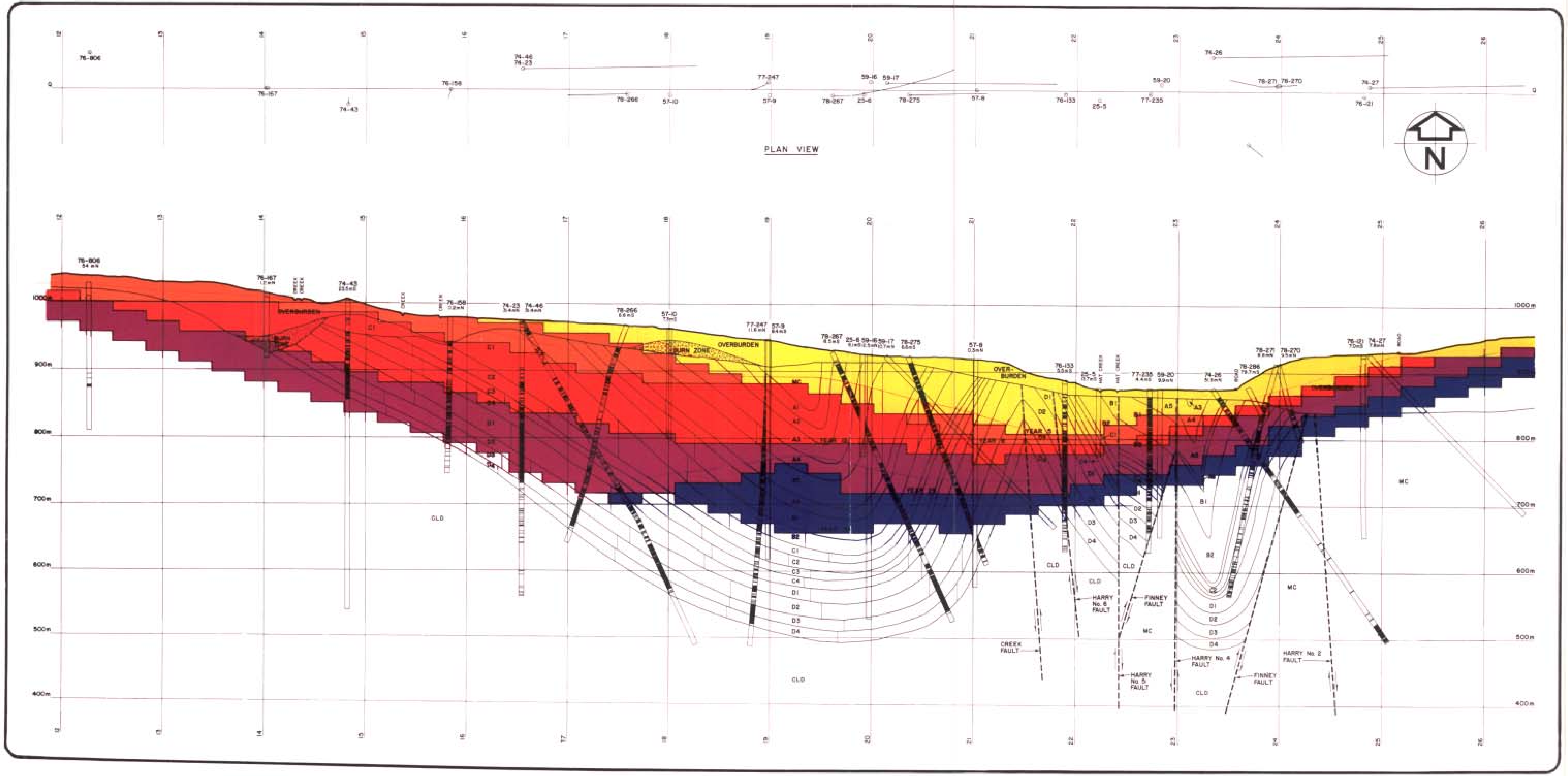
(Capacity by Lift - million bank m^3)

Embankment Crest at 1,130 m Elevation

Elevation	Retained Waste 35-m Lift	Retained Waste Cum.	Embankment 35-m Lift	Embankment Cum.	Total 35-m Lift	Total Cum.	Year Start	Year Complete
floor-1,040	2.11	2.11	7.26	7.26	9.37	9.37	12	14
1,040-1,075	11.46	13.57	10.16	17.42	21.62	72.74	15	18
1,075-1,110	29.75	43.32	15.04	32.46	44.79	117.53	18	26
1,110-1,145	40.85	84.17	9.29	41.75	50.14	167.67	26	(35)
1,145-1,170	16.16	100.33	-	4.75	16.16	183.83	-	-

Available capacity for mine waste after 35 years = 29 million bank m^3

Potential capacity by raising embankment crest from 1,130 m to 1,200 m elevation = 310 million loose m^3 for mine waste and ash.



LEGEND

- First Increment to Year 5
- Year 5 to Year 8
- Year 8 to Year 15
- Year 15 to Year 25
- Year 25 to Year 35

100m 0 100 200 300m

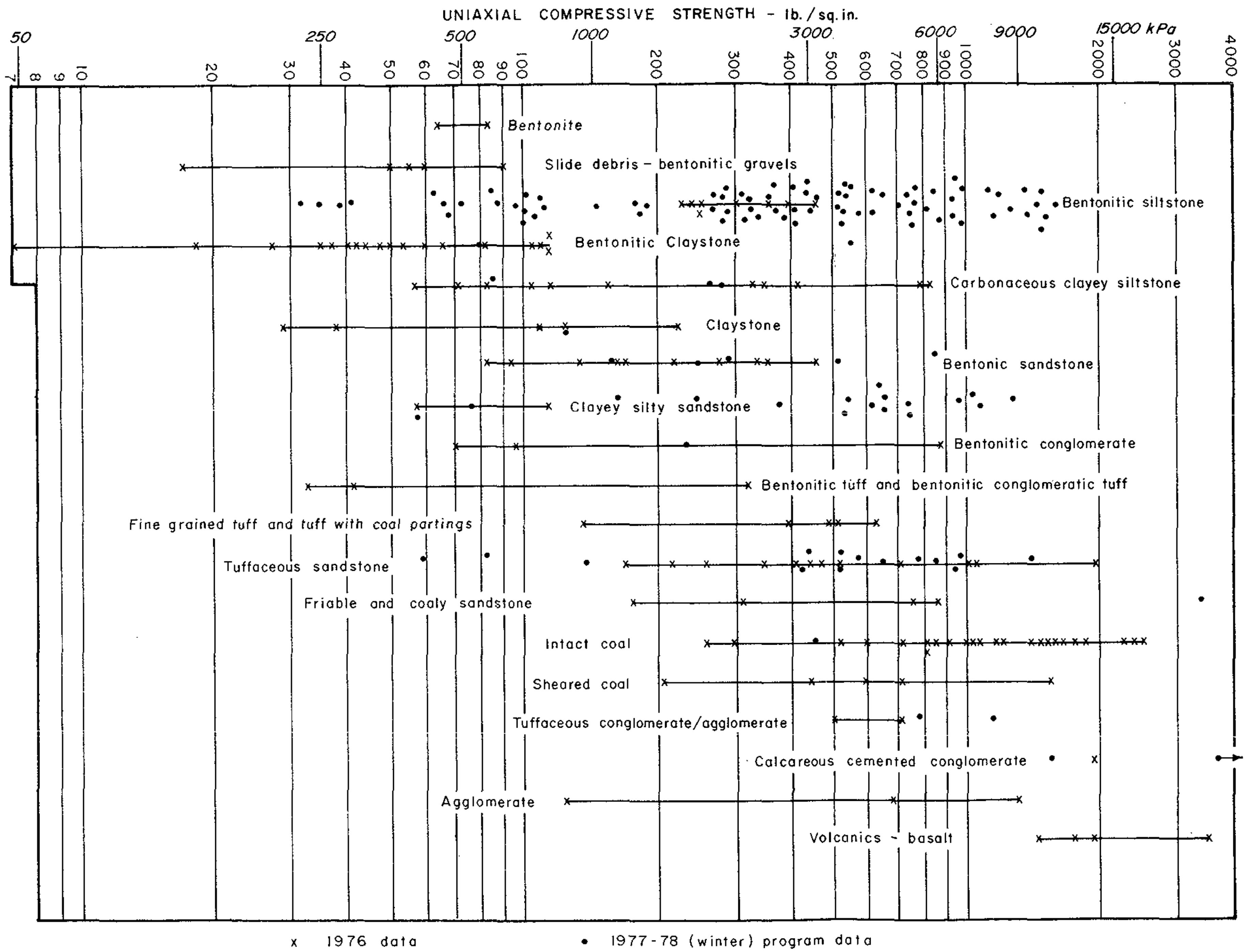
HAT CREEK PROJECT

FIGURE 5-1

Interim Pit Development: Cross Section Q

00142 1/2 (11)

SOURCE: British Columbia Hydro and Power Authority

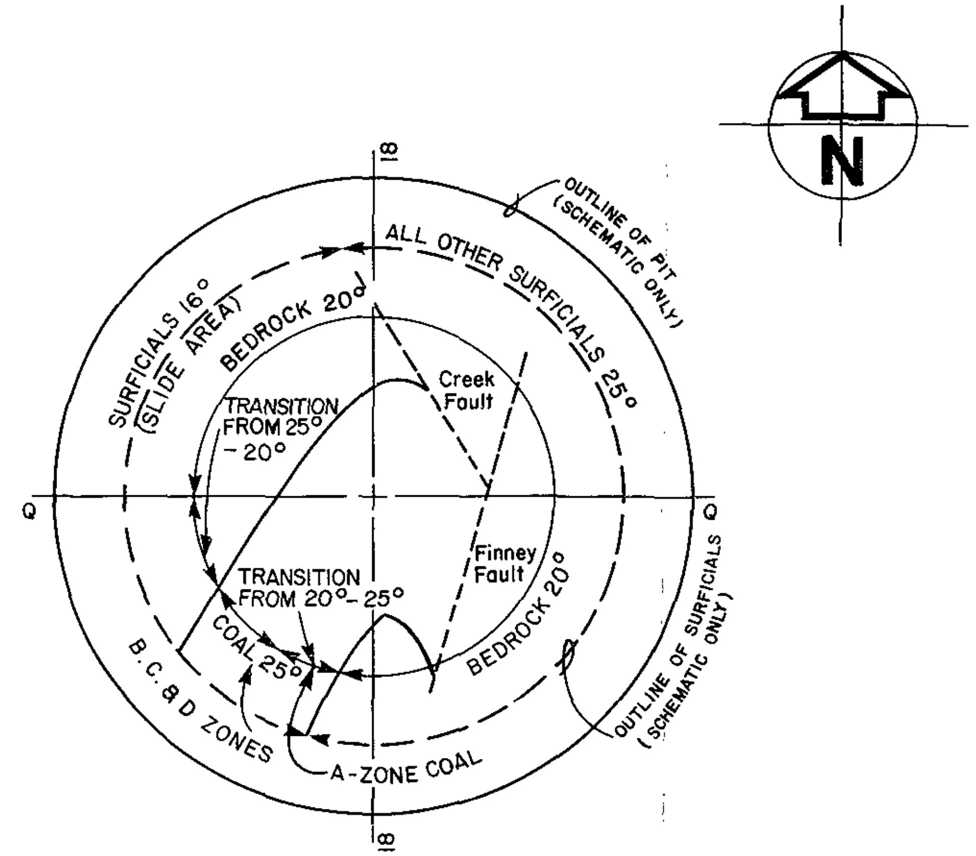


HAT CREEK PROJECT

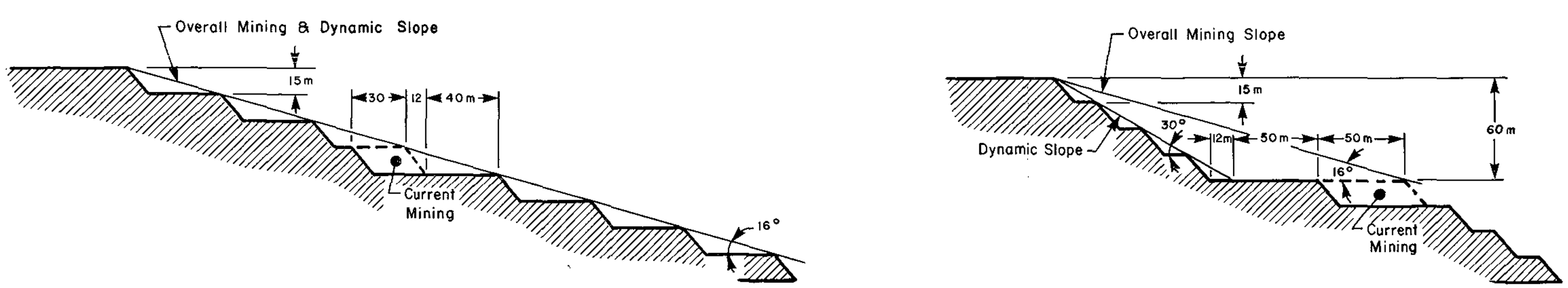
FIGURE 5-2

UNIAXIAL COMPRESSIVE STRENGTHS

SOURCE: Golder Associates



35 YEAR PIT SLOPE ANGLES



(a) FOR MINING COAL

(b) FOR MINING WASTE AND OVERBURDEN

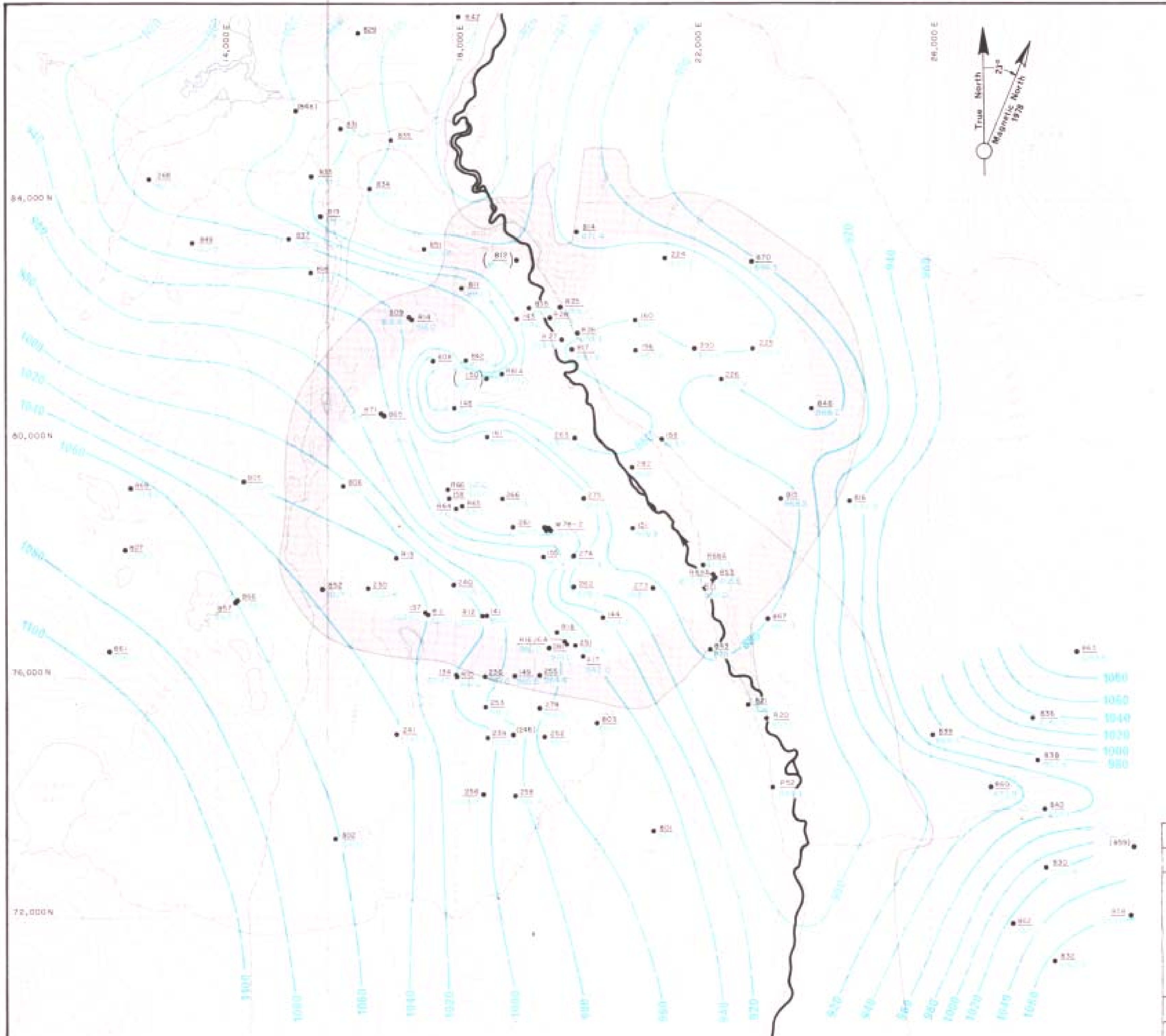
DYNAMIC SLOPE CROSS SECTION

HAT CREEK PROJECT

FIGURE 5-3

PIT SLOPES

SOURCE: Golder Associates



LEGEND

Piezometric contours, 20 metre intervals elevations in metres

Inferred piezometric contours

829 Data point, upper (black) number = hole number lower (blue) number = piezometric elevation in metres

Bedrock groundwater discharge areas, other than those along Hat Creek

NOTES

1. Due to the variation of piezometric elevations both with depth and time, the contoured piezometric surface illustrated in this drawing must be viewed as a generalized representation only.

2. Piezometric elevations are for October 1978.

3. Data points in brackets require the following explanatory notes:
- Artesian pressures exist in drill holes DDH-77-848, DDH-77-246 and DDH-78-859. Although the piezometric elevations at these locations are not known at present, the artesian pressures have influenced the configuration of the piezometric contours.
- The piezometer in drill hole DDH-76-812 has recently become inoperative. The represented piezometric elevation at this location has been estimated from the extrapolated piezometer hydrograph.
- The piezometric elevation at drill hole DDH-76-150 is anomalously low and not fully contoured at present.

SCALE

200 100 0 200 400 600 800 1000 METRES

1000 500 0 1000 2000 3000 4000 FEET

00142 1/2 (12)

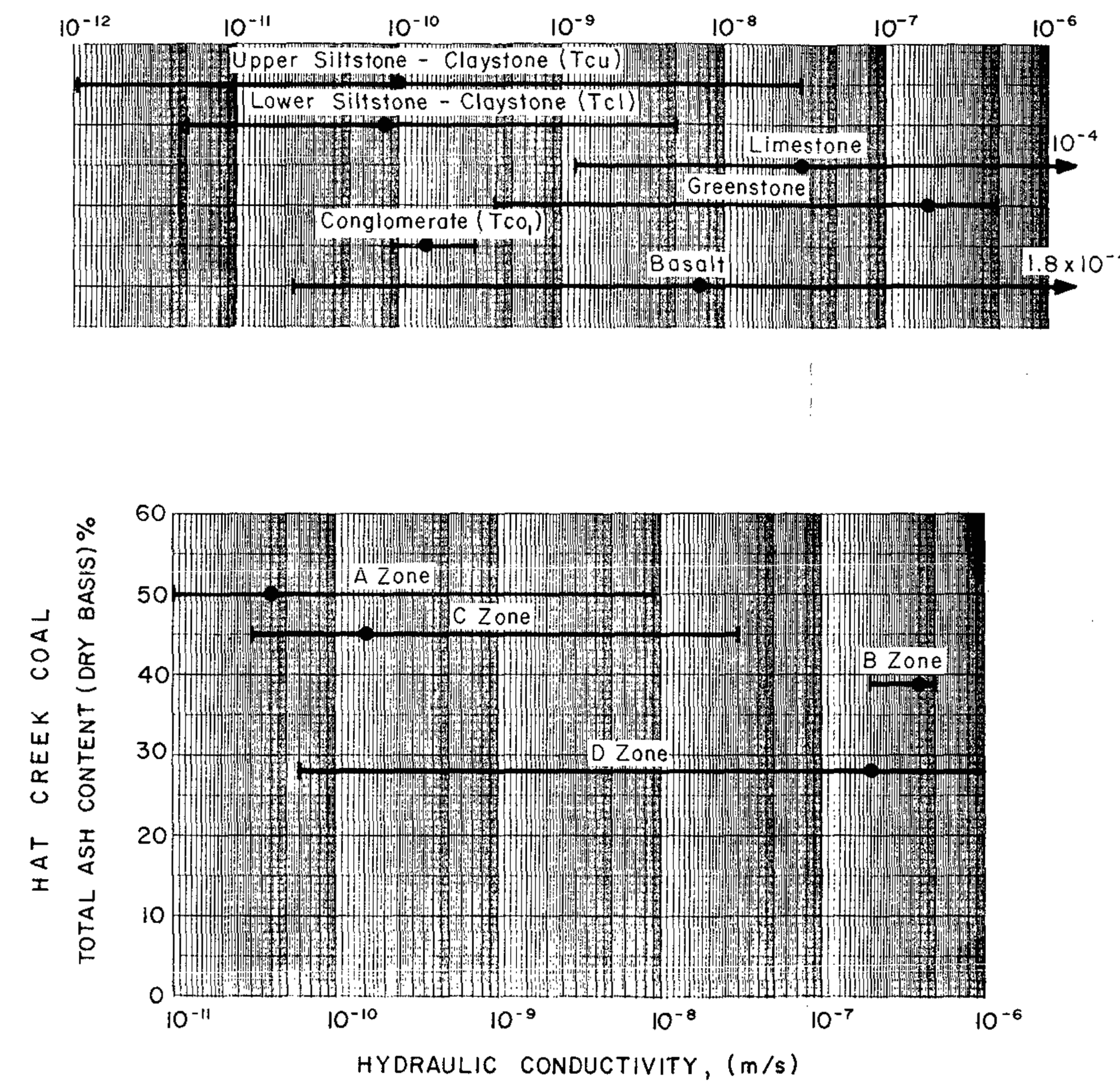
Golder Associates

B.C. Hydro & Power Authority

HAT CREEK GEOTECHNICAL STUDY

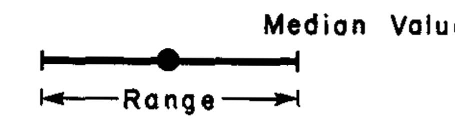
GENERALIZED PIEZOMETRIC SURFACE CONTOURS OF BEDROCK MATERIALS

Drawn S.H.	Checked	Reviewed
Date DEC 1978	Scale AS SHOWN	FIGURE 5-4



NOTE: Values are based on falling head test results shown in Appendix 12, Volume 6 and on pumping test data summarized in Table 6.

LEGEND

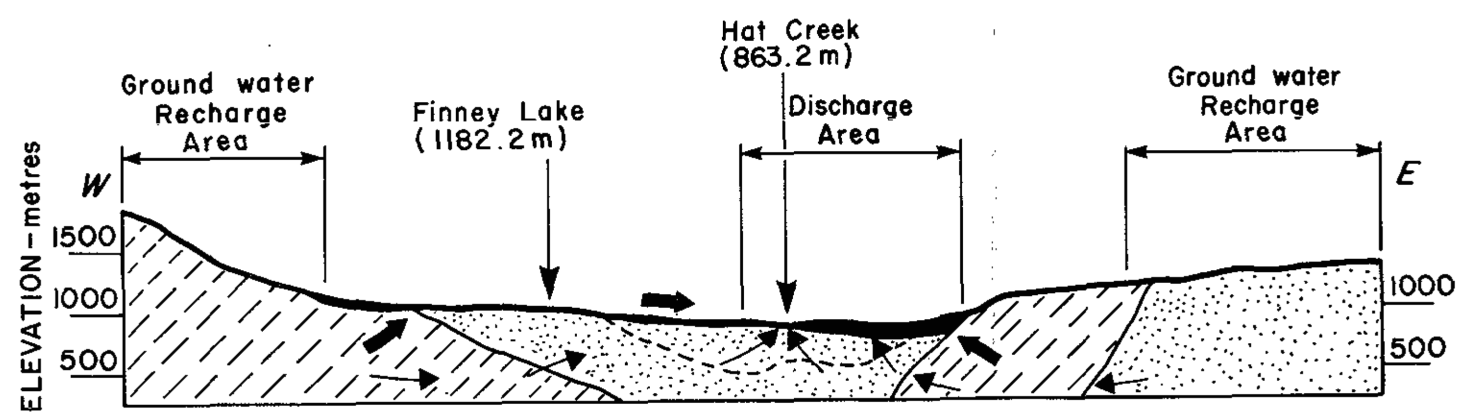


HAT CREEK PROJECT

FIGURE 5-5

HYDRAULIC CONDUCTIVITY VALUES DETERMINED IN-SITU

SOURCE: Golder Associates



NATURAL SCALE

LEGEND

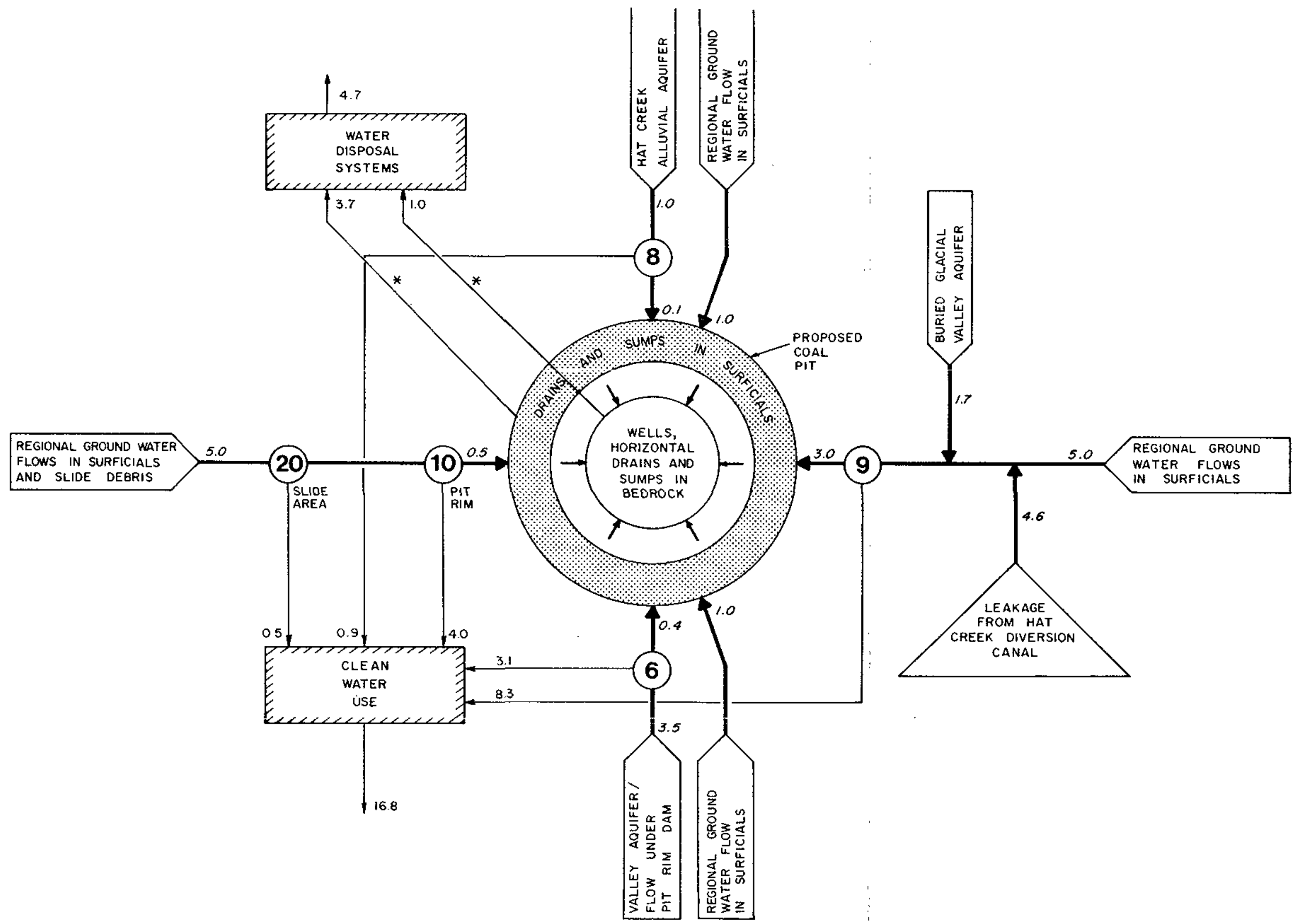
- Rocks with relatively high hydraulic conductivity (greater than 10^{-7} m/s)
- Rocks with relatively low hydraulic conductivity (less than 10^{-7} m/s)
- Surficial sediments
- Low rate of groundwater flow
- Higher rate of groundwater flow
- (863.2 m) Approx. elevation of water surface
- Outline of proposed pit

HAT CREEK PROJECT

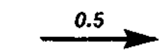
FIGURE 5-6

Hydrology
Section Through Hat Creek Valley

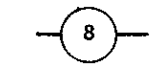
SOURCE: Golder Associates



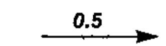
LEGEND



Direction and magnitude of estimated average ground water flows ($10^{-3}m^3/s$)



Estimated number of pumped or bailed wells



Ground water either piped or conveyed by truck for treatment and/or disposal

* These figures allow for water losses of up to 40% due to evaporation

Ground water flow collected from bedrock in base of pit ($1.7 \times 10^{-3}m^3/s$)

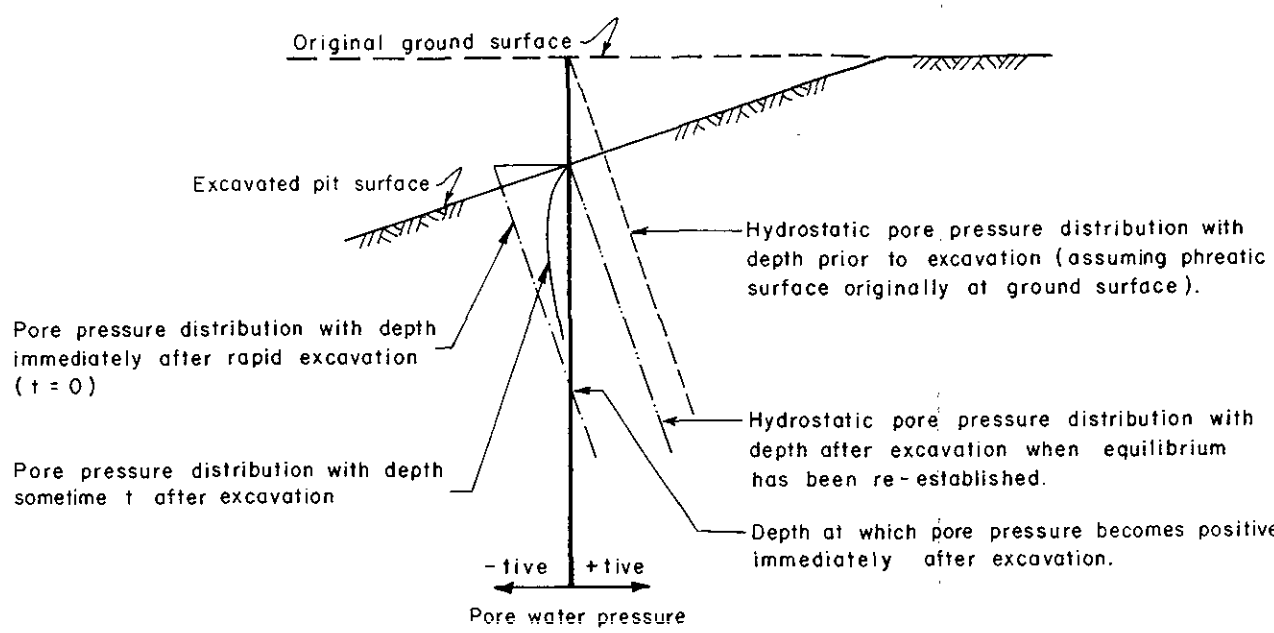
HAT CREEK PROJECT

FIGURE 5-7

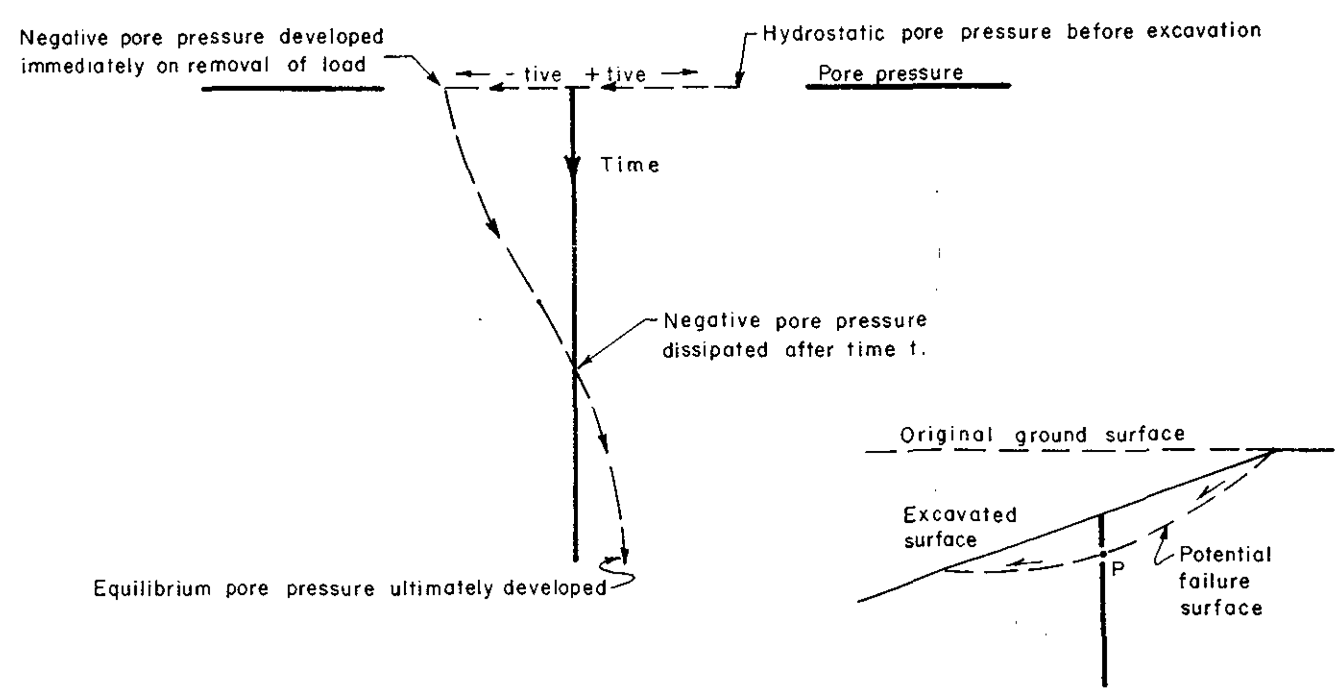
MINE SEEPAGE AND DEWATERING FLOW CHART

SOURCE: Golder Associates

a) PLOT OF DEPRESSURIZATION WITH DEPTH OF EXCAVATION



b) PLOT OF DEPRESSURIZATION WITH TIME FOR ANY POINT P ON A POTENTIAL FAILURE SURFACE.

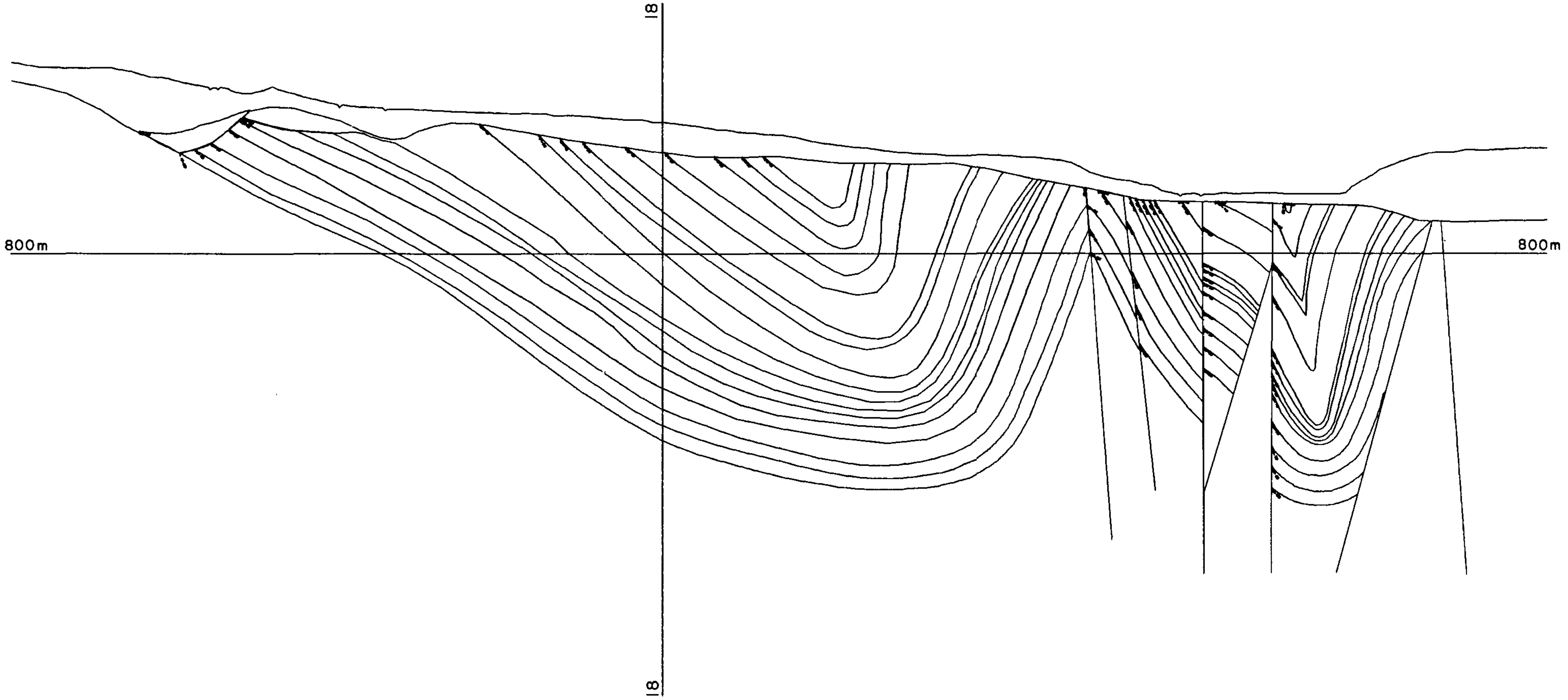


HAT CREEK PROJECT

FIGURE 5-8

DEPRESSURIZATION BY EXCAVATION

SOURCE: Golder Associates



100m 0 100m 200m 300m

HAT CREEK PROJECT

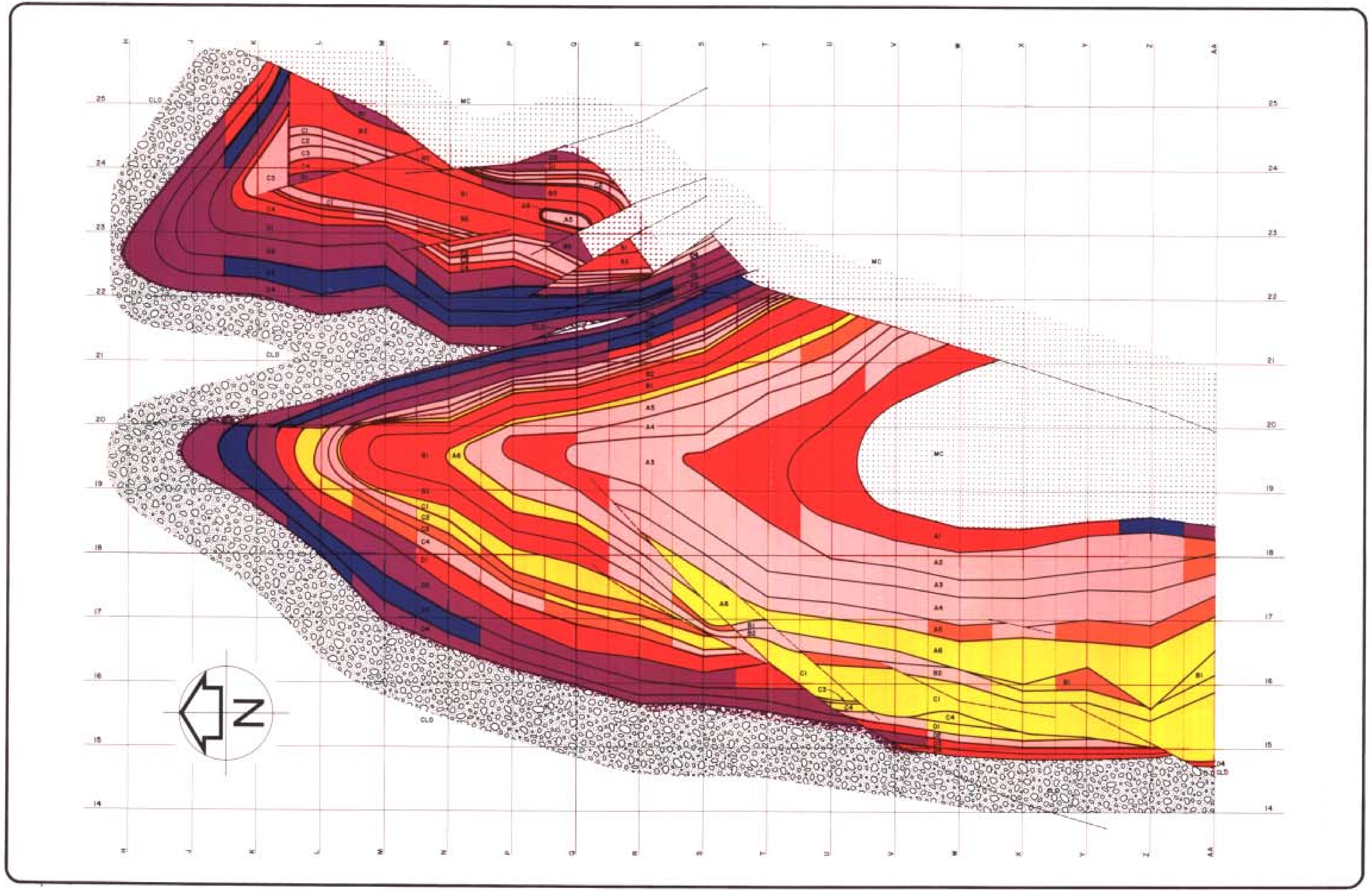
FIGURE 5-9

Typical VBM Cross-Section
Section Q

SECTION DRAWN LOOKING NORTH

00142 1/2 13

SOURCE: Mintec, Inc.



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

- MEDICINE CREEK FORMATION
- COLDWATER FORMATION
- A1 SUB-ZONE
- FAULT
- CONTACT

100m 0 100 200 300 400m

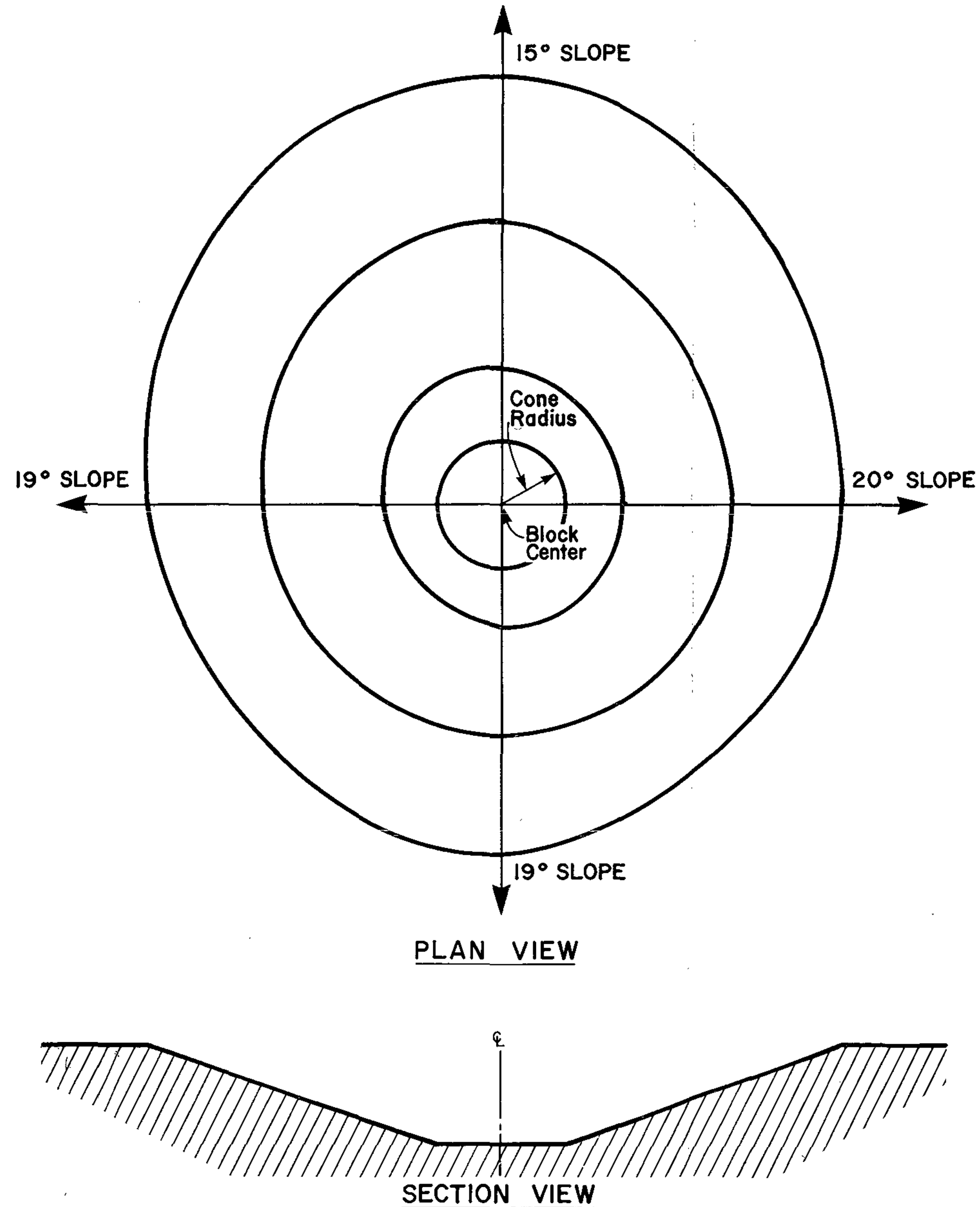
HAT CREEK PROJECT

FIGURE 5-10

Bench Plan-Elevation 775

00142 1/2 (14)

SOURCE: Mintec, Inc.

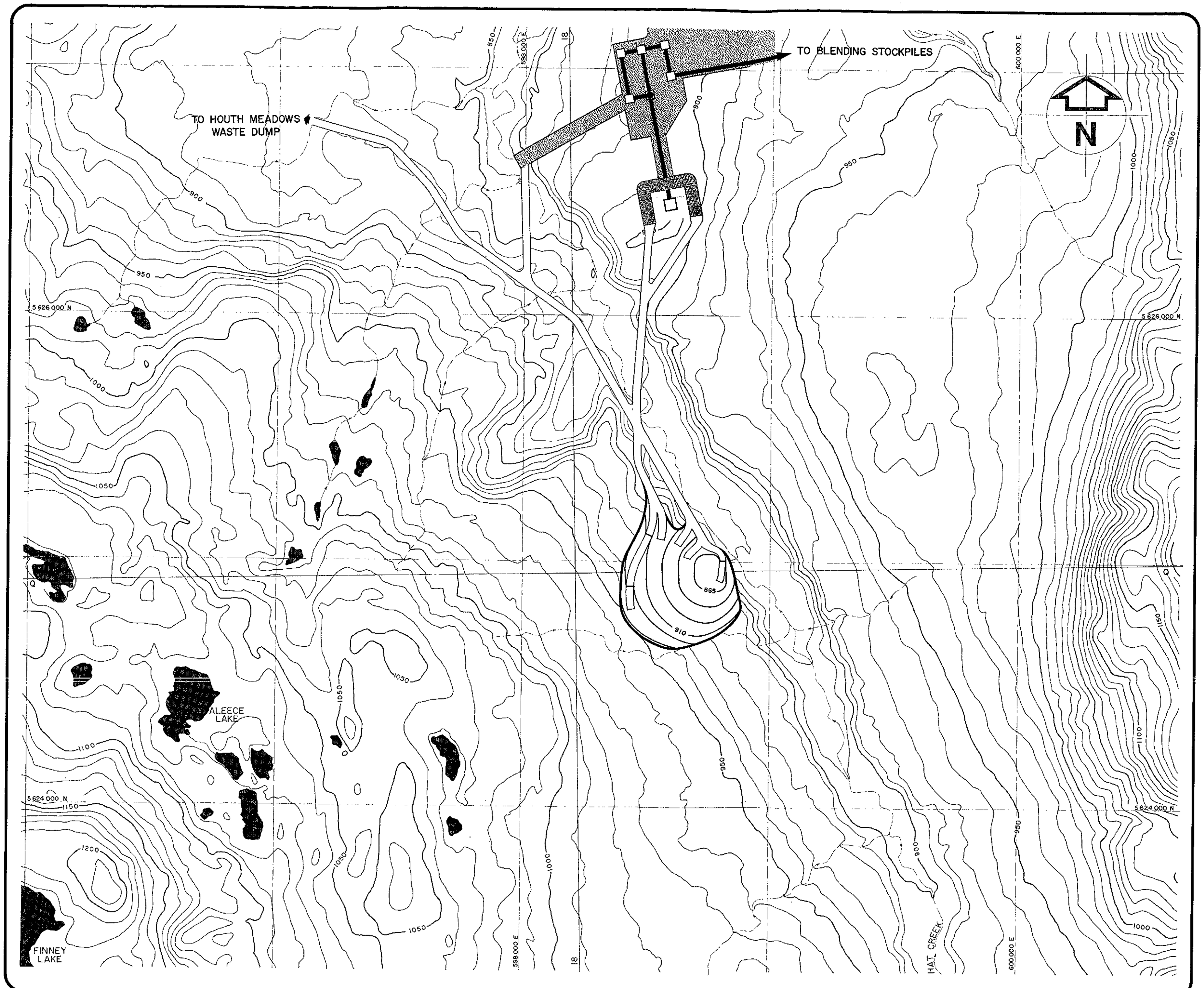


HAT CREEK PROJECT

FIGURE 5-11

Cone Geometry

SOURCE: Mintec, Inc.



LEGEND

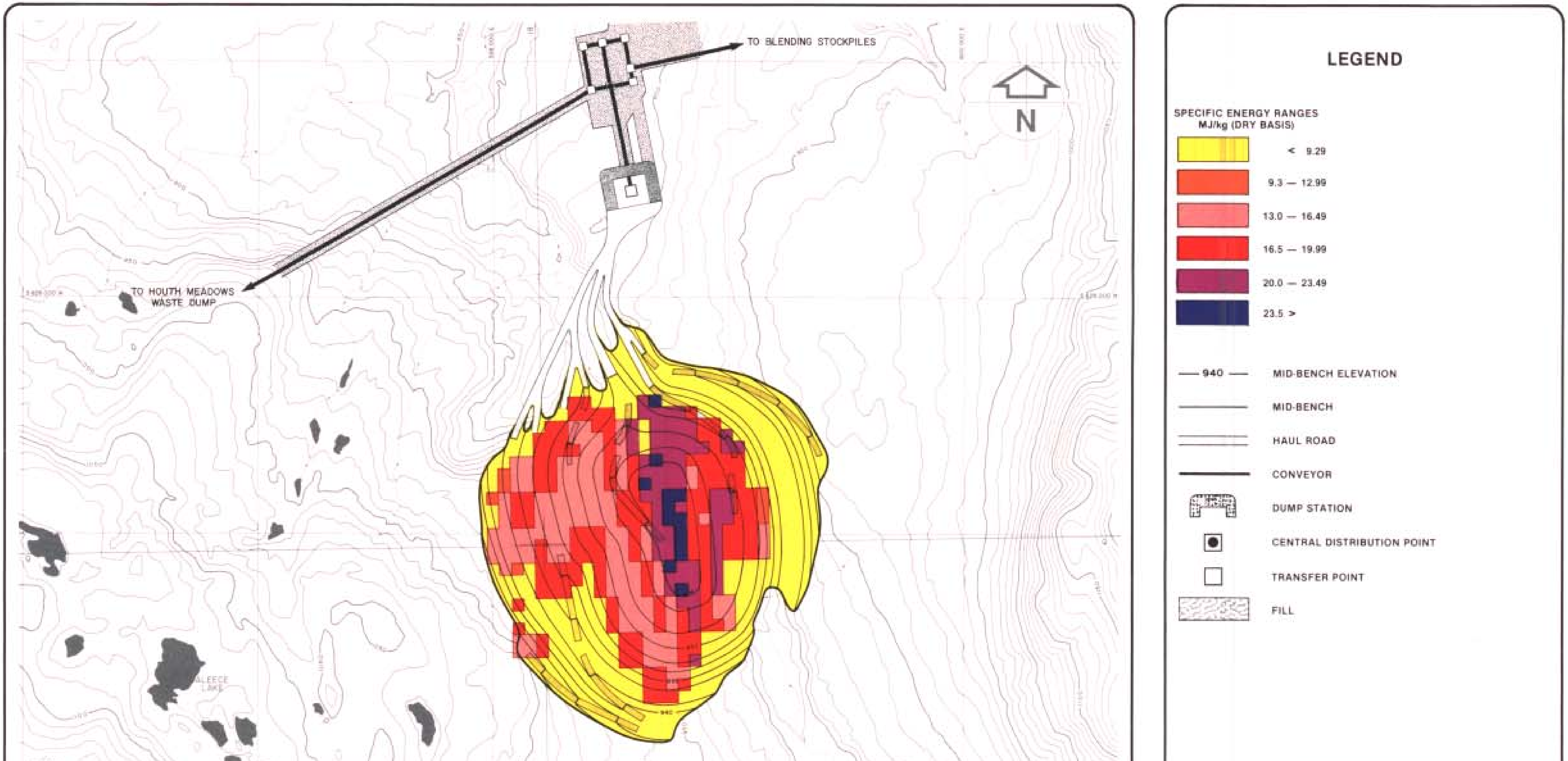
— 910 —	MID-BENCH ELEVATION
	MID-BENCH
	HAUL ROAD
	CONVEYOR
	DUMP STATION
	CENTRAL DISTRIBUTION POINT
	TRANSFER POINT
	FILL

HAT CREEK PROJECT

FIGURE 5-12

Pit Development Year-1

SOURCE: British Columbia Hydro and Power Authority



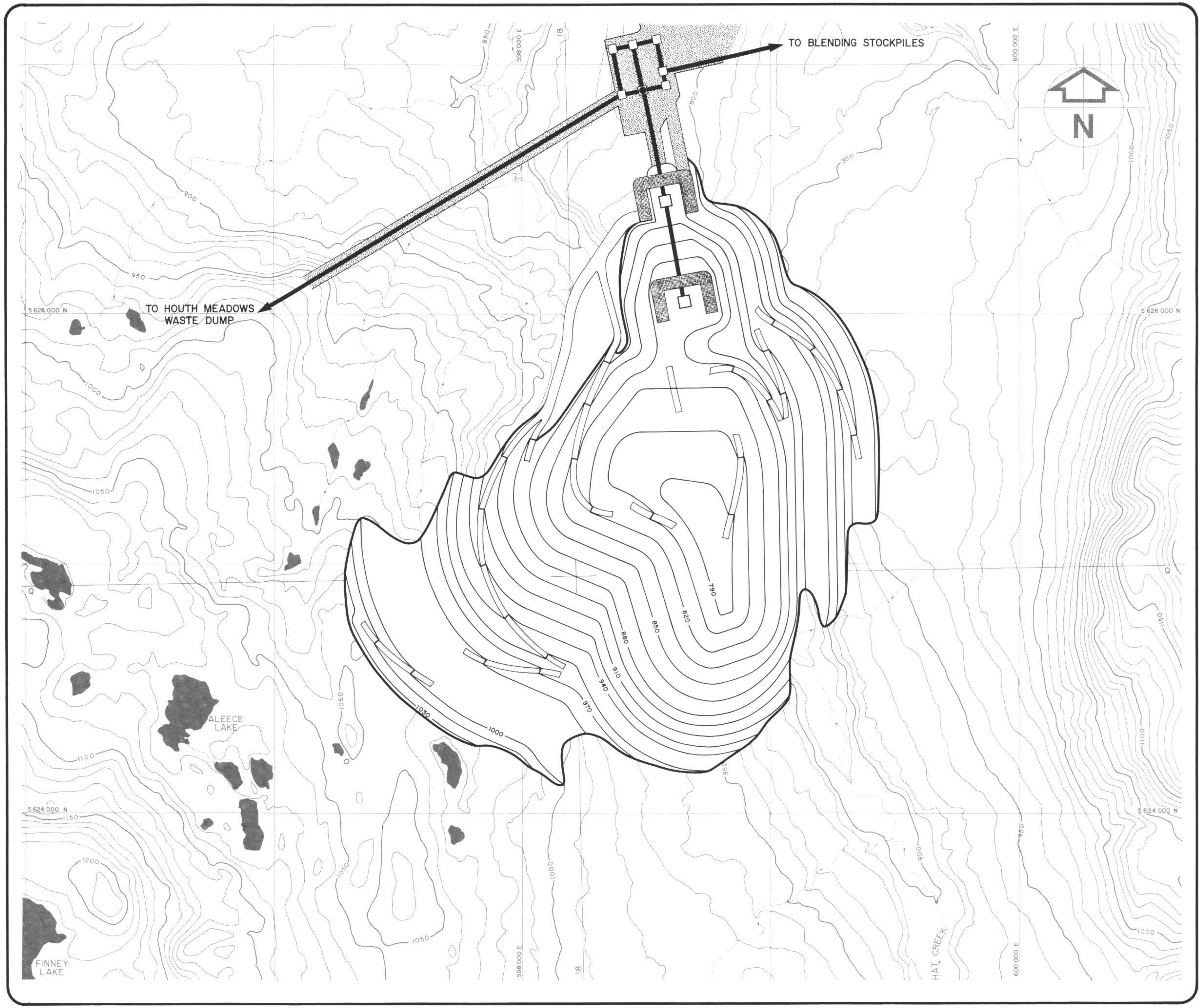
HAT CREEK PROJECT

FIGURE 5-13

Pit Development Year 5

00142 1/2 15

SOURCE: British Columbia Hydro and Power Authority



LEGEND

- 850 — MID-BENCH ELEVATION
- MID-BENCH
- HAUL ROAD
- CONVEYOR
- DUMP STATION
- CENTRAL DISTRIBUTION POINT
- TRANSFER POINT
- FILL

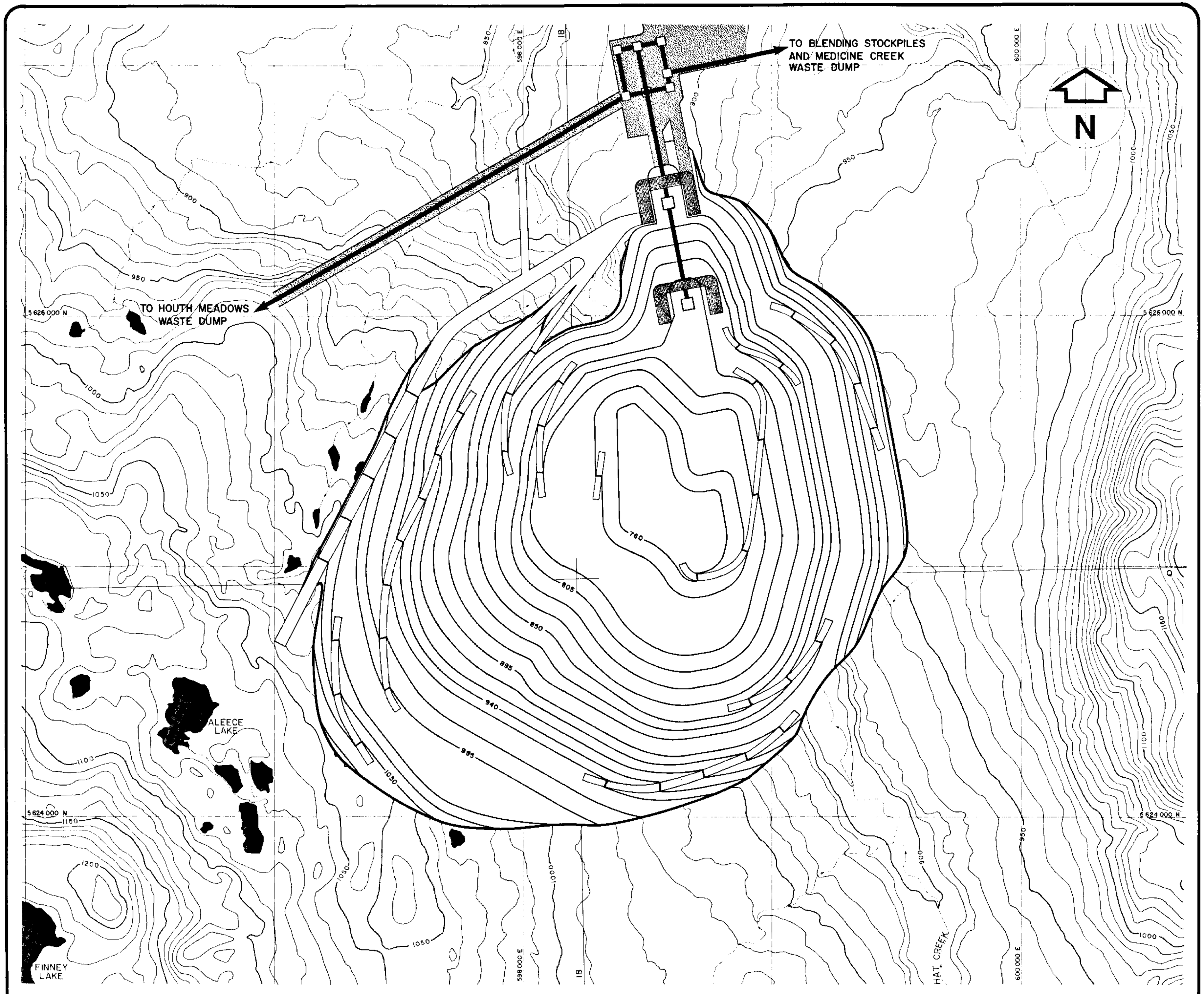
HAT CREEK PROJECT

FIGURE 5-14

Pit Development Year 8

00142 1/2 16

SOURCE: British Columbia Hydro and Power Authority



LEGEND

- 850 — MID-BENCH ELEVATION
- MID-BENCH
- HAUL ROAD
- CONVEYOR
- DUMP STATION
- CENTRAL DISTRIBUTION POINT
- TRANSFER POINT
- FILL

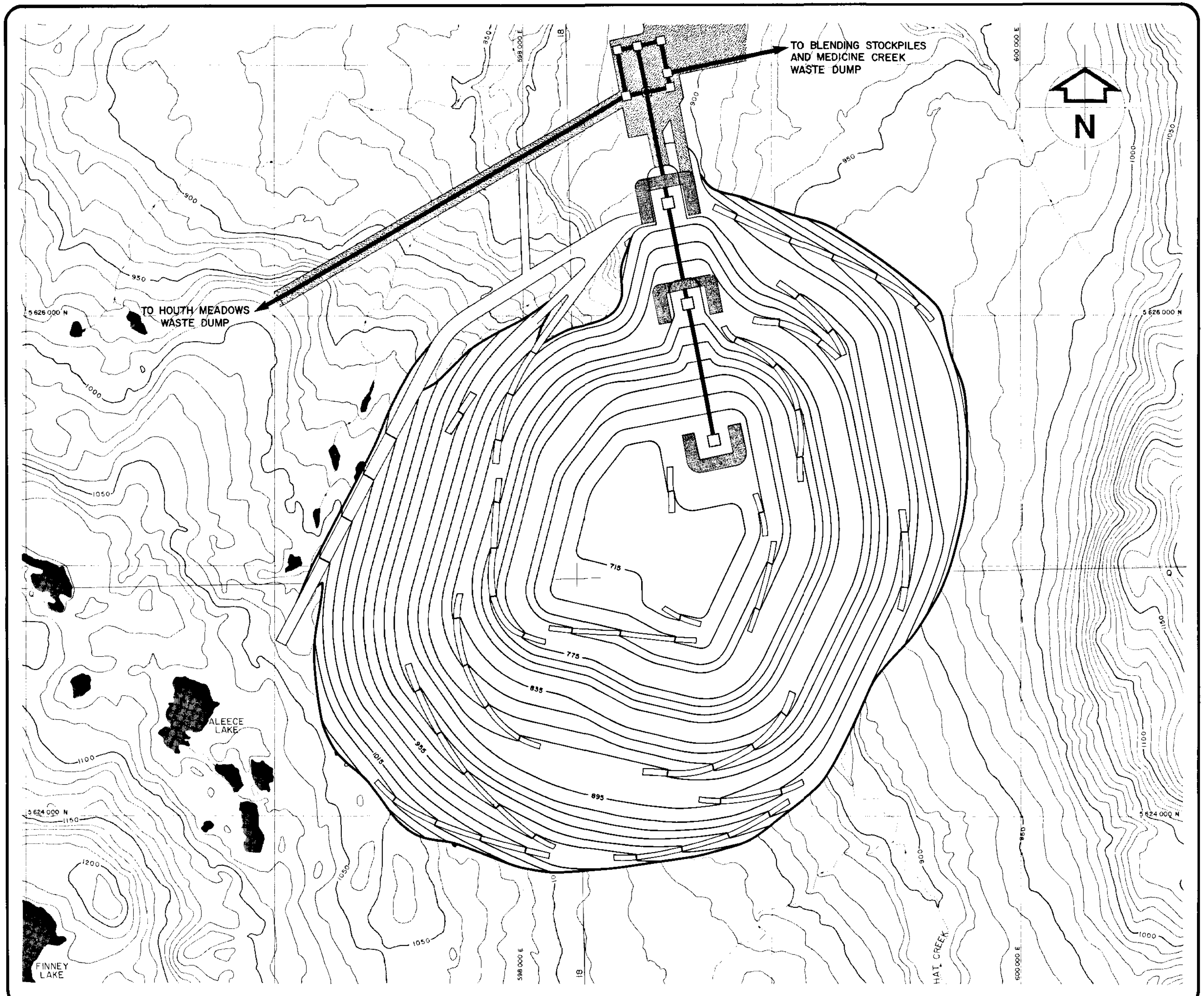
HAT CREEK PROJECT

FIGURE 5-15

Pit Development Year 15

00142 1/2 17

SOURCE: British Columbia Hydro and Power Authority



LEGEND

— 715 —	MID-BENCH ELEVATION
	MID-BENCH
	HAUL ROAD
	CONVEYOR
	DUMP STATION
	CENTRAL DISTRIBUTION POINT
	TRANSFER POINT
	FILL

HAT CREEK PROJECT

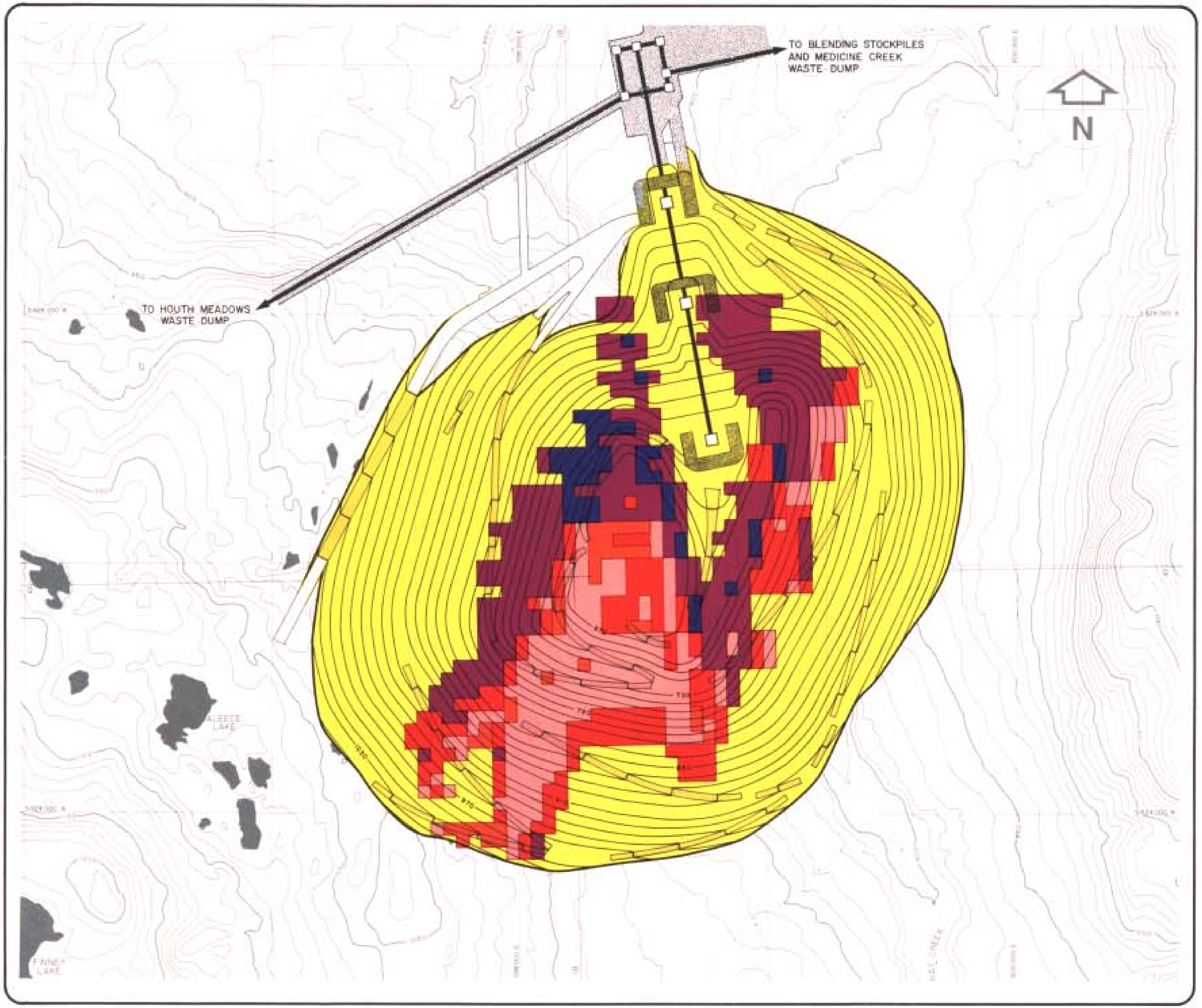
FIGURE 5-16

Pit Development Year 25

00142 1/2

18

SOURCE: British Columbia Hydro and Power Authority



LEGEND

SPECIFIC ENERGY RANGES
MJ/kg (DRY BASIS)

< 9.29

9.3 — 12.99

13.0 — 16.49

16.5 — 19.99

20.0 — 23.49

23.5 >

— 910 — MID-BENCH ELEVATION

MID-BENCH

HAUL ROAD

CONVEYOR

DUMP STATION

CENTRAL DISTRIBUTION POINT

TRANSFER POINT

FILL

HAT CREEK PROJECT

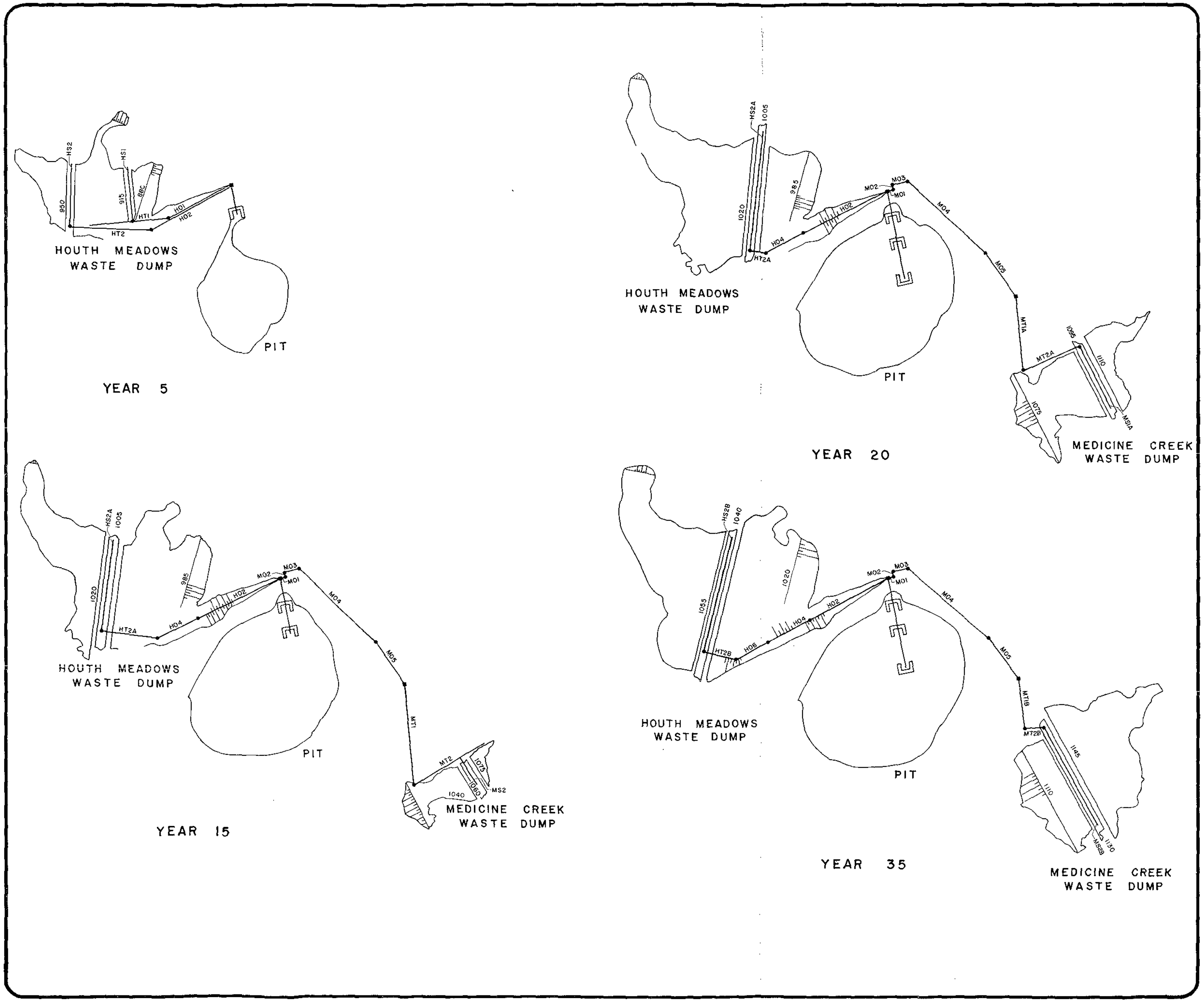
FIGURE 5-17

Pit Development Year 35

00142 ½

(19)

SOURCE: British Columbia Hydro and Power Authority

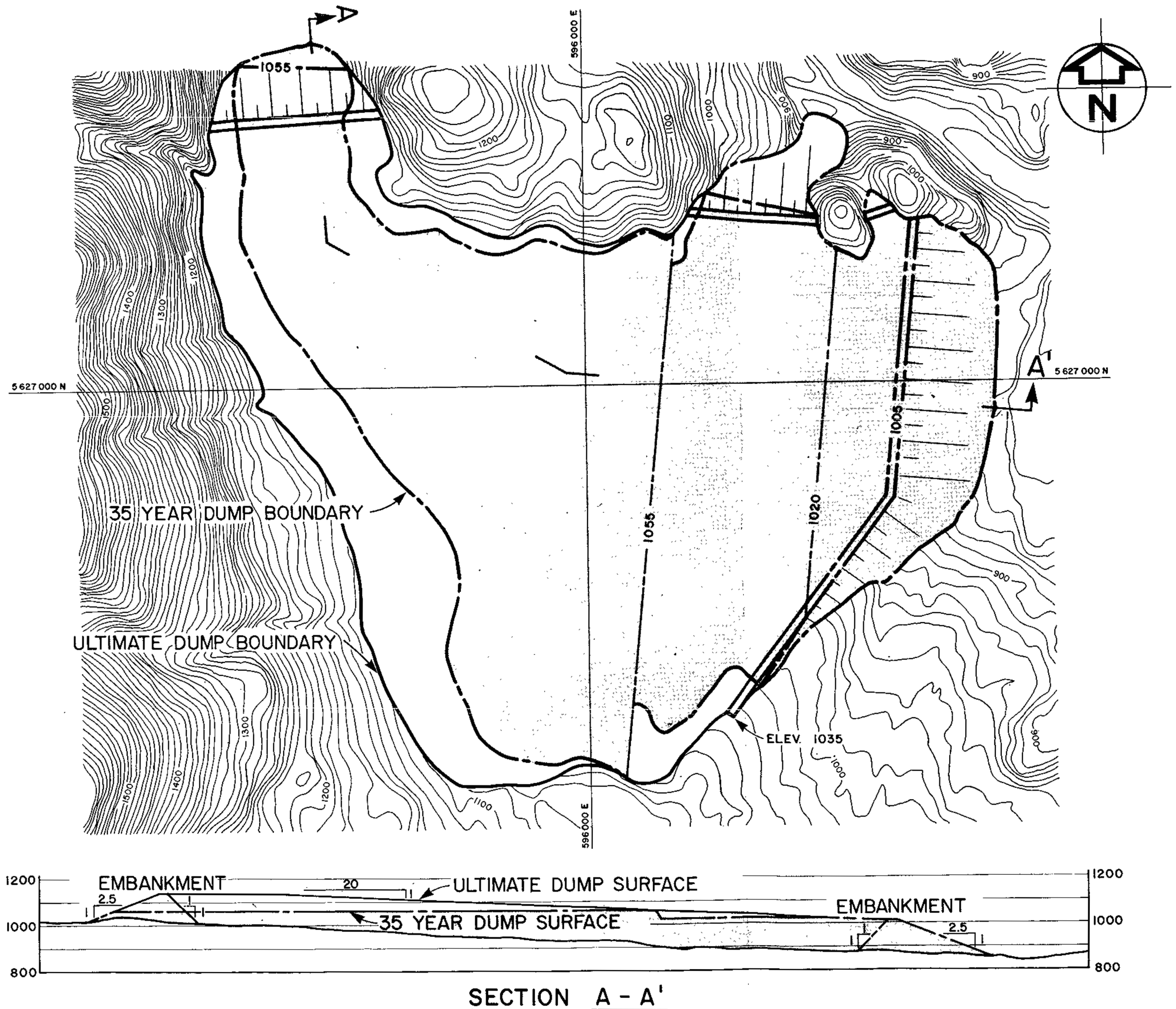


HAT CREEK PROJECT

FIGURE 5-18

Waste Dump Development

SOURCE: *British Columbia Hydro and Power Authority*

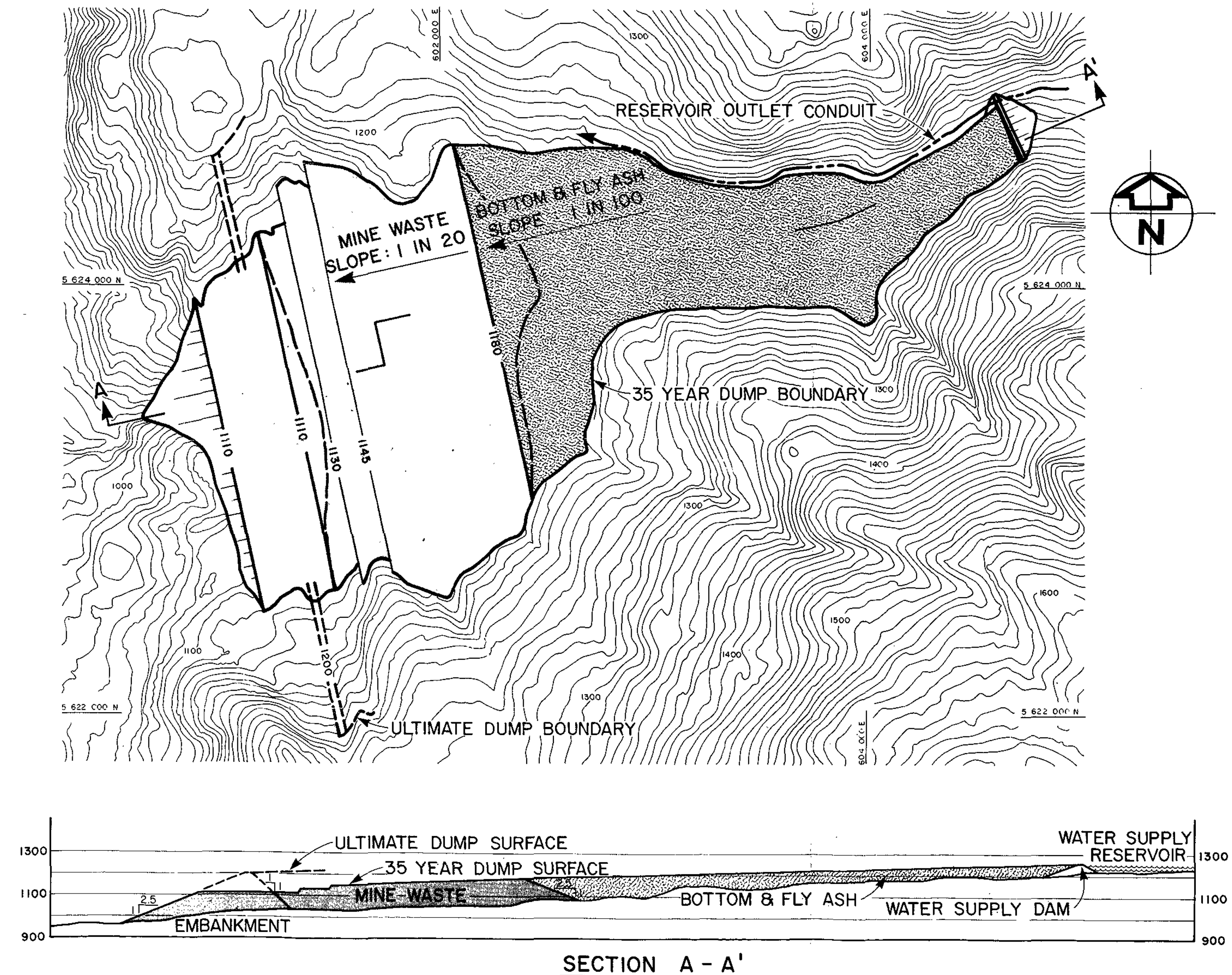


HAT CREEK PROJECT

FIGURE 5-19

Houth Meadows Waste Dump

SOURCE: British Columbia Hydro and Power Authority

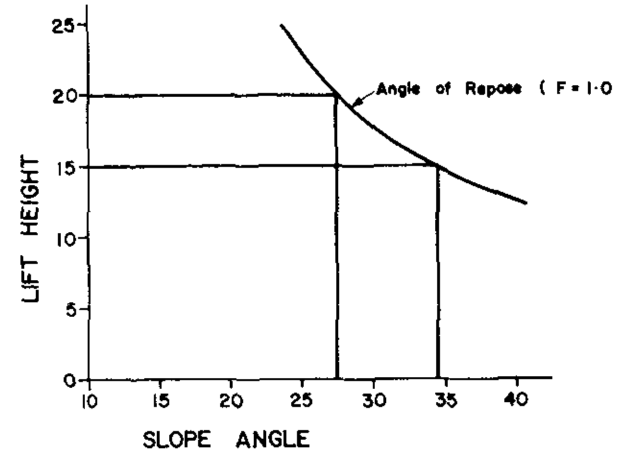


HAT CREEK PROJECT

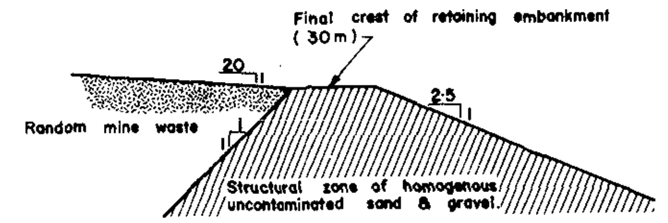
FIGURE 5-20

Medicine Creek Waste Dump

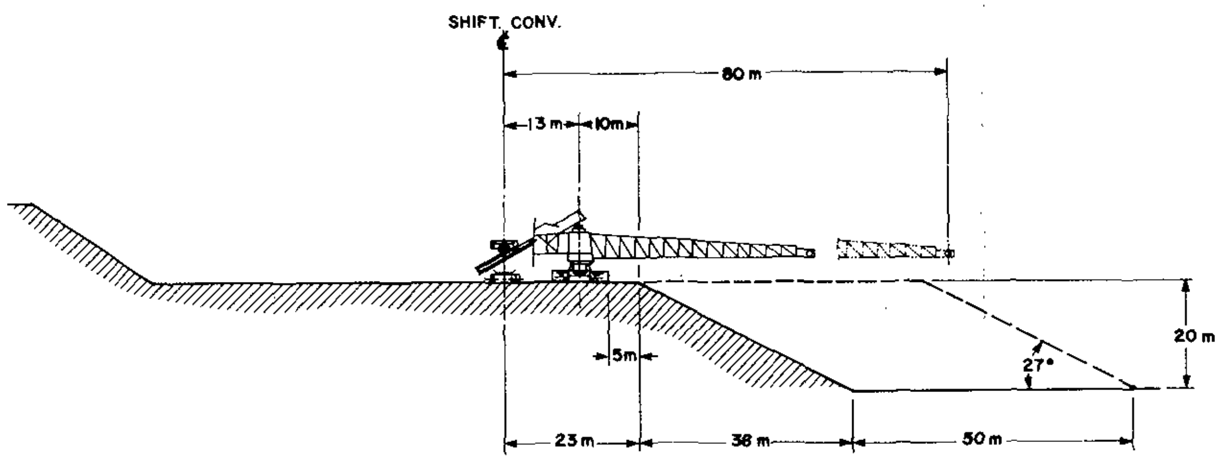
SOURCE: British Columbia Hydro and Power Authority



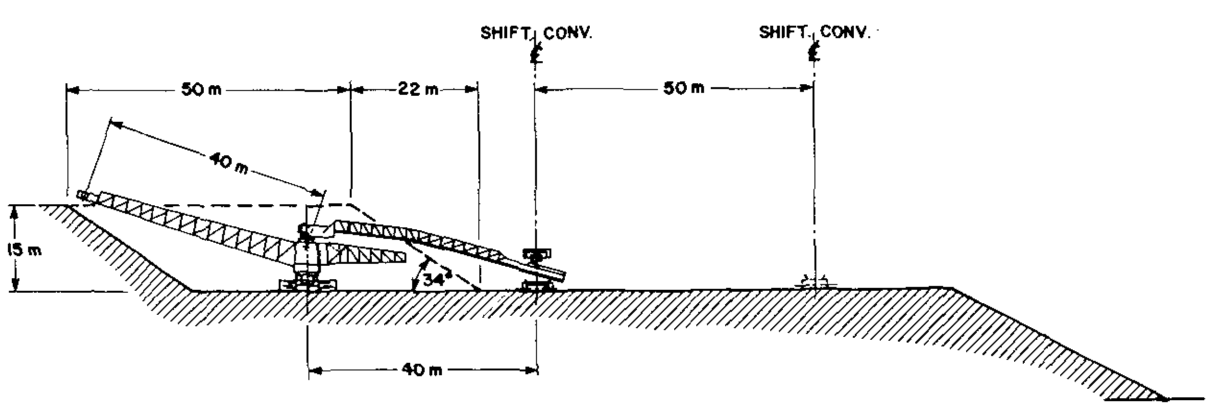
(a) WASTE MATERIAL ANGLE OF REPOSE



WASTE EMBANKMENT SLOPE ANGLES



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



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

HAT CREEK PROJECT

FIGURE 5-21
Waste Dump Slopes

SOURCE: CMJV/Golder Associates

6 THE MINE DRAINAGE PLAN

No.	LIST OF TABLES	Page

LIST OF FIGURES

SECTION 6

THE MINE DRAINAGE PLAN

6.1 INTRODUCTION

Without effective mine drainage, no open-pit mining operation on the scale of Hat Creek could hope to succeed; nor could it satisfy today's stringent environmental requirements. The Mine Drainage Plan devised as a result of painstaking studies by our consultants, Cominco-Monenco Joint Venture, does both (CMJV 1979). It is believed to be a comprehensive mine drainage plan which provides for environmental protection in initial mine planning. Its objectives are:

1. to keep the mine dry enough to ensure continuous operation;

2. to prevent flood damage to both excavations and equipment;

3. to ensure the stability of slopes and embankments;

4. to 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.

This report covers in detail all elements of the Hat Creek Mine Drainage Plan during the first 35 years' mining of the No. 1 Deposit, and the continuing measures after the mine has closed to ensure that the environment is restored as nearly as possible to its former condition.

6.2 MINE WATER: SOURCE AND QUANTITY

The principal sources of drainage flow within the mining area are:

1. Direct precipitation and runoff;
2. Creeks entering the mine site;
3. Standing surface water in lakes and ponds;
4. Groundwater flow;
5. Wastewater from mine operations.

6.2.1 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. Figure 6-1 shows the seasonal variation of precipitation and the frequency of annual and 24-hour precipitation. Roughly 16% of the annual precipitation which falls in the valley appears as streamflow, which indicates a high loss of moisture to infiltration and evapotranspiration. Most runoff occurs in Spring and early Summer, the most intense rainstorms in mid-Summer. Flood hydrographs show that only 24% of the precipitation appears as direct runoff due to the high storage potential of the surface cover and high losses to evapotranspiration (Beak 1978). Mining activities are expected to reduce this surface storage capability and increase the runoff, resulting in increased peak flow rates from the watersheds. Maximum flow rates are expected during high intensity rainfall in Summer, calculated by the method used by the USDA Soil Conservation Service (1964). This volume of runoff is correlated to peak flow rates which have been assembled from field data for small agricultural watersheds (USDA SCS 1975).

Surface runoff at the top of the active waste dumps is expected to be negligible. Leachate from waste dumps, which is expected to be low due to the low hydraulic conductivity of dumped waste, will be collected at the toe of the downstream waste embankments. Seepage and runoff from the coal and waste rock strata within the pit will be of similar quality to the stockpile and waste dump effluents. An average water yield of 80 mm has been assumed for these areas, giving mean annual flows of 0.003 m^3/s - 0.01 m^3/s during the lifetime of the mine. Flow rates for waste dump leachate and pit seepage as estimated by Golder in 1979 are presented in Table 6-1.

6.2.2 Creeks, Lakes and Ponds:

6.2.2.1 Creeks:

The principal creeks flowing through the proposed mine area are Hat Creek, Medicine Creek, Houth Creek, and Finney Creek. Of these, Hat Creek is the largest, and flows have been continuously recorded since 1960. Figure 6-2 shows the range of monthly variation of Hat Creek. Flow guages established in four other creeks in 1977 have as yet produced insufficient data to provide statistical analysis of flows, but such data as do exist indicate that the flow regimes are similar to that of Hat Creek. Flood frequency curves derived from regional streamflow data are shown on Figure 6-2.

The proposed development of the open pit will require diversion of flows from various small watersheds and tributary creeks. Regional streamflow data shown as a flood nomograph gives estimates of flood flows for watersheds greater than 10 km^2 in area.

6.2.2.2 Lakes and Ponds:

Most lakes and ponds in the project area occur on the West side of Hat Creek Valley. There are approximately 80 small lakes and ponds to the West of the proposed pit perimeter.

Geotechnical studies of this area have identified both active and inactive slide masses in the overburden which may cause instability of the West pit slope during mining (Golder 1977, 78/79). Stabilization measures require that Aleece Lake and 61 other lakes and ponds be drained. Finney Lake and 15 other small ponds lie in a more stable and remote area, and therefore drainage is not considered essential at the outset of the project. Monitoring of the slide during mining should give an advance indication of any need to drain Finney Lake and these other ponds. Fifteen to 20 small lakes and ponds in the Houth Meadows Waste Dump Area should be drained prior to being covered with waste.

6.2.3 Groundwater:

Studies to date have identified three major geohydrologic units within the general mine area (Golder, 1978) which comprise:

(1) the surficial deposits, which vary from slide debris and till in the West to gravels in the East. This is the major waterbearing unit of highest average conductivity 10^{-6} m/s;

(2) the coal, which exhibits highly variable conductivity, estimated to average 5×10^{-9} m/s;

(3) the upper and lower Coldwater sediments which are essentially impermeable with an average conductivity of 5×10^{-11} m/s.

General groundwater flow within the Upper Hat Creek Valley recharges in upland areas and discharges in the valley bottom. Most of the groundwater flows through surficial deposits. Less than 2% is estimated to move through clastic sediments in the valley bottom.

The Eastern areas are reasonably well drained due to the greater depths of surficial deposits, whereas they are thinner in Western areas in addition to being of lower permeability.

The two main aquifers in the pit area are a small alluvial aquifer along the central valley and a buried bedrock channel on the East side of the valley, flow of which is estimated to be in the area of $3 \times 10^{-2} m^3/s$.

Due to the low permeability of the coal and bedrock units, water yield from seepage and draining operations during mining is predicted to be minimal (Golder, 1978). Extensive depressurization of pit slopes is not likely, and dewatering wells will therefore be selectively located in pervious zones, where higher benefits can be realized.

Flow from peripheral dewatering wells is estimated to be 0.02 m^3/s one year prior to commencement of mining, decreasing to a steady rate of 0.017 m^3/s throughout the remainder of the project. Groundwater which by-passes this system and appears as seepage in the pit is expected to average 0.0047 m^3/s, of which 0.0037 m^3/s would seep from the surficial deposits and 0.001 m^3/s from the bedrock zone at the base of the pit (Golder, 1979, Appendix 2).

6.2.4 Mine Wastewater:

Three main sources of wastewater produced by the mining operations have been identified:

1. effluent from the Mine Services Area;

2. runoff and leachate from coal-handling areas, waste dumps, and low-grade stockpiles;

3. runoff and seepage from coal and bedrock strata in the open pit.

The major source of waste from the Mine Services Area will be sanitary effluent from the daily work force peaking at about 700 persons. The mean daily flow is estimated at 140 m^3/d, plus an allowance of 90 m^3/d for vehicle washdown and general use.

Runoff and leachate from coal and low-grade stockpiles will require special drainage and disposal systems due to the predicted high levels of dissolved salts. (B.C. Hydro Thermal Division 1979 - 1978 Environmental Field Program.) Water yield from the 33 ha Low-grade Coal Stockpile is expected to average 50 mm/a, with the 22 ha Coal Blending Area yielding an estimated 80 mm/a. These yields correspond to annual volumes of 16,500 m^3 and 17,600 m^3 respectively.

The overburden and waste rock material from the open pit will be retained in valley-fill type dumps in Houth Meadows and Medicine Creek Valley. Any runoff and leachate from mine waste disposal areas will require a special drainage system because of the predicted level of dissolved solids and trace elements in excess of regulatory guidelines for discharge to streams (Beak, 1978/79).

6.3 MINE DRAINAGE SYSTEM:

The proposed mine drainage system will consist of:

1. Diversion canals to divert creeks which flow through the mine site;
2. Perimeter drains around the open pit, slide area, and waste dumps;
3. Dewatering wells around the pit perimeter and the unstable slide area;
4. Surface water drains to collect stormwater in the pit and mine service areas;
5. Field drains to collect leachate from waste dump and stockpiles;
6. Sanitary sewers to collect sewage from the Mine Services Areas.

A schematic of the system is shown on Figure 6-3 and a geographic layout plan on Figure 6-4.

6.3.1 Design Criteria and Selection of System Capacity:

The calculation of system capacity has taken into account the risk of flood damage, should the system fail. Design criteria are shown on Table 6-2 and design flows for the system on Table 6-3. The larger drains or canals have been designed on the basis of the 1,000-year average return period flood, which has a 3% chance of being exceeded during the 35-year mining period. Smaller components are designed to withstand lesser flood risk.

6.3.2 Drainage of the Mine Development:

6.3.2.1 The Open Pit:

1. Diversion of Hat Creek and Finney Creek:

To prevent flooding the excavation, Hat Creek and Finney Creek must be diverted.

The Hat Creek Diversion will consist of a headworks dam with a canal intake and an emergency spillway located downstream of Anderson Creek; approximately 6.4 km of diversion canal on the East side of Hat Creek Valley; and 1.9 km of buried conduit with intake and outlet works to convey the flow back to Hat Creek. A pit rim dam, spillway, pumphouse, and pipeline between the headworks dam and the minepit will intercept seepage and local inflow immediately upstream of the pit. The diversion works have been designed to accommodate a flow of 18 m^3/s (100-year recurrence interval flood), and, as an emergency condition, a flow of 27 m^3/s (100-year recurrence interval flood). The proposed Finney Creek Diversion Canal is 2.75 km long and will divert Finney Creek flows South, along the West side of Hat Creek, with discharge to the Hat Creek Diversion Headworks Pond. The design capacity of the canal is 5.5 m^3/s, which is also based on the estimated 1,000-year recurrence interval flood.

2. Perimeter Drainage:

The open pit will be surrounded by approximately 6 km of open perimeter drainage ditches, some of which are illustrated in Figure 6-4. The drain to the North-East will collect runoff from areas of heavy traffic for discharge to sedimentation lagoons North of the mine. North-West of the open pit, an open drain will discharge to the buried drainage pipe located in the conveyor causeway. To the South of the mine there will be three similar drains: the upper South-West perimeter drain, which discharges to the Finney Creek Canal; and the lower South-West and South-East perimeter drains, which discharge to the pit rim reservoir.

3. In-Pit Surface Water Drainage:

Surface water and seepage will be collected in open bench drains alongside bench haul roads. Runoff and seepage from surficial material above the mouth of the mine will flow by gravity to the North end of the pit, where it will be collected and discharged to sedimentation lagoons. Runoff from surficials below the mouth of the mine will be collected by bench drains, discharged to small pump sumps and raised to upper gravity bench drains by portable pumps. The lining of major bench drains will probably be required. Runoff and seepage from coal and bedrock strata in the base of the pit will drain via bench drains to sumps located near the main pit access. Temporary sumps and pumps will be placed in low areas on the floor of the pit to collect and remove accumulations of water. A major system of pumps will be installed on the pit incline. This system will discharge to a leachate storage lagoon to the North of the pit. During Summer, water tankers used for dust suppression on bench and haul roads will be filled directly from sumps within the pit.

4. Dewatering Wells:

A staged program of groundwater withdrawal is planned:

Starting in Year 5: Two systems of wells will be drilled, 25 inside the perimeter, and 10 to 15 outside;

Year 10 to Year 15: A final set of wells will be established beyond the perimeter of the 35-year pit. By Year 15, 75 pairs of wells should have been drilled and be operating, one deep and one shallow in each pair.

Total water yield is expected to be low - an average of 0.017 m^3/s or 1,470 m^3/d (Golder, 1979), and while surface water may be discharged to Hat Creek via sedimentation lagoons, water from wells in coal or clastic sedimentary rock will have to be collected in drainage sumps along with surface runoff and pumped to leachate storage lagoons.

6.3.2.2 South-West Slide Area:

Geotechnical studies have determined that stabilization of the slide areas to the South and South-West will depend primarily on drainage (Golder, 1979). Surface water drainage will be required to prevent the groundwater system re-charging, and sub-surface drainage to drain or de-pressurize the groundwater.

1. Perimeter Drainage:

Two diversion drains will minimize surface runoff from small creeks and watersheds at the back of the slide. The North Slide Diversion will be a 1.5 m^3/s capacity open drain 1.7 km long, discharging to the West perimeter drain near the South-West corner of the Houth Meadows Waste Dump. The South Slide Diversion will be an 0.75 m/s capacity open drain 1.2 km long, discharging to the North end of Finney Lake.

Diversion drains will either be fully lined or lined on the downstream side with a layer of impermeable soil to minimize seepage.

2. Surface Drainage Within The Slide Area:

The system will drain approximately 62 small lakes and ponds by improving natural drainage channels and deepening outlets. Drainage will be carried out prior to coal production.

The slide area uphill and to the West will be drained to the West Perimeter Drain via two secondary drains - one draining the existing lake chain, the other draining the series of hollows above the active slide area.

Draining the active slide area will require deepening and improving existing channels down the slide, which will drain to the surface water collection system at the North end of the upper pit benches and ultimately discharge to the North valley sedimentation lagoons. The area to the South and South-West contains a system of lakes and hollows, the existing channels of which will require deepening and improving. The area downhill of the South-West Perimeter Drain will be drained by a secondary drain system joining Finney Creek at its diversion point.

3. Well System:

Provision has been made, for a 20-well system and three km of collector piping, which would be buried to allow for a year-round use (Golder, 1979).

6.3.2.3 Houth Meadows Waste Dump:

1. Perimeter Drainage:

During construction, surface water from the Upper Houth Meadows Watershed will be diverted around the dump via the West Perimeter Diversion. This diversion consists of a 5 km x 8 m wide open drain, with discharge to a buried pipe 2.2 km in length in the conveyor causeway. This pipe will drain into Hat Creek, North of the mine.

The diversion is designed to carry the 1,000-year flood, and a typical cross-section is shown in Figure 6-4. The channel will be unlined on minor gradients. On steeper gradients, a riprap lining will be laid to prevent scour. Though icing may occur, no special design configurations are deemed warranted.

Two further small perimeter drains will be constructed on the North slopes of Houth Meadows, which will discharge to the Marble Canyon Watershed.

2. Drainage of Lakes:

Approximately 20 small lakes and ponds within Houth Meadows will be drained before dump construction. Since these lakes are expected to be high in nutrients, their draining would be carried out during freshet, in order to prevent enrichment of creeks.

3. Surface Water Collection:

During construction of the dump, it is expected that the surface of the waste will be undrainable and that the precipitation will be trapped and lost primarily to evaporation. Minor drainage below the perimeter drains will be collected by an open drain and discharged to the North valley sedimentation lagoons by a buried pipe in the conveyor causeway. During operation of the waste dump, this drain will dispose of surface water from the conveyorway and service roads. Drainage from the re-claimed dump surface will be channelled to this drain by small diversion dykes or swales.

4. Leachate Collection:

Leachate from the main waste embankment will be collected by a line of perforated subsoil drains and discharged to the leachate storage lagoon in the North valley. Monitoring of water quality downstream may be advisable to determine whether de-watering wells should be installed to return leachate to the dump surface for disposal by evaporation.

6.3.2.4 Medicine Creek Waste Dump:

1. Perimeter Drainage:

The Medicine Creek Valley would be extensively used by this project. The powerplant reservoir would be constructed in the Eastern portion of the upper valley. Canals would be constructed to collect runoff from the downstream area, directing it to the reservoir starting in Year 1. Powerplant ash would be dumped in the valley immediately adjacent to the reservoir. In Year 16, mine waste would be dumped in the valley, but starting from the Western end.

During the first 15 years, runoff and seepage from the ash disposal area would be collected in the valley bottom and pumped to a powerplant holding pond for use in dust control. Normal runoff in the lower valley would enter the Hat Creek diversion directly. Once mine waste disposal commences, two minor sidehill drains will be constructed to direct small amounts of runoff occurring below the major collection canals.

2. Surface Water Drainage:

A special collection system will be constructed to collect runoff and treat it for sediment control before discharge.

3. Leachate Collection:

Leachate will be collected by a perforated subsoil drain and discharged to a leachate storage lagoon for Summer disposal by spray irrigation on the active dump surface.

6.3.2.5 Coal Blending Area:

This covers an area of 22 ha and consists of four stockpiles totalling 15 ha. A compacted till blanket overlain by a pervious sand and gravel drainage layer will form the foundation of the stockpiles. Surface water and leachate will be drained to the North-West perimeter, from where it will be collected and piped to a leachate holding pond for temporary storage before final disposal by re-cycling for dust control within the mine.

6.3.2.6 Low-Grade Coal Stockpile:

This should consist primarily of claystone material with a varying percentage of coal, which will be compacted as it is placed. The permeability will, therefore, be low. Non-active stockpile surfaces will be covered by a non-sodic buffer material and suitable surface soil for re-planting. Runoff and leachate will be collected in a sump and discharged to a leachate lagoon.

6.3.2.7 Topsoil Storage Areas:

Surface water will be diverted from the upper perimeters by small ditches to minimize erosion. The stockpile surface will be progressively re-planted, which will both minimize erosion and avoid contamination of downstream surface water.

6.3.2.8 Mine Services Area:

To collect surface runoff from the Mine Services Area, yards will be sloped to open drains at the perimeter, and drainage around buildings will be handled in buried stormwater drains. Drainage will be channelled West to the main sedimentation lagoons via primary treatment to remove sediment and oil.

6.3.2.9 Mine Roads:

Major roads in the North-West and North-East quadrants of the mine area and within the pit will drain to sedimentation lagoons for primary treatment. Roads to the South will drain to a temporary sedimentation lagoon. Temporary construction and haulage roads will drain to the Medicine Creek Sedimentation Lagoon via a buried conduit beneath the Hat Creek Diversion Canal.

Small service and access roads will drain to local watercourses by sidehill drains. Particular care will be taken to limit erosion and scour by the use of stable drains and by early re-planting of disturbed areas.

6.3.2.10 Sewage:

Sanitary effluent from the Mine Services Area will be biologically treated and directed to the Zero Discharge System where it will be re-cycled to dust-control use in the mine. Provision has been made for treating up to 140 m^3/d.

6.4 WASTEWATER DISPOSAL

6.4.1 Discharge Objective:

To protect the environment in compliance with applicable government regulations, the quality of water discharged from the Hat Creek Mine should be within the British Columbia Ministry of the Environment Pollution Control Board's 'Level "A" Effluent Discharge Guidelines for the Mining Industry'.

6.4.2 Projected Quality of Mine Drainage:

Chemical analyses of groundwater from surficial materials would seem to indicate that it is of very similar quality to that of Hat Creek during low flow periods. Hence drainage and seepage from surficials is considered suitable for direct discharge except for sediment control.

Based on present data, seepage and well-drainage from bedrock is expected to be unsuitable for direct discharge. Projections of water quality from various sources are given in Table 6-4.

1. Slide Area:

Drainage from the wells will have high suspended solids concentrations. As a consequence, surface water and drainage from the wells will require sedimentation if the bentonitic slide debris is disturbed.

2. Waste Dumps:

Runoff from waste is not expected due to the hummocky nature of the dumped waste surface. During the reclamation of waste dumps, non-sodic materials would be added to the dump surface; runoff from these areas would need to be treated for sediment prior to discharge.

Tests of leachate from waste materials has shown it to be of a quality unsuitable for discharge to surface waters.

3. Coal-Blending Stockpiles:

Leachate will be unsuitable for discharge due not only to high concentrations of chemical contaminants, but also to low pH. Runoff and leachate will be virtually inseparable due to the semi-pervious nature of the stockpiles.

4. Low-Grade Stockpiles:

Leachate will contain roughly the same level of contaminants as in the coal-blending stockpiles. Runoff will probably be unsuitable for direct discharge.

5. Disturbed Land:

Projections have been made on the basis of previous mining experience. Runoff from stripped or disturbed land will contain high concentrations of suspended sediment. Average sediment yield may increase by a factor of three. Experience in North Dakota has shown that, even after re-planting, erosion rates may remain high. Sedimentation lagoons should therefore be kept in service until sediment has fallen to acceptable concentrations.

6. Mine Services Area:

Washdown water may contain high concentrations of oil, grease, coal fines, and suspended sediment.

6.4.3 Proposed Treatment to Meet Discharge Objective:

6.4.3.1 Zero Discharge System:

6.4.3.1.1 General:

Seepage and leachate flows of quality unsuitable for discharge from, for example, the pit, waste dumps, coal stockpiles and sewage treatment plant, will be stored in a "Zero Discharge" lagoon system and evaporated in Summer-time by re-cycling the water for dust-control operations on coal stockpiles and pit roads. The surplus will be used for spray irrigation on the active surfaces of waste dumps. An annual water deficit will occur at the mine site ranging from 170 mm to 350 mm, according to elevation. To take advantage of this evaporative potential, storage is required to hold back winter leachate discharges. To this end, a large lagoon will be constructed at the bottom of Hat Creek Valley, which will store 99% of the annual leachate production. A smaller secondary lagoon at Medicine Creek will store the other 1%.

6.4.3.1.2 Inflow, Outflow, and Lagoon Capacity:

The selection of the required capacity depends on three factors: the acceptable risk of a leachate spill; the quantity and time distribution of annual inflow; and the quantity and time distribution of annual outflow. In this feasibility study, sufficient capacity has been allowed to cope with the maximum projected groundwater flow plus twice the projected mean inflow from surface runoff. In practical terms, the worst flood envisaged has a 3% chance of exceeding lagoon capacity during the lifetime of the mine. Flows from smaller, disturbed watersheds will probably vary over a greater range, and an annual probability factor of between one and two percent is likely to be representative of the risk.

Three additional safety factors should be considered:

1. The bulk of inflow is pumped from the lower pit under the control of operations staff. When excessive inflow is likely to occur, it may be possible to store leachate in sumps in the bottom of the pit until capacity is available in the lagoon;

2. The increasing volume of inflow over the mining period requires a system which grows. Provision can be made to bring forward planned increments to lagoon capacity when called for, or deferred should the reverse happen;

3. In the unlikely event of a spill, the flow would be discharged back to the mine, as in (1) above.

Taking all factors into account, the chances of a spill are almost negligible.

Hydrographs of projected inflows and outflows to and from the zero discharge lagoons have been prepared for Years 5, 15 and 25.

The following conclusions have been drawn:

Year 5: A total lagoon capacity of 200,000 m^3 is required. In mean years, a water deficit for dust control of about 120,000 m^3 will exist, which will require make-up water from sedimentation lagoons. In an extreme year, all inflow could be consumed by dust-control operations in one year;

Year 15: A total lagoon capacity of 360,000 m^3 is required. In a mean year, inflow will exceed dust-control outflow requiring spray irrigation on a dump area of about 5-10 ha. In an extreme year, approximately 100 ha of spray irrigation would be required to empty the lagoon before the next season;

Year 35: A total lagoon capacity of 560,000 m^3 is required. In a mean inflow year, 50-60 ha of spray irrigation will be required, and in an extreme year 200-210 ha.

Based on these data, the proposed scheme at Hat Creek is both feasible and manageable.

6.4.3.1.3 North Valley Lagoon

The North Valley Lagoon will cover an area of up to 9 ha and be constructed in the bottom of Hat Creek Valley near the confluence with Houth Creek. The proposed layout features zoned earthfill dams at each end of the lagoon which can be raised in three 5 m stages to elevation 845 m. A further 5 m increase in dam height to 850 m has been allowed for as an emergency measure.

Material for dam construction will come from the pit surficials and from the East and West sides of the valley. A buried-membrane lining consisting of two metres of till overlying a 0.8 mm thickness PVC sheet will be laid on the prepared pond bottom and a lining of 0.8 mm PVC, one metre of till and one metre of sand and gravel will be placed on the pond sides.

The pond inlet and outlet will be at the South end of the pond. The pond outlet will consist of a concrete tower which will house leachate recycling pumps of total capacity 175 l/s. The buried discharge pipeline will supply pond effluent to:

- Sprinkle monitors at the coal-blending stockpiles;
- Water tanker filling points on the North pit incline;
- A discharge point at the top of the low-grade stockpile;
- A discharge point near the South abutment of the Houth Meadows Waste Embankment to service the spray irrigation system required in the latter part of the project.

An emergency spillway of capacity equal to the 1,000-year return period flood will be located on the West abutment of the North Dam; overflow would be directed to the open pit.

6.4.3.1.4 Medicine Creek Valley Lagoon:

The required leachate storage capacity is estimated at 12,000 m^3, which will be created in a small pond of 0.7 ha. This pond will be lined with one metre of till over a 0.8 mm PVC liner, and will allow for expansion above projected storage requirements.

Inflow to the pond will be from field drains at the embankment base, and outflow will be pumped away to be disposed of by spray evaporation on the active dump surface.

An emergency spillway and runoff diversion drains will also be provided.

6.4.3.1.5 Operation:

The Zero Discharge System will require minimum maintenance. Seasonal inspection of the pond lining should be done in late Autumn when the pond level is at its lowest. The selection and maintenance of pumps and piping systems requires care, due to the presence of sediment and potentially aggressive water.

In relation to the pond volume of between 200,000 and 600,000 m^3, the annual sediment build-up in the large lagoon of between 65 to 250 t/a will be insignificant, and the sediment will build up in the pond for the life of the project.

Geotechnical studies have shown that even full saturation of the waste dump surface would not affect the stability of the planned 5% slope, though Golder recommends that the materials near the transfer conveyor should be kept dry in order to improve the stability of the bench on which it operates. In relation to the large storage capacity of the lagoon, spray irrigation at the low rate of 250 mm/a should permit sufficient flexibility to allow satisfactory operation. Measures will be taken to ensure that no conflict arises between spray irrigation and the spreading of waste.

When the active life of the mine comes to an end, the mean annual lagoon inflow will decrease from 470,000 m^3 to 25,000 m^3. The Medicine Creek system will remain in operation until such time as the seepage is considered fit for discharge. Sewage, after biological treatment, will also be dealt with within the Zero Discharge System, and ultimately used for dust control. In the North valley, natural evaporation from the leachate pond will dispose of the residual leachate from the Houth Meadows Dump and the low-grade coal storage area. A flow hydrograph for Year 35 for these systems is shown in Figure 6-5.

6.4.3.2 Sedimentation Lagoon System:

6.4.3.2.1 General:

This is required to reduce projected high sediment concentrations in runoff otherwise fit for discharge. This runoff comes from natural rangeland stripped of soil-cover during construction and operation, pit surficials, permanent stormwater drainage, and re-graded and reclaimed waste dumps.

Two sets of lagoons are required, as shown in Figure 6-4. The first will consist of three lagoons constructed before mining begins to the North of the pit; the second two-lagoon system will be constructed in Year 16 downstream of the Medicine Creek Waste Dump.

6.4.3.2.2 Design Criteria:

The sediment removal efficiency of the lagoon system takes into account the Level "A" discharge objectives of the Pollution Control Board.

During larger flood flows, the efficiency of sediment removal will decrease, but as the natural suspended sediment concentration in Hat Creek itself will rise (specially during freshet), the net effect on receiving water quality should be low.

6.4.3.2.3 Inflow:

An analysis of land use in relation to the size of watersheds has produced the following 10-year 24-hour volumes of runoff:

Year 5 and 15: 78,000 m^3

Year 35 : 91,000 m^3

Annual mean discharges for the lagoons are estimated to total 1,050,000 m^3 in Year 5; 1,093,000 m^3 in Year 15; and 1,181,000 m^3 in Year 35. A breakdown of lagoon inflows for Year 35 is shown on Table 6-5.

6.4.3.2.4 Sediment Tests:

These were carried out by B.C. Research in 1978 and show that only runoff from glacial-fluvial sand and gravel may be expected to satisfy the guidelines without chemical treatment. Alum has been found to be effective as a coagulant where concentration of sediment exceeds the guidelines. It should be noted that only sediment with a substantially higher settling velocity than the design value will be admitted to the lagoons, a measure arising out of the recognition that the use of chemical coagulants should be minimized in order to avoid observed increases in sulphate concentrations.

6.4.3.2.5 North Valley Sedimentation Lagoons:

The three-lagoon system to be constructed North of the pit will consist of a primary sedimentation and flow balancing lagoon of 1.5 ha and two secondary lagoons totalling 4.5 ha. Total storage volume will be 250,000 m^3. The materials for the retaining dams and dykes will be excavated from deposits in the mine area. Test drilling reveals that conditions may be encountered during construction which require that a low permeability till lining be applied to the bottom of the lagoons.

Inflow to the primary pond will be via a stilling basin and inlet manifold, and outflow will be controlled by two decant towers. Inflow to the secondary lagoons will be via a pipe manifold and outflow via an overflow weir. When chemical treatment is required, chemicals will be added at two mixing points.

During high inflow, the two secondary lagoons will operate in parallel; under low inflow, in series. This is designed to improve treatment efficiency and reduce the use of chemical coagulants. An emergency spillway channel will pass flows in excess of outlet capacity.

6.4.3.2.6 Medicine Creek Sedimentation Lagoons:

Two lagoons totalling 1.8 ha will be constructed before stripping operations in Year 15. The system will consist of a small primary and a larger secondary lagoon.

6.4.3.2.7 Lagoon Discharge:

The mean discharge hydrographs for the sedimentation lagoons are show in Figure 6-6. The flood discharge hydrograph following 10-year 24-hour rainstorm is shown in Figure 6-7.

The effect of lagoon discharges on water quality have been assessed for three cases:

Case 1: where, under dry weather condition, Hat Creek would be at its lowest and the main inflow would be from de-watering wells.

Conclusion:

Water discharged will meet Pollution Control Board's "A" guidelines except for a higher sulphate concentration. The total dissolved solids concentration of receiving water will increase by less than 2%.

Case 2: where, under Spring runoff conditions, the main inflow would be from surface water in the lower pit. Hat Creek flows would be high.

Conclusion:

The North lagoon effluent will be suitable for discharge; only the sulphate concentration would exceed level A discharge objectives. Discharges from the pit rim reservoir would meet level A objectives for all parameters except copper which would be less than level B. The total dissolved solids concentration in receiving water would rise by 2%.

Case 3: where, under Summer rainstorm conditions, a large amount of surface runoff may occur in proportion to the rest of Hat Creek Valley.

Conclusion:

These are essentially the same as in Case 2 above, except that the solids concentration in receiving waters would increase by less than 5%.

The greatest increase in sulphate concentration occurs in Case 1, but amounts to only 31 mg/L, increasing from 54 mg/L to 85 mg/L.

Present Canadian standards for drinking water define 500 mg/L as acceptable and 250 mg/L as desirable. The natural concentration of sulphate in Hat Creek near the mine site measures approximately 59 mg/L and 76 mg/L further downstream. Taking all this into account, the lagoon effluent may therefore be deemed acceptable by the regulatory authorities.

The final concentration of copper in the receiving water is well below the acceptable level of 1 mg/L of the Canadian Drinking Water Standards 1968.

6.4.3.2.8 Operation:

To achieve the required discharge water quality, the lagoon system will require careful operation, maintenance, and regular checks and inspections of all components.

The total storage capacity of 100,000 m^3 in the North lagoons and 30,000 m^3 in the Medicine Creek Lagoon is calculated to be greater than the expected lifetime yield of sedimentation of 10,000 m^3 and 500 m^3 respectively. No clean-out will therefore be necessary.

After the mine has closed, the lagoon system will remain in operation until land reclamation has reduced sediment concentration in runoff to acceptable levels. During this time, the stored water may be used for irrigation.

The Mine Drainage Section of this report is based upon the CMJV Mine Drainage Report, October 1979, and has not been adjusted to reflect changes in the 1979 Mining Plan. The economic and environmental effects of such adjustments would be insignificant.

TABLE 6-1

Projected Groundwater Yield From The Mine Development
Hat Creek Project Mining Feasibility Report 1979

	YEAR 5 VOLUME $m^3 \times 10^3$	YEAR 15 VOLUME $m^3 \times 10^3$	YEAR 35 VOLUME $m^3 \times 10^3$
OPEN PIT			
Peripheral Wells	520	520	520
Seepage:			
- Surficials	90	120	120
- Bedrock	20	50	30
Total	630	690	670
HOUTH MEADOWS DUMP			
Embankment Seepage			
- No. 1	9.5	11	11
- No. 2	1.5	3	4
- No. 3	0	2	5
Subtotal	11	16	20
To Regional Groundwater	0.3-3	1.5-15	6-32
Total	11-14	17-31	26-52
MEDICINE CREEK DUMP			
Embankment Seepage	0	4	12
To Regional Groundwater	0	0.3-3	1-6
Total	0	4-7	13-18

Source: Golder 1979 Refer Appendix 2

TABLE 6-2

Design Criteria For Planning Of Mine Drainage System
Hat Creek Project Mining Feasibility Report 1979

Type of Drainage Element	Description	Design Flood	Probability of Exceedence in 35-Year/Mine Life
Major Creek Diversions	Hat Creek	1,000 year*	3%
	Finney Creek	1,000 year*	3%
	Houth Creek	1,000 year	3%
	Upper Medicine Creek	Probable Max. Flood*	---
Perimeter Drains	Around Pit Waste Dumps & Slide Area	100 year	30%
Surface Water Drains within mine development	Permanent Major Drains	100 year	30%
	Temporary Minor Drains	10 year	97%
Leachate Collection Systems	Field Drains	Max. Seepage Rate	---
Dewatering Wells	Collection Systems	Max. Pumping Rate	---
Sedimentation Lagoons	Emergency Spillways	1,000 year	3%
	Treatment Capacity	10 year	97%
Leachate Storage Lagoons	Emergency Spillways	1,000 year	3%
	Storage and Disposal Capacity	2x Mean Annual Flow	---

* Refer BCH HEDD 1978 and Monenco 1977 for Design Criteria

TABLE 6-3

Design Flows for Preliminary Planning of the Mine Drainage System
Hat Creek Project Mining Feasibility Report 1979

Code (as on Schematic)	Description	Watershed Area km^2	Flow Frequency	Flow Type	Estimated Flow m^3/sec	Estimated Volume m^3 x 1000	Data Sources	Remarks
HAT CREEK								
Q1	Hat Creek u/s of mine	248	A	M	0.63	-	1	-
Q2	Hat Creek d/s Medicine Creek	308	A	M	0.67	-	1	$52 km^2$ to PP Res
Q3	Hat Creek d/s of mine	383	A	M	0.72	-	1	-
-	Diversion Canal Capacity	-	1000F	P	27	-	3	Under emergency
DIVERSION DRAINS								
D1	Upper SW pit	2.0	100R	P	0.75	-	1	
D2	South Slide runoff	3.7	100R	P	1	-	1	
D3	South Slide Diversion	1.3	100R	P	1	-	1	
D4	Finney Ck Canal	21	1000F	P	3.50	-	1	
D5	Ambusten + SE Watershed	35	1000F	P	7	-	1	
D6	Pit Rim Pump	4.4	-	P	0.12	-	3	Pump capacity
D7	Medicine Ck Runoff Canal	-	-	-	-	-	-	-
D8	Medicine Ck Runoff Canal	-	-	-	-	-	-	-
D9	East Watershed	2	100R	P	1.2	-	1	
D10	North Slide runoff	1.2	100R	P	0.6	-	1	
D11	North Slide Diversion	4.5	100R	P	1.75	-	1	
D12	West Perimeter Diversion	25	1000F	P	4.2	-	1	
D13	North Perimeter Diversion	1	100R	P	1	-	1	
P1	Lower SW Diversion	1.7	100R	P	0.7	-	1	
P2	SE Diversion	0.5	100R	P	0.5	-	1	
P3	Watershed below Canal	3	100R	P	1.5	-	1	
L1	Canal Leakage		DY	M	0.01-0.025	-	3	
MINE DRAINAGE COLLECTION SYSTEM						24hr Volume		
S1	Houth Meadows Dump	-	10R	P		15	1	Project at max size
S2	Disturbed slide area runoff	100	10R	P	-	6	1	"
S3	Slide dewatering wells		DY	P	-	0.044	2	"
S4	Runoff from Pit Surficials	335	10R	P	-	48	1	"
S5	Groundwater from pit Surficials		DY	P	-	2	2	"
S6	North Valley Services area	200	10R	P	-	20	1	"
S7	Washdown water	-	DY	M	-	0.090	1	"
S8	Medicine Creek Dump	-	10R	P	-	13	1	"
DISCHARGE OF TREATED DRAINAGE								
W1	North Valley Sed. Lagoons		10R	P	0.8	56	1	From Hydrograph
W2	Medicine Creek Sed. Lagoons		10R	P	0.2	13	1	" "
ZERO DISCHARGE SYSTEM						Est. Annual Volume		
Z1	Sanitary Effluent	-	DY	M	0.0016	51	1	700 man shifts/day
Z2	Coal Blending Leachate	0.22	A	M	-	20	1	Project at max size
Z3	Low-Grade Coal Leachate	0.33	A	M	-	16	1	"
Z4	Houth Dump Leachate	-	A	M	-	11	2	"
Z5	Pit Coal & Rock Leachate	-	A	M	-	332	1	"
Z6	Dust Control consumption	-	A	M	-	319	1	"
Z7	Evaporative Disposal	-	A	M	-	129	1	"
Z8	Medicine Dump Leachate	-	A	M	-	12	2	"
WATER SUPPLY SYSTEM								
H1	Mine Services Area	-	DY	M	0.0041	101	1	700 shifts/day + garden + washroom
H2	Reveg Nursery	-	A	M	-	75	1	10ha

Key to Symbols in Table:
100R - 100-year Av recurrence interval rainstorm flood
1000F - 1000-year Av " " rain-snowmelt flood
10R - 10-year Av " " rainstorm flood
DY - Daily
A - Annual
P - Peak Discharge
M - Mean Discharge

Sources of Data:
1 CMJV Estimate
2 Golder Assoc. 1978, 79
3 BCH HEDD, 1978

NOTE:
These data are based on Preliminary Mine Planning Data, Hydrological and Hydrogeological Studies. Surface water flows from small watersheds and seepage flows are estimates based on several arbitrary assumptions as to runoff infiltration factors and hydraulic conductivities. They therefore should be upgraded when further site-specific data becomes available. Where a range of flow is shown, this identifies the variability of flow in terms of the assumptions made. Areas used correspond to the estimated maximum effective area of natural watersheds, disturbed areas, or mine facilities to be drained.

TABLE 6-4

Projections of Water Quality of Mine Drainage
Hat Creek Mining Feasibility Study 1979

		NATURAL SURFACE WATER						MINE DRAINAGE				DISCHARGE GUIDELINES	
Parameter (mg/l)	Hat(1) Creek	Medicine Creek Area	Finney Lake	Aleece Lake	Medicine Creek Dump Runoff	Ash Leachate	Mine Waste(2) Leachate	Coal Leachate	Low Grade Coal Leachate	Slide Debris Ground-water	Pit-water(2) Bedrock	Pit-water(2) Sur-ficials	PCB Level A Objec-tives
pH (units	8.4	8.3	8.2	7.6	8.0–8.5	8.0–9.0	8.1	5.0*	4.6*	8.0	7.8	7.9	6.5–8.5
Filterable Residue	336	275	17.9	N.A.	1900– 2760*	4800– 8900*	1125	8400*	5400*	1070	1950	350	<2500
Non-Filterable Residue	8–	0–110	N.A.	N.A.	>50*	N.A.	N.A.	N.D.	N.D.	N.D.	N.D.	N.D.	<50
BOD_5	<1	N.A.	N.A.	N.A.	<115–150	<35–195	137	N.D.	N.D.	N.A.	N.D.	N.D.	N.D.
TOC	8	19	18	N.A.	N.A.	N.A.	N.A.	N.A.	N.D.	50	50	21	N.D.
Alkalinity	212	221	123	217	332–360	1120– 1260	123	<27	<0.5	570	1185	310	N.D.
Chloride	1.2	0.4	0.5	<0.5	58–61	175–190	27	14	0.88	28	42	4	N.D.
Fluoride	0.14	0.12	0.22	N.A.	0.7–1.1	3.3–4.9*	0.06	0.1	N.D.	0.16	0.2	0.2	2.5
Nitrate (as N)***	<0.06	0.04	<0.02	N.A.	3.5–4.2	2.4–3.3	4.4	N.D.	N.D.	<0.14	<0.06	<0.2	10
Kjeldahl Nitrogen (as N)	0.19	0.26	0.83	N.A.	N.A.	N.A.	N.A.	N.A.	N.A.	<11	14	<0.2	N.D.
Ortho Phosphate (as P)	0.038	0.01	0.025	N.A.	0.27– 0.3	0.14– 0.31	0.3	0.01	N.D.	<0.03	<0.03	<0.03	2
Sulfate	50	20	5	52*	330–350*	1500– 1580*	21	3700*	3800*	380*	<321*	270*	50(3)
Arsenic	<0.005	<0.005	<0.005	N.A.	<0.18– 0.56*	<0.6– 2.4*	0.07*	<0.005	<0.005	<0.005	0.006	<0.005	0.05
Boron	<0.01	<0.1	<0.1	N.A.	<0.7–0.8	<3.0–3.6	0.04	0.31	0.7	<0.21	0.31	<0.1	N.D.
Cadmium	<0.005	<0.005	<0.005	N.A.	0.022*	<0.1	<0.002	N.D.	N.D.	<0.005	<0.005	<0.005	0.005
Calcium (as $CaCO_3$)	145	130	60	85	260–275	1050– 1130	48	1900	1075	208	180	200	N.D.
Chromium	<0.01	<0.01	<0.01	N.A.	<0.13– 0.14*	<0.12– 0.20*	0.13*	<0.01	<0.01	<0.01	<0.01	<0.01	0.05
Copper	<0.005	<0.005	<0.005	N.A.	<1.2– 1.3*	<0.23– 0.33*	1.5*	0.04	<0.007	<0.008	<0.008	<0.005	0.05
Iron	<0.018	<0.02	<0.04	<0.05	<1.4– 1.5*	1.95– 2.05*	1.25*	0.26	<0.01	<0.06	<0.075	<0.031	0.3
Lead	<0.01	<0.01	<0.01	N.A.	<0.026	<0.05	0.02	N.D.	N.D.	<0.03	<0.013	<0.01	0.05
Magnesium (as $CaCO_3$)	74	85	33	100	72–75	220–230	33	2240*	1680*	118	124	116	652(as $CaCO_3$)
Mercury	<0.00038	<0.0005	<0.00033	N.A.	<0.0015– 0.0017*	<0.0013– 0.0023*	0.0015*	<0.0003	<0.0003	<0.0003	<0.0003	<0.0003	0.001
Sodium	20	11	15	38	115–120	325–335	63	190	150	230	412	93	N.D.
Vanadium	<0.005	<0.005	<0.005	N.A.	<0.05– 0.06	<0.18– 0.22	0.01	<0.04	0.006	<0.006	<0.007	<0.005	N.D.
Zinc	0.008	0.009	<0.006	N.A.	0.29– 0.64*	0.82– 2.5*	0.15	0.11	0.18	<0.36	0.52*	<0.03	0.5

SOURCE: Beak 1978, 1979 NOTE: (1) Mean of measurements taken Sept. 1976-1977 during a low flow year.
(2) Surface runoff has been projected to be of this quality (Beak 1979).
(3) Subject to review.
* indicates parameter is in excess of PCB Level A Guideline

TABLE 6-5

Estimated Sedimentation Lagoon Inflow - Year 35
Hat Creek Project Mining Feasibility Report - 1979

Source	Area (ha)	CN	Qm (m^3x10^3)	Q10 (m^3x10^3)	Q35 (m^3x10^3)	Q100 (m^3x10^3)
NORTH VALLEY LAGOONS						
1. Open Pit Mine						
Runoff above EL 900	250	90	200	38	65	100
Runoff below EL 900	85	90	68	(38)10*	(22)10*	(17)10*
Dewatering flow			656**	2	2	2
2. North Valley						
Service areas, roads, and open space	200	85	100	20	38	64
3. Slide Area						
Disturbed land	100	80	50	6	13	24
4. Houth Meadows Waste Dump						
Stripped land	-	-	-	-	-	-
Levelled waste	24	90	12	4	6	10
Reclaimed land	190	80	95	11	25	46
Total North Valley Lagoons	849	-	1181	91	159	256
MEDICINE CREEK LAGOONS						
5. Medicine Creek Dump						
Stripped land	-	-	-	-	-	-
Levelled waste	24	90	12	4	6	10
Reclaimed land	148	80	74	9	19	36
Total Medicine Creek Lagoons	172	-	86	13	25	46

Note:

CN = Curve number for soil cover complex refer Fig.
Qm = Mean annual volume of runoff.
Q10 = 10-year recurrence interval 24-hour runoff volume.
Q35 = 35-year recurrence interval 24-hour runoff volume.
Q100 = 100-year recurrence interval 24-hour runoff volume.

It is assumed that maximum 24-hour inflows will occur during summer rainstorms.
Curve numbers for soil cover complexes have been estimated from literature (USSCS 1964, 1975).
* *Contribution to pond inflow limited by pump capacity.*
** Includes 16,000 m^3 from slide area.

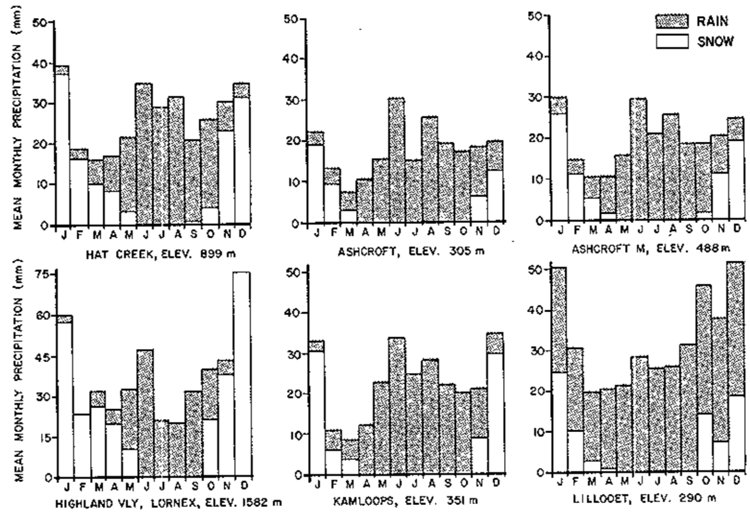
TABLE 6-6

Projected Quality of Lagoon Discharge and Hat Creek - Case III*
Hat Creek Project Mining Feasibility Report 1979

Parameter (mg/l)	Projected Effluent North Lagoon	Projected Pit Rim Dam Discharge	Existing Hat Creek	Projected Hat Creek After Mixing
pH (Units)	8.4	8.3	8.4	8.4
Temperature °C	N.D.	N.D.	N.D.	N.D.
Filterable Residue	376	450	342	435
Non-Filterable Residue	≤50	≤50	95	82
TOC	11	20	9	12
Total Hardness (as $CaCO_3$)	220	196	224	227
Alkalinity (as $CaCO_3$)	223	200	226	230
Chloride	2.3	5.0	1.1	3.6
Fluoride	0.16	0.13	0.16	0.19
Total Nitrogen (N)	<0.43	0.60	0.24	<0.43
Phosphorus (P)	<0.05	<0.06	0.043	<0.05
Sulfate	57	35	54	63
Arsenic	<0.008	<0.019	<0.005	<0.023
Boron	<0.1	<0.09	<0.1	<0.13
Cadmium	<0.005	<0.005	<0.005	<0.006
Calcium (as $CaCO_3$)	140	122	143	146
Chromium	<0.015	<0.03	<0.01	<0.016
Copper	<0.07	<0.26	<0.005	<0.066
Iron	<0.08	<0.23	<0.026	<0.09
Lead	<0.01	<0.012	<0.01	<0.01
Magnesium (as $CaCO_3$)	76	73	77	77
Mercury	<0.0004	<0.0006	<0.0004	<0.0007
Sodium	24	24	20	24
Vanadium	<0.005	<0.006	<0.005	<0.007
Zinc	<0.014	<0.03	<0.007	<0.035

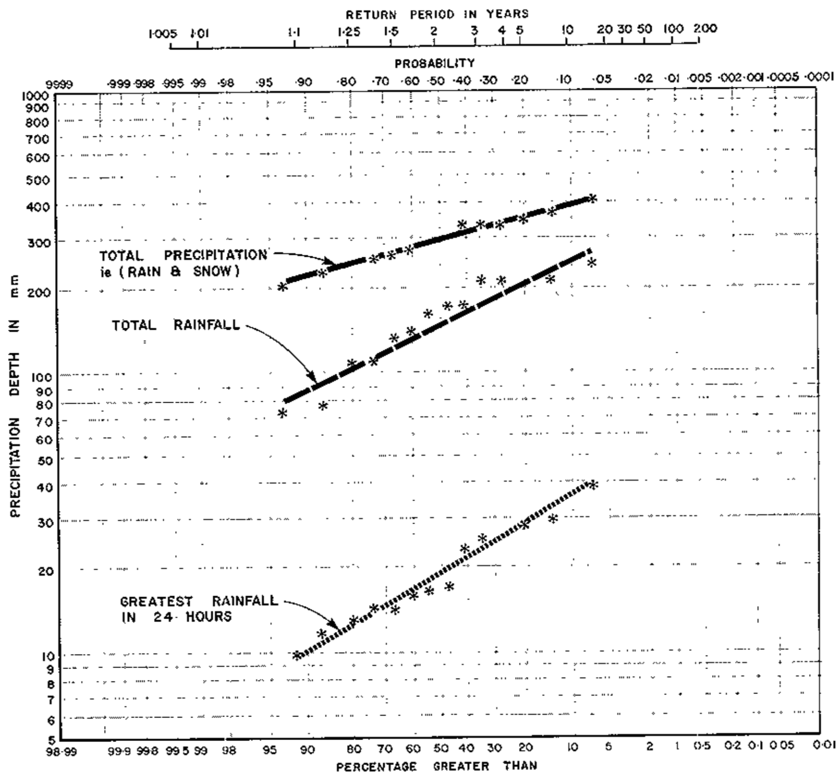
* Summer Rainstorm Condition (Year 35) Discharges to Hat Creek via sedimentation ponds include surface runoff caused by a 10-year 24-hour rainfall, dewatering flows from pit surficials and from the slide area. Hat Creek discharge was assumed to be 1.68 m^3/sec. Surface runoff and dewatering rates are from CMJV estimates. Flow attenuation has been assumed to occur in the lagoons.

(Source: Beak 1979)



SEASONAL DISTRIBUTION OF MEAN MONTHLY PRECIPITATION
IN THE HAT CREEK REGION

from Atmospheric Environment Service
Canadian Normals, 1973



DISTRIBUTION OF PRECIPITATION DATA FOR HAT CREEK VALLEY

from Atmospheric Environment Service, Climatological Station Data

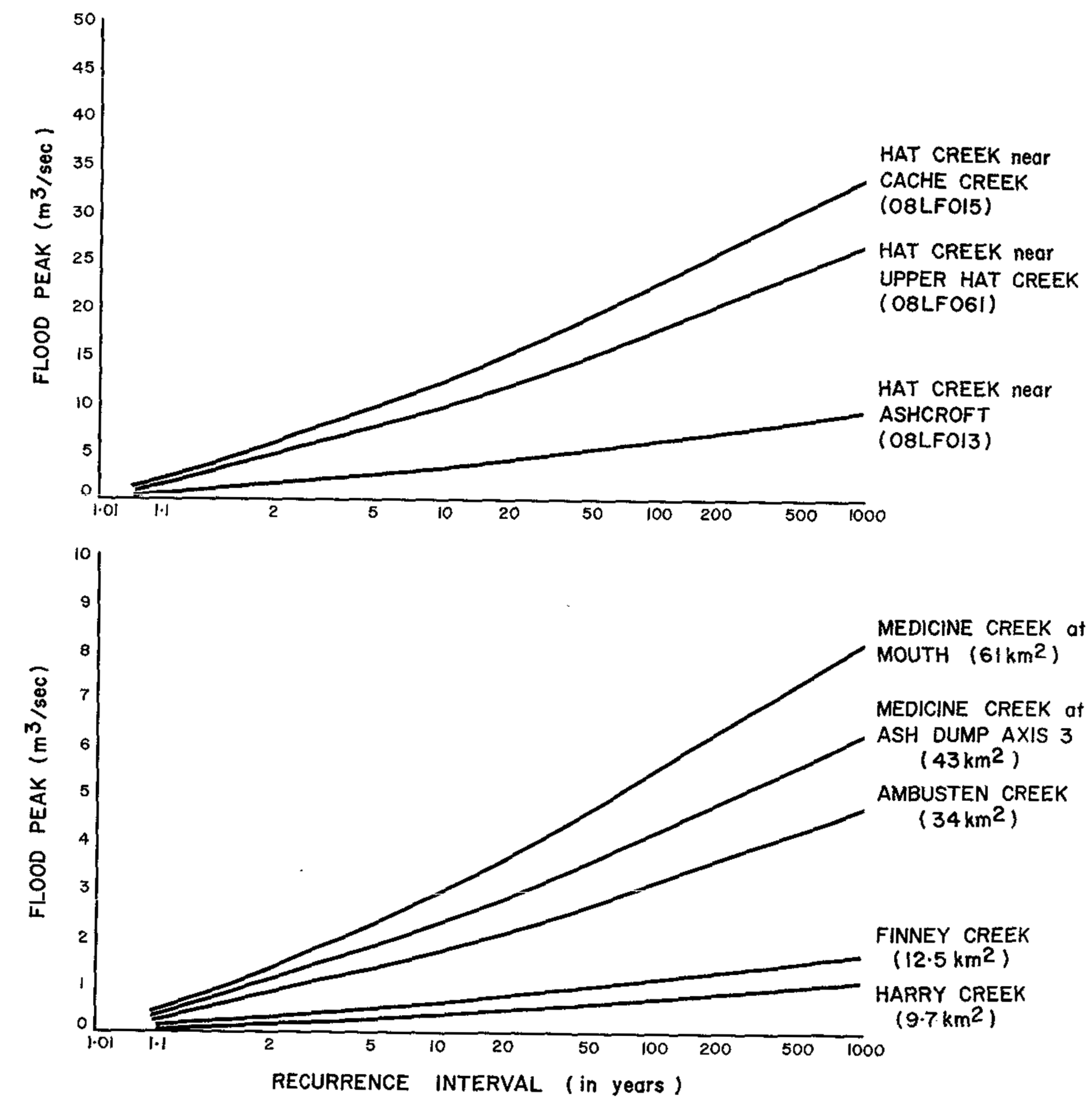
Source: Beak Hydrology Inventory Study,
Draft March 1978

FIGURE 6-1

BRITISH COLUMBIA HYDRO & POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT
PRECIPITATION DATA

cominco-monenco joint venture

FLOOD FREQUENCY CURVES DERIVED FROM REGIONAL DATA



NOTE:
CURVES SHOWN ARE FOR ANNUAL SNOWMELT FLOODS

Source: B.C. Hydro H.E.D.D. Report 913, 1978.

MONTHLY FLOW IN HAT CREEK JUST ABOVE THE MINE SITE

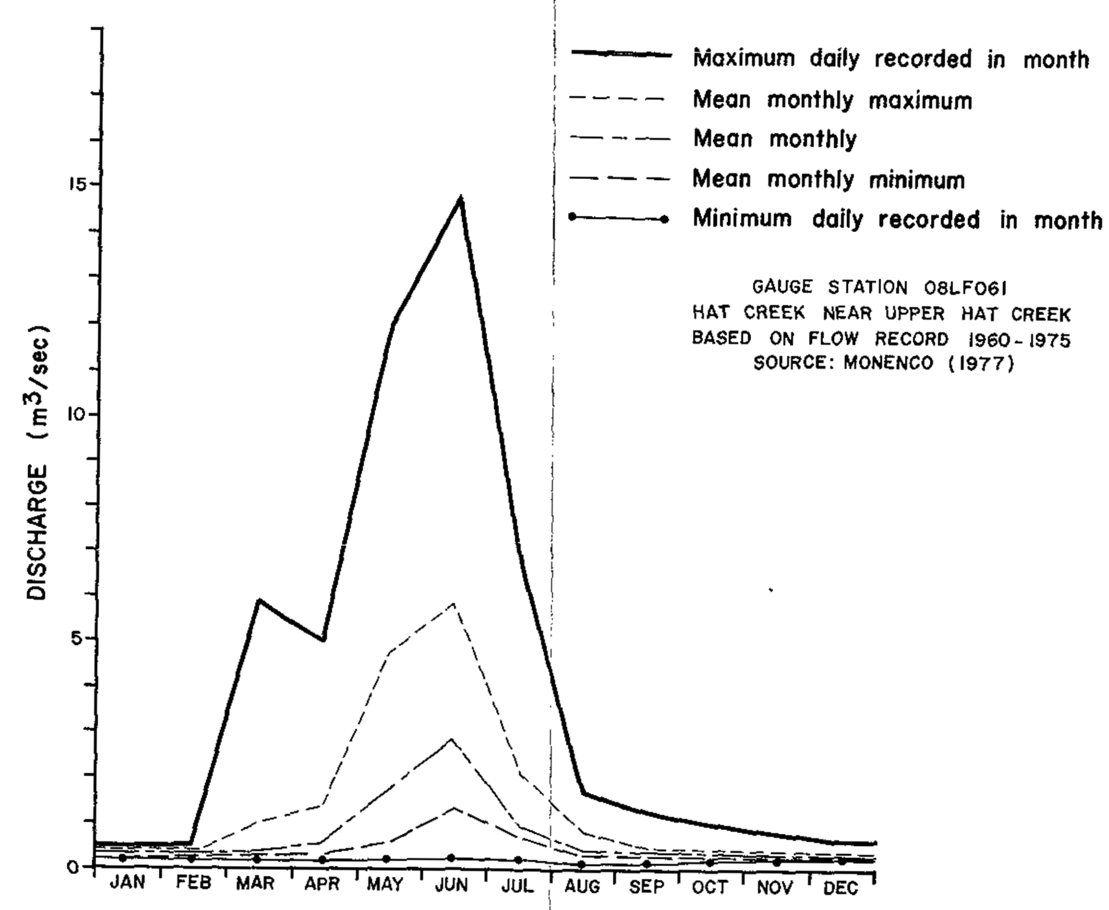


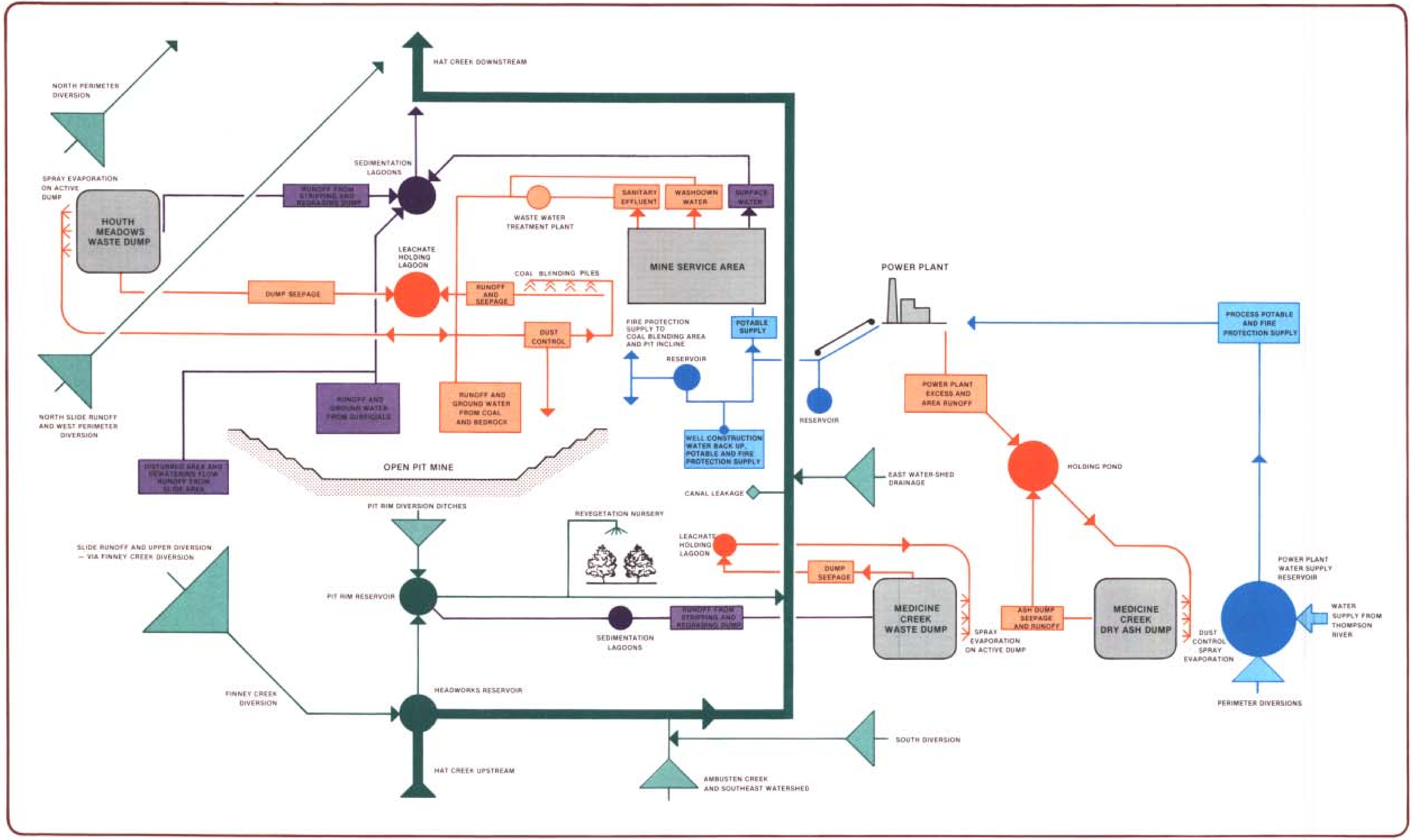
FIGURE 6 - 2

BRITISH COLUMBIA HYDRO & POWER AUTHORITY

HAT CREEK PROJECT

MINING FEASIBILITY REPORT

STREAMFLOW DATA



LEGEND

- PROCESS POTABLE AND FIRE PROTECTION SUPPLY
- HAT CREEK AND NON-CONTAMINATED WATER
- WATER FROM DISTURBED AREAS REQUIRING TREATMENT
- WATER UNSUITABLE FOR DISCHARGE

HAT CREEK PROJECT

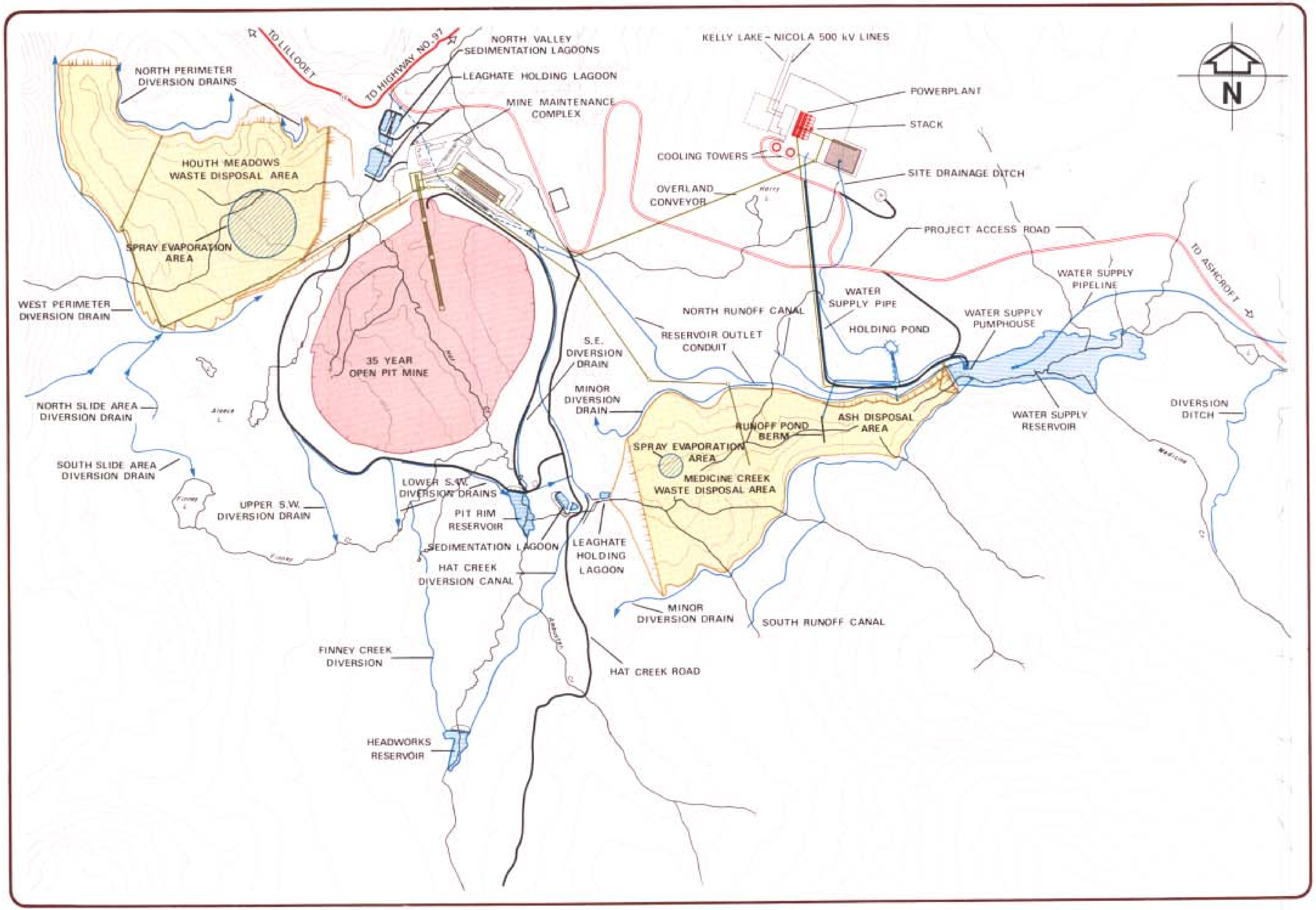
FIGURE 6-3

PROJECT DRAINAGE AND WATER SUPPLY FLOW DIAGRAM

00142 1/2

20

SOURCE: Cominco — Monenco Joint Venture (6)



LEGEND

RESERVOIRS AND LAGOONS

DRAINAGE AND DIVERSION

SCALE — 1:40,000

1 km 0 km 1 km 2 km

CONTOUR INTERVAL — 50 METRES

HAT CREEK PROJECT

FIGURE 6-4

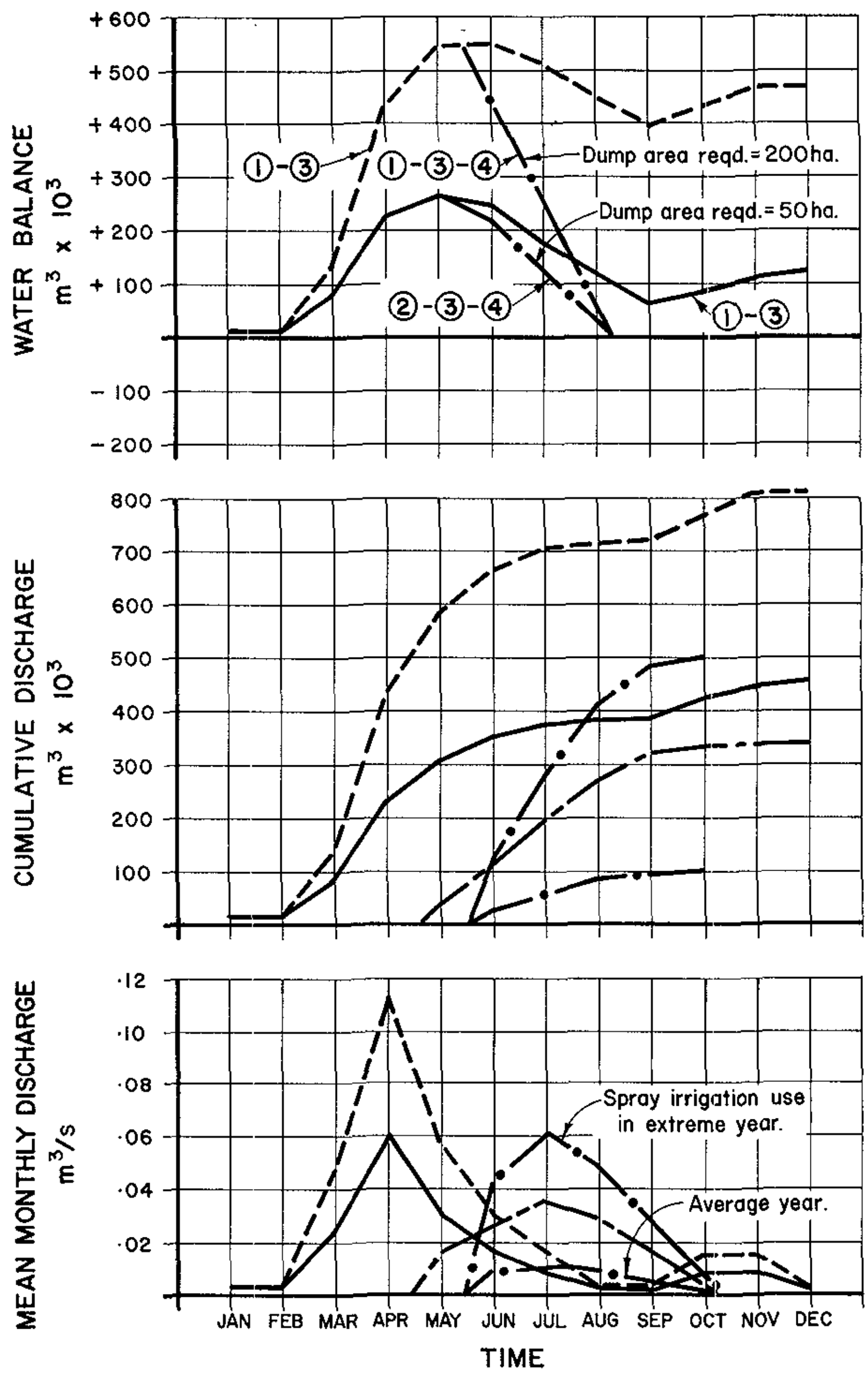
PROJECT DRAINAGE AND DIVERSION SYSTEM

00142 1/2

21

SOURCE: Cominco — Monenco Joint Venture (6)

cominco-monenco joint venture



LEGEND

① ------ EXTREME INFLOW

② ——— MEAN INFLOW

③ —·—·— DUST CONTROL OUTFLOW

④ —•— SPRAY IRRIGATION OUTFLOW

FIGURE 6-5

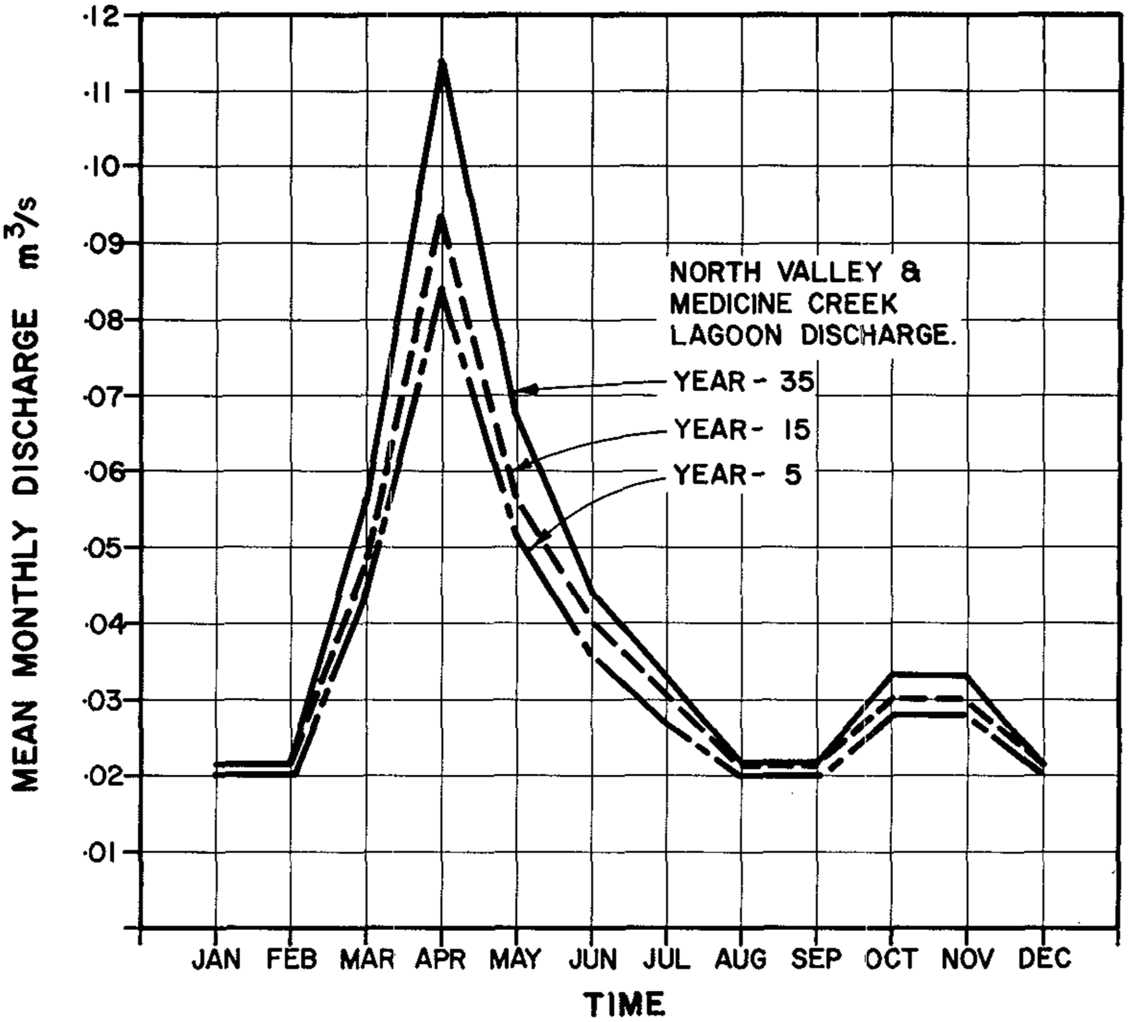
BRITISH COLUMBIA HYDRO & POWER AUTHORITY

HAT CREEK PROJECT

MINING FEASIBILITY REPORT

ZERO DISCHARGE SYSTEM

HYDROGRAPHS-YEAR 35



NOTE

Pond loss to seepage and evaporation plus dust control use early in mine development may reduce summer flows by ·010 – ·025 m ^{3}/s

FIGURE 6 - 6

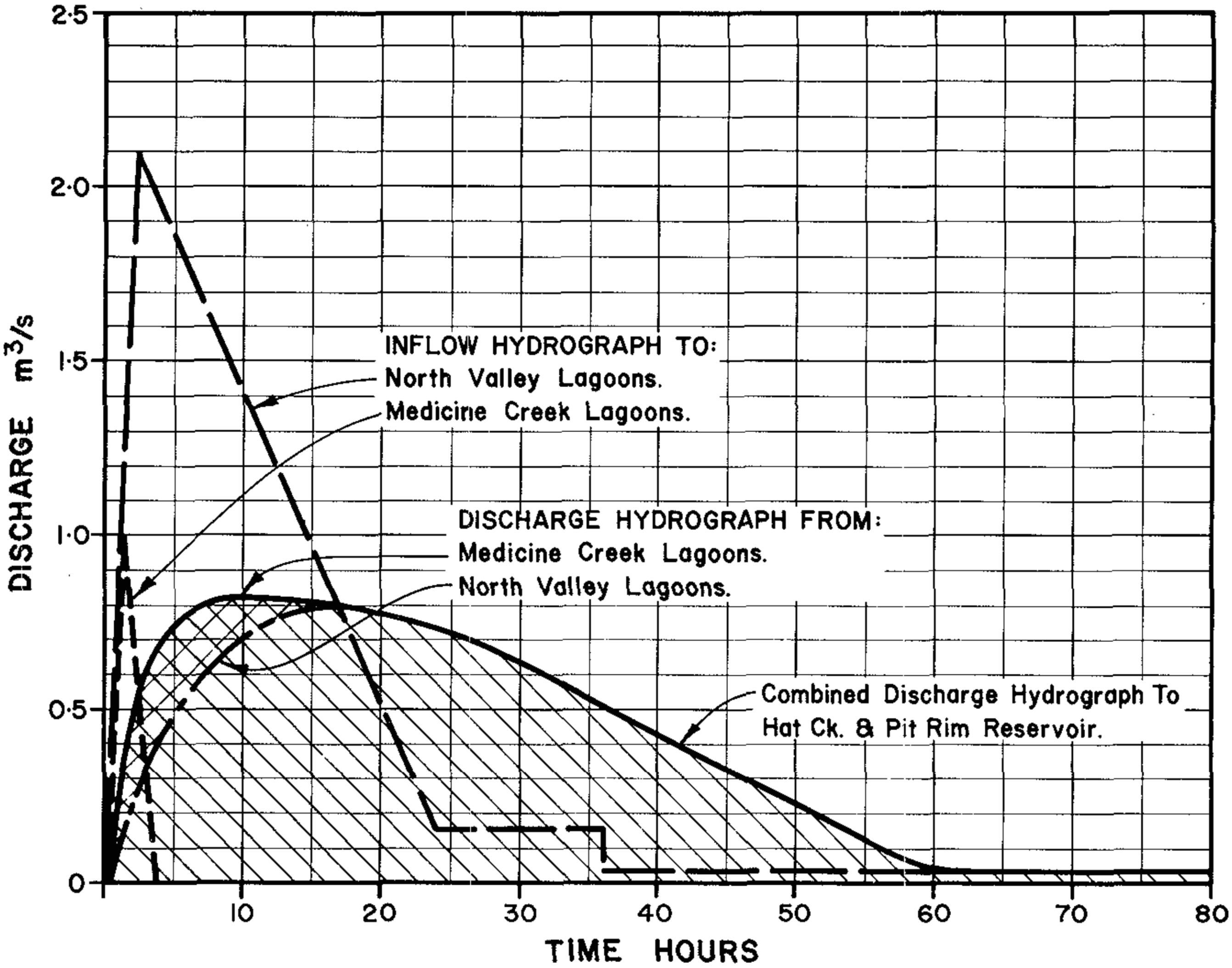
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

MINING FEASIBILITY REPORT

SEDIMENTATION LAGOONS
ESTIMATED MEAN DISCHARGE
HYDROGRAPHS

cominco-monenco joint venture



NOTE

The lagoons are sized on the basis of these hydrographs. Emergency Spillways will be sized for the 1:1000 yr. return period flood.

FIGURE 6-7

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

MINING FEASIBILITY REPORT

SEDIMENTATION LAGOONS

10 YEAR 24 HOUR

FLOOD DISCHARGE HYDROGRAPH

7 FUEL QUALITY

No.	LIST OF TABLES	Page

SECTION 7

FUEL QUALITY

7.1 INTRODUCTION

The quality of the coal to be supplied as boiler fuel has a major impact on the design, economics, and the environment of both the mine and the powerplant. Because of the wide range of variability of the coal in the Hat Creek No. 1 Deposit, it is possible to produce a number of fuels of different quality. As a basis for the selection of the project performance fuel, the following objectives were established:

(1) The performance fuel must be within the design limitations for conventional North American boilers and pulverizers;

(2) A consistent quality of coal within specified tolerance limits must be supplied to the powerplant;

(3) Utilization of the coal resource should be maximized;

(4) Adverse environmental impacts should be minimized;

(5) The energy cost should be minimized. This requires a careful balancing of capital and operating cost factors between the mine and the powerplant.

To meet these objectives, a mining method has been developed that will economically produce performance fuel for the boilers, while providing for a high level of resource utilization and minimizing environmental risk.

A sequence of mining plans and production schedules developed for the anticipated life of the powerplant demonstrate that fuel of a consistent quality - 18.0 MJ/kg - can be produced by selective mining within a tolerance of 1.0 MJ/kg. To smooth out short-term fluctuations in the fuel quality, a comprehensive blending stockpile and reclaim facility is planned. The mining plan is flexible; it will always permit access to higher quality, low-sulphur coal when necessary to cope with predicted short-term sulphur dioxide excursions beyond regulated ambient levels.

The beneficiation of coal by washing was studied at length and rejected on technical, economic, resource utilization, and environmental grounds.

7.2 HAT CREEK COAL

The quality of the coal in the Hat Creek coal deposits varies over an unusually wide range. The reasons for this are presented in Section 5.3.8.1, which discusses how the coal was deposited and how the coal formation grades from good coal through low-grade coal to clay.

It is difficult to present a clear cut classification system that consistently and accurately describes the different grades of coal, except on the extremes of the range. The good coal is shiny, black, thinly bedded, hard, and breaks with a glassy conchoidal fracture. This coal is typical of the D-zone coal, particularly in the D3 subzone, contains approximately 20% ash, and has a heating value of 23 MJ/kg. At the other extreme, the carbonaceous claystone is a soft, grey to dark-grey, earthy clay matrix with finely disseminated carbonaceous particles, and has greater than 80% ash and a heating value less than 2.3 MJ/kg. Between these two extremes there is a complete spectrum of coal quality developed by an increasing frequency of partings in the good coal from one end of the scale, and of an increasing percentage of carbonaceous particles in the clay matrix from the other.

For example, cut-off grade quality (9.3 MJ/kg, 59% ash) coal could occur in two different ways:

(1) Bands of equal thickness of good coal and pure clay;

(2) A massive low-grade coal band with 50%, by volume, high-quality carbonaceous particles in the clay matrix, or by some combination of these.

Table 7-1 presents a broad classification of Hat Creek coal and the principal related characteristics.

It is important to recognize the nature of the variability of the Hat Creek coal and the numerous zones of transition between coal and waste. These factors have a major impact on the quality of run-of-mine coal and on the processes that can be applied to improve the quality of fuel supplied to the powerplant.

A review of the data presented in Table 7-1 is helpful in understanding how the decision between fuel and non-fuel material was made. Categories 4, 5, and 6 were rejected because they contain a very high proportion (>73%) of non-combustibles: ash and moisture. Including such poor material in the fuel reduces the boiler efficiency, increases wear and tear in pulverized-coal-fired boilers, and creates handling problems in the powerplant coal system.

Category 3, the low-grade coal, was considered marginal fuel for the boilers and is discussed further in Section 7.5. The 35-year design pit contains 21.7 million tonnes of low-grade coal. The inclusion of this quantity with the boiler fuel would increase the total heat available by 2.7%, but would be accompanied by an increase of 11.9% in the quantity of ash to be processed through the boilers and disposed of.

The fuel selected for the boilers is a blend of coal from categories 1 and 2, produced by selectively mining bands of fuel and non-fuel materials down to 2 m in thickness. The resulting fuel over 35 years will average 18.0 MJ/kg with 33.47% ash and 0.51% sulphur (dry-coal basis) and 23.5% moisture content. The non-combustible content of this fuel is slightly less than 50%.

7.3 COAL BENEFICIATION

Coal beneficiation is a broad term which includes any process that improves the quality of coal. In dealing with boiler fuels, this generally implies raising the heating value and reducing the ash content of the coal. Beneficiation, however, can also be used to reduce the moisture or sulphur content. The majority of the proven beneficiation processes in use are wet, gravity-separation processes. Dry processes have been used in the past, and new dry processes are under development.

An extensive program of investigations into coal beneficiation has been completed and is outlined below.

7.3.1 Testing Programs

The initial investigations into coal beneficiation were directed towards establishing the characteristics of the proposed beneficiation plant feed and the performance of coal samples in standard laboratory washability tests. Data from these tests were used to predict the performance of the coal in various beneficiation processes. Larger samples of the coal were then processed through pilot-scale beneficiation plants. The results of these pilot plant operations were used to validate the predictions made from the laboratory tests and to develop plant design criteria.

In 1976, three bulk samples of Hat Creek coal were obtained by drilling a series of 0.91-m diameter bucket-auger holes. These three samples represented coals of different quality: 13.2, 18.1, and 20.2 MJ/kg (dry-coal basis). A portion of each sample was tested in the laboratory of Birtley Engineering to determine the size distribution of the material and to establish the sink-float characteristics. The results of this testing form the basis for the prediction of performance in gravimetric processes.

The remainder of the three bulk samples was crushed to -20 mm. The (20 mm by 28 mesh) fractions were cleaned, using heavy-media cyclones, and the -28 mesh fractions using water-only cyclones. In the heavy-media process, the clay coated the media, creating density-control problems and high magnetite loss. Part of the raw and washed coal samples were shipped to CCRL Ottawa for pilot-scale burn tests.

In 1977, three samples were obtained during the bulk sample program: two from Trench A and one from Trench B. Particular care was taken in obtaining three samples to ensure that they represented "as mined" coal rather than the finer coal obtained using the bucket-auger. These samples were sent to Warnock Hersey Professional Services, Calgary, for a laboratory testing program designed by Simon-Carves Canada Ltd. This program was essentially similar to that conducted in 1976, except that a wet attrition test, based on an Australian standard method, was introduced to permit the anticipated degradation during processing to be evaluated in the laboratory.

A 73-t sample obtained from Trench A during the bulk sample program was submitted to the Western Research Laboratory of Energy Mines and Resources, Edmonton, for evaluation of its beneficiation performance in their compound water cyclone pilot-plant. A second objective of this program was to evaluate the production and treatment of the liquid tailings effluent.

7.3.2 Conclusions Drawn from Test Results

(1) Hat Creek coal is subject to severe breakdown in water, especially where there is attrition. The clay particles from the coal form a suspension which can interfere with gravity-separation processes;

(2) Washability data show that the degree of beneficiation achieved would be relatively low for the effort expended; approximately half the normally expected improvement would be gained;

(3) The finer size fractions have increasingly difficult washability characteristics. Since all cleaning processes are less efficient for the finer size fractions, the overall efficiency of any process treating the fine size fraction would be abnormally low;

(4) The finer size fractions have increasingly higher ash content. This would limit the effectiveness of a commonly used process for thermal coals where washed coarse coal is blended with unwashed fine coal;

(5) The better quality (D-zone) coal should not be washed, because the small improvement in quality would not offset process losses;

(6) The tailings produced by any process-washing of Hat Creek coal would be largely a clay-water suspension, which would be extremely difficult and costly to dewater. The quantity of tailings produced by any process would be dependent on the size of the material and the duration of contact between the coal and water;

(7) There would be some reduction in the sulphur content per unit of heating value of the coal through washing, with resulting lower powerplant sulphur emissions;

(8) Practical beneficiation plants could be designed and operated to clean the Hat Creek coal and their performance could be predicted with reasonable confidence from laboratory tests;

(9) The design of a practical tailings disposal scheme would require pilot-plant work and further research.

7.3.3 Alternative Beneficiation Processes Considered

A wide range of possible beneficiation processes were reviewed in the light of the results of the test programs and the process characteristics. The processes were evaluated on the basis that only coal from the A, B, and C-zones would be washed, while the better quality D-zone coal would be blended with the wash plant product. The plant feed would be divided into coarse and fine fractions by screening at a nominal 13 mm. Six practical plant schemes were selected for evaluation:

(1) Heavy-media bath (coarse coal) and water-only cyclone (fine);

(2) Heavy-media bath (coarse) with untreated fines;

(3) Baum jig (coarse) with untreated fines;

(4) Untreated coarse with dried and classified fines;

(5) Water-only cyclones for coarse and fine coal which would require crushing coarse coal to -40 mm. This scheme would be similar to the EMR pilot process;

(6) Heavy-media bath (coarse) with dried and classified fines.

For each scheme a preliminary modular plant design was prepared and capital and operating cost estimates made. Predictions of plant performance were made based on the available test data.

7.3.4 Tailings Disposal

The disposal of tailings from a beneficiation plant received very close attention, because of the known difficulty experienced elsewhere by the tarsand, phosphate, diamond, and china clay operations in dealing with tailings with a high clay content.

The concentration of clay particles would build up in the plant process water to a level that is unsuitable for use. Under natural conditions, the clay settles very slowly. Under lagoon storage conditions, it is anticipated that over a period of years natural sedimentation would produce a sludge with 40% solids. Any further improvement beyond this level would be extremely slow, requiring many years. The settling can be accelerated by the use of flocculants, which will produce a layer of relatively clear water for re-use in the process and a settled layer with a solids content of up to 40%. However, there are indications that the use of flocculants limits the long-term compaction that can be achieved.

The only possible alternative to lagoon sedimentation and storage is mechanical dewatering by the application of solid-bowl centrifuges. Laboratory work on Hat Creek tailings conducted at EMR, Edmonton, indicated that a cake of 75% solids material could be produced. Operating plant experience suggests that a 45% solids product is a more realistic estimate. For the total beneficiation schemes evaluated, approximately 50 million m^3 of 45% solids sludge will be produced over 35 years.

The physical handling and disposal of this material presents some difficult problems. One method of disposal is to convey the sludge with the wash plant solid discard material to the Houth Meadows Waste Disposal Area, a distance in excess of 2 km. This would create conveyor problems - especially in sub-zero temperatures. Testing would be required to ensure that the sludge-solid discard mixture can be conveyed up 10% gradients. The alternative method of sludge disposal is by storage in a lagoon similar to that provided for the sedimentation process, although in this case the lagoon would be smaller.

Of the two alternative methods for sludge disposal, only the lagoon sedimentation and storage approach can be considered proven and practical. There are some serious drawbacks to using this method: lack of a suitable storage space; the cost of building retaining structures for the lagoons; and the possible permanent alienation of the land in the storage area should it prove impossible to reclaim.

The mechanical dewatering process would require further research and testing, particularly on the performance of centrifuge equipment and the handling and disposal of sludge, before it could be proposed with any confidence. Should dry disposal of the sludge prove impractical, the mechanical dewatering process could prove to have the same disadvantages as the storage and sedimentation approach and prove more expensive to operate.

7.3.5 Conclusions

An evaluation of the costs and benefits was conducted based upon the estimated capital and operating costs and the predicted plant performance of the selected schemes. The principal conclusions were:

(1) Hat Creek coal can be beneficiated to produce a fuel averaging 21.0 MJ/kg, compared to 18.0 MJ/kg for run-of-mine coal;

(2) Sulphur emissions could be reduced by up to 20-25% using beneficiated fuel;

(3) The disposal of clay tailings remains a major technical and economic problem, with potentially severe environmental impacts;

(4) Resource utilization would be reduced by 5-8% because of process losses to tailings. This is partially offset by improved boiler efficiency; but the remaining losses must be made up by mining additional tonnages of coal at higher marginal stripping ratios;

(5) The estimated capital and operating costs of the beneficiation plant exceed the anticipated savings in the powerplant.

Based upon these conclusions, it was decided to eliminate beneficiation from further consideration in the base plan.

7.4 BOILER FUEL SPECIFICATION DEVELOPMENT

7.4.1 Introduction

The boiler fuel specification is a critical project document whose reliability must be assured for the design of appropriate boilers and ancillary equipment for the powerplant. The penalties of a design based on an incorrect fuel specification are severe and include the inability to produce at rated capacity and excessive maintenance costs.

In March, 1979, the Paul Weir Company (Weirco) were retained to review and refine the boiler fuel specification previously developed by B.C. Hydro staff. The scope of the assignment included:

1. Data Assessment

(1) A review of the quantity and quality of the data available for the purpose;

(2) A review of the procedures followed in analysing the data and of the conclusions drawn;

(3) Identification of any requirements for additional testing and recommendations of appropriate testing procedures;

(4) An assessment of bench-quality variability.

2. Fuel Assessment

An assessment of the suitability of the fuel for the design of a large steam generator and identification of any potential problem areas in design and operation.

3. Preparation of Boiler Fuel Specification

Presentation of the coal fuel characteristics and any necessary description in a form suitable for inclusion in a boiler specification document.

7.4.2 Data Assessment

The first phase of the assessment program was an evaluation of the internal consistency of each of the four laboratories used, as well as the ability to reproduce results between the laboratories. This comparison was conducted on nine of the most important characteristic values. As a result of this examination, the results of one laboratory were excluded from further evaluation. Weirco does not believe that this exclusion significantly affects its overall conclusions, because the samples distributed to each laboratory were not concentrated in a limited area and the excluded laboratory's participation was relatively small. During this phase the data base was screened for apparently erratic results.

A series of regression studies were performed during the second phase of the program to establish certain relationships that are typical of Western coals. These correlations were obtained from the data accepted in Phase I:

$$CO_2 - \% = 0.058 \times \%\ \text{Ash} - 0.269$$

(This equation is used to adjust the volatile matter content for CO_2.)

$$\text{Adjusted Volatile Matter} - \% = 48.90 - 0.475 \times \%\ \text{Ash}$$

$$\text{Equilibrium Moisture} - \% = 25.145 - 0.0617 \times \%\ \text{Ash}$$

$$\text{As Received Moisture} - \% = 28.439 - 0.1566 \times \%\ \text{Ash}$$

A series of tightly controlled determinations of the Hardgrove Grindability Index (HGI) at approximately 10% moisture were made on coal samples with varying ash content. Weirco calculated the following exponential curve as the best fit for the data:

$$\text{HGI} = 24.40\ e^{0.02 \times \%\ \text{Ash}}$$

Because of Weirco's previous experience with the under-reporting of the alkali content of Western coals, a number of samples from each sub-zone were analyzed by two methods: the standard and a modified method. On an overall average basis, Na_2O was under-reported by 36.4% and K_2O by 17.0%. Based on these results, the alkali-content data was adjusted. These adjustments eliminated most of the undetermined error from the analytical data.

The third phase of the assessment program was the preparation of a series of data summaries for use in preparing the final boiler fuel specification. These summaries were prepared initially on a zone-by-zone basis and then on a composite basis, where each zone is weighted in proportion to its contribution to the designed pit. In developing the data summaries, the regression equations were used to adjust the volatile matter, the HGI, and the ultimate analyses except chlorine, sulphur, and ash.

Concurrently, Weirco also examined the mining plan to evaluate its impact on coal quality. The principal conclusions drawn were:

(1) Core examination indicates that the run-of-mine coal quality can be upgraded by selective mixing practices. Material exceeding 60% ash content should be excluded to the maximum practical extent;

(2) No further allowance should be made for dilution, because the sampling procedures have included significant quantities of waste material with the good quality coal. This included waste could not be eliminated in the evaluation of selective mining;

(3) The *short-term fluctuations are the daily or weekly swings in* quality which are a function of where the coal is being mined from a given bench or series of benches. On a weekly basis, the dry-ash content can probably be controlled to approximately ±1.5 percentage points, which equates to a heating value range of ±0.6 MJ/kg. The daily fluctuations would be approximately double the weekly range.

7.4.3 Fuel Assessment

7.4.3.1 Testing Programs

To establish the feasibility of burning various qualities of Hat Creek coal and to develop design parameters for full-size boilers and their associated equipment, two test programs were undertaken. The initial program was on a pilot-scale research boiler, followed by a bulk burn test in a small commercial unit.

Pilot-scale Testing

Pilot-scale testing was conducted in the research boiler at the Canadian Combustion Research Laboratory (CCRL) in Ottawa.

Six samples of Hat Creek coal were tested along with a coal of known performance from Sundance, Alberta. The Hat Creek samples were obtained from the bucket-auger drilling program and consisted of three raw samples and three washed samples obtained from the test-washing program conducted by Birtley Engineering.

The principal conclusions and comments reported were:

(1) Hat Creek coals having a heating value of 13.9 MJ/kg or more, on an equilibrium-moisture basis, can be successfully burned using conventional pulverized-fired technology. This heating value is equivalent to approximately 18.1 MJ/kg on a dry-coal basis. However, in the design of steam generators for this coal, it is imperative that reliable facilities be provided for removing the large quantities of ash that would be produced;

(2) All three samples of raw Hat Creek coal burned during the program produced stable flames without support fuel;

(3) The three samples of washed Hat Creek coals generally produced hotter, more stable flames than the raw coals. The removal of much of the extraneous clay by washing facilitated handling and drying noticeably. Reactivity was also improved;

(4) High clay and moisture content in the Hat Creek coal makes handling difficult. This problem could be minimized by drying the coal to less than equilibrium moisture.

The results of the CCRL pilot-scale tests were considered in the planning of the bulk burn test at Battle River.

Bulk Burn Testing

The principal objective of the burn test was to monitor the behaviour of Hat Creek coal of a quality at or near the anticipated minimum acceptable level in a commercial scale powerplant, and to obtain data needed for steam generator and ancillary equipment design. Key parameters observed included:

- coal-handling;

- pulverizer performance;

- combustion characteristics (flame stability and ignitability);
- slagging and fouling characteristics;
- ash-handling;
- precipitator performance.

The burn tests were conducted in Unit No. 2, a 32 MW (nominal capacity) unit at the Alberta Power Ltd. (APL), Battle River Station near Forestburg, Alberta, during August, 1977.

In order to establish with confidence a lower limit for the practical burning of Hat Creek coal, the fuel selected for the test burn was below the minimum recommended by CCRL. The coal used in the test averaged 15.2 MJ/kg on a dry-coal basis, with individual tests being successfully run on samples as low as 13.0 MJ/kg. The "as received" moisture content was 21.8% (see Table 7-2).

The bulk burn test provided important practical data to establish the reasonable minimum quality of Hat Creek coal that can be used as powerplant fuel.

7.4.3.2 *Comparison with Other Plants*

In assessing the suitability of Hat Creek coal as a boiler fuel, it is useful to examine the design fuels for other powerplants. The Brazos Plant, San Miguel, Texas, has a 400 MW (net) unit scheduled for commercial service in early 1980, fuelled by raw lignite.

Table 7-2 compares some of the principal characteristics of the San Miguel fuel with Hat Creek performance coal and the fuel tested at Battle River.

Considering the results of the burn test and the San Miguel design fuel, the proposed Hat Creek performance coal appears to be well within the range of boiler technology and provides a reasonable basis for design.

7.4.4 The Boiler Fuel Specification

The boiler fuel specification is used to design the steam generator and also to establish a reference point for evaluation of manufacturers' performance guarantees. This is the average, or performance, fuel for the project. The second fuel that is of major significance to the project is the low-sulphur or MCS coal.

The specifications for these fuels is presented in Table 7-3.

The performance fuel is the normal product that the mining operation is designed to deliver at all times, except for a small percentage of the time when high-grade, low-sulphur coal is required for implementing the Meteorological Control System.

The size distribution of the fuel that will be delivered to the powerplant silos is a significant factor in pulverizer design. Estimates of the size distribution have been developed from the results of laboratory and field crushing tests. Table 7-4 presents two estimates of size distribution: The first is for the normal coal flow from the blending pile to the silos, and the second is for coal subjected to long-term storage and compaction prior to utilization.

7.5 LOW-GRADE COAL

Low-grade coal is a fuel of marginal quality that should not be incorporated into the powerplant fuel unless it can be improved. It is defined as having a heating value between 7.0 and 9.3 MJ/kg and an ash content of 59-66% (dry-coal basis). At a moisture content of 20-22%, it contains between 68% and 72% non-combustible materials. As discussed in Section 7.2, low-grade coal is not a simple, well-defined material, but occurs as the result of a combination of many different depositional conditions. Within the designed 35-year pit there are 21.7 million tonnes of low-grade coal averaging 63.5% ash content and 8.0 MJ/kg.

There are two alternatives available for improving the quality of the low-grade coal: washing and dry beneficiation. The wet process was quickly eliminated from consideration because of its cost, the low recovery, and the magnitude of the tailings problem that would be created. It is estimated that low-grade coal would produce three times the volume of sludge per tonne washed compared to run-of-mine coal.

Based on observations of results obtained during dry screening tests, the theory was postulated that a limited degree of beneficiation could be achieved by screening low-grade coal at 13 mm or 20 mm and discarding the undersize.

Tests were conducted on low-grade samples available in the bulk sample trench. These tests indicated that some improvement could be achieved, and a possible plant layout was developed (see Section 8). However, there are some reservations that must be eliminated by further testing before committing the construction of this plant:

(1) The results are based on limited samples and are not necessarily representative;

(2) The moisture content has a major influence on the efficiency of the screening;

(3) The performance of the soft, massive, silty coal in screening is not known.

To resolve these questions will require testing materials from greater depth in the deposit when access to them can be gained.

In the plan presented in this report, it has been assumed that a low-grade dry beneficiation plant will be constructed and its costs incorporated into the cost projections. No allowance has been made for the recovery of additional heating value. Should further testing prove that the process is not practical, the material-handling system will be revised to circumvent the proposed plant. Without this plant, there are four options for disposal of the low-grade coal:

(1) Use as a raw material for an alternative use such as alumina production;

(2) Disposal as waste;

(3) Stockpile for possible alternative uses;

(4) Incorporate with the run-of-mine fuel, should experience prove that no serious problems would be created in the boilers.

7.6 FUEL QUALITY CONTROL

7.6.1 Introduction

The fuel supplied to the powerplant must maintain a consistent quality in heating value to permit stable boiler operation, and in sulphur content to meet emission standards. This consistency must be achieved over both long-term and short-term periods. The ability to meet the quality requirements over the life of the project has been established in developing the mine plan and production schedule. This work showed that on an annual basis, the 18.0 MJ/kg can be produced with a tolerance of 1.0 MJ/kg and that the 0.51% sulphur content can be met with a tolerance of 0.05%.

Having established that control can be maintained in the long-range plan, short-range control can be achieved through the selection of appropriate mining systems and the design and implementation of planning and monitoring procedures.

The key to reducing short-term fluctuations in coal quality is to smooth out the variations that occur in nature. The selected mining methods and equipment make this practical. The application of selective mining techniques eliminates much of the poor-quality material from the fuel. The number and size of shovels selected ensure that in normal operation coal can, and will, be mined from multiple locations of varying qualities. There will be some mixing of coals from different mining locations through the conveying and crushing systems. The blending scheme is specifically designed to provide a stream of reclaimed coal to the powerplant with minimal variation from the mean of the blending pile. All of these factors combine to form an effective variance-reduction system.

7.6.2 Control Program

The control program has two primary elements: planning and monitoring. During operations, each week's production will be planned and scheduled to deliver the quantity and quality of coal required to the blending plant. This coal will be laid down in a blending pile to be reclaimed to meet the powerplant's fuel requirement for the succeeding week. In a typical week, the production requirement will necessitate in excess of 30 shovel-operating shifts. These shifts will be scheduled based on the quality of coal available to meet the required average over the week. The stacker will normally lay this material down in 100 windrows to ensure that the variability of the reclaimed fuel is minimized. The reclaimer recovers the coal, taking slices perpendicular to the direction in which the pile was constructed.

The key to being able to prepare useful weekly production schedules is the ability to predict the quality of the coal to be mined. Based on the data available from the diamond-drill holes at 150 m spacing, the heating value for an individual block of coal can be predicted with a standard error of 5%, and the sulphur, which is more erratic, has a standard error of prediction of 10-12%. When a number of different blocks are combined, as in a weekly production schedule, these standard errors would be reduced.

While this level of predictability is very good at this stage of the project, it can be improved upon considerably as more data becomes available as the mine is opened up. As the mine develops, it is planned to acquire additional data through geological mapping, in-fill drilling, face sampling and monitoring actual production to improve quality predictions to a high level of reliability.

Provision has been made in the design of the material-handling system for continuous ash monitors, which, when integrated with signals from the weightometers, can produce a record of the status of the blending pile. Composite samples will be collected once or twice a shift for laboratory analysis to provide verification of the results of the ash monitor. Sulphur monitors are still in the development prototype stage. These would be installed when proven. Until that time, sulphur monitoring would be provided through laboratory analysis of the composite samples, which could be taken more frequently, should it prove necessary.

The monitoring results on a shift or daily basis provide an opportunity for comparing actual versus forecast quality, which is useful for improving the prediction process and for initiating modifications to the current week's production schedule where required. The monitoring data would be a key item on daily production reports to management. This system provides timely data for corrective action and control.

The quality of coal reclaimed and conveyed to the power-plant will be monitored in a similar manner on the Overland Conveyor as a confirmatory check on quality.

7.6.3 Special Operations

The mine will produce two qualities of fuel: performance coal and low-sulphur coal. The low-sulphur coal will be produced only to meet the requirements of the Meteorological Control System, which is designed to eliminate unfavourable ambient sulphur dioxide concentrations. It is estimated that these conditions will occur about 2-4% of the time on a seasonal basis. The low-sulphur coal will be produced from the D-zone, which is characterized by its low sulphur and high heating value. The D-zone represents approximately 40% of the coal to be mined over the project life. When the production of low-sulphur coal is required, this coal would bypass the coal-blending facility and be conveyed directly to the powerplant. During normal operations, it would be necessary to keep one of the coal shovels in D-zone coal to control the sulphur content in the blended performance coal. One of the coal shovels will be diesel-powered for added mobility, and this shovel can be relocated to any required quality of coal to replace a shovel that is inoperative or at other times of low output.

During the early years of operation, before four generating units are on stream and the coal production is limited, there is some concern that coal quality can be controlled within acceptable tolerances. To provide assurance that the tolerances can be maintained, the coal-stacking system has been designed to permit blending piles to be built in 200 passes instead of the normal 100 passes.

TABLE 7-1

Classification of Hat Creek Coal

Grp.	Cat.	Physical Character	Chemical Data (Dry Basis) % Ash	% Moisture	HHV MJ/kg	Equiv. Coal Cont. %
1	good coal	shiny black, hard, thinly bedded, light	<30	25	19.0+	90+
2	coal	black to brownish-black, moderately hard, well bedded, moderately light	30-59	22-24	9.3-19.0	50-90
		cut-off				
3	low-grade coal	black to dark-grey, hard but slightly softish, thickly bedded, light but heavier than the above	59-66	20-22	7.0-9.3	40-50
4	silty coal	dull black to dark-grey, soft, massive and earthy, relatively heavy	66-72	20	4.7-7.0	25-40
5	coaly clay-stone	dark-grey to grey, soft and weak when wet, rubbly when dry, earthier and heavier than the above	72-80		2.3-4.7	10-25
6	carb. clay-stone	grey, soft and very weak when wet, sheared when dry, very massive and earthy texture, heaviest	>80		<2.3	<10

TABLE 7-2

Comparison of Hat Creek and San Miguel Fuel Characteristics

Parameter	San Miguel Design Fuel	Hat Creek Battle River Test Average	Hat Creek Performance Coal
Heating value - as received MJ/kg	11.6	11.9	13.7
- dry basis MJ/kg	16.6	15.2	18.0
Moisture content (%)	30.0	21.8	24.0
Ash content - as received (%)	28.4	33.6	25.4
Weight of ash/heat input kg/GJ	24.4	28.3	18.5
Weight of water/heat input kg/GJ	25.8	18.4	17.5
Weight of coal/heat input - as received kg/GJ	86.0	84.3	73.0
HGI	92	44	45

TABLE 7-3

Boiler Fuel Specification

	Performance Coal		Low-sulphur Coal	
	Dry-coal Basis	As Received	Dry-coal Basis	As Received
Moisture %				
Equilibrium	-	23.1	-	23.6
As Received	-	23.5	-	24.5
Proximate Analysis %				
Ash	33.5	25.6	24.6	18.6
Volatile Matter	33.0	25.3	37.2	28.1
Fixed Carbon	33.5	25.6	38.2	28.8
Ultimate Analysis %				
Carbon	46.2	35.3	54.3	41.0
Hydrogen	3.6	2.8	4.0	3.0
Nitrogen	0.9	0.7	0.8	0.6
Chlorine	0.03	0.02	0.02	0.02
Oxygen (by difference)	15.4	11.8	16.0	12.1
Sulphur Forms %				
Pyritic	0.13	0.10	0.04	0.03
Sulphate	0.02	0.01	0.02	0.02
Organic	0.36	0.28	0.24	0.18
Total	0.51	0.39	0.30	0.23
Higher Heating Value - MJ/kg	18.1	13.85	21.3	16.08
MAF Basis	27.2	-	28.3	-
Hardgrove Grindability Index (at 10% moisture)	45.0	-	38.0	-

...continued...

	Performance Coal	Low-sulphur Coal
Mineral Analysis of Ash %		
SiO_2 } Acid	52.6	54.1
Al_2O_3 } Acid	28.3	27.5
TiO_2 } Acid	1.0	1.0
Fe_2O_3 } Base	8.5	7.2
CaO } Base	3.4	3.9
MgO } Base	1.5	1.2
K_2O } Base	0.7	0.4
Na_2O } Base	2.1	2.9
P_2O_5	0.2	0.1
SO_3	1.8	2.0
Mn_3O_4	0.2	0.2
V_2O_5	0.1	0.1
Base Acid Ratio	0.197	0.189
T_{250} °C	1500	1510
Water Soluble Alkalies %(dcb)		
Na_2O	0.51	0.64
K_2O	0.069	0.026
CO_2 % (dcb)	1.8	1.2
Fusibility of Ash °C (Range)		
Reducing - Initial Deformation	1170-1500+	1160-1500+
Softening	1210-1500+	1200-1500+
Hemispherical	1250-1500+	1230-1500+
Fluid	1290-1500+	1270-1500+
Oxidizing - Initial Deformation	1310-1500+	1330-1500+
Softening	1330-1500+	1340-1500+
Hemispherical	1340-1500+	1350-1500+
Fluid	1360-1500+	1360-1500+

TABLE 7-4

Size Consist - Powerplant Feed

Size mm	Normal Coal Weight %	Stored Coal Weight %
50-25	10	7[1]
25-13	16	15
13-6	17	16
6-3	15	15
3-1.5	13	10
1.5-0.6	14	12
0.6-0	15	25
Total	100	100

[1] Effective top size 40 mm or less.

B.C. HYDRO AND POWER AUTHORITY
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MINING REPORT — Volume 2 (of 2)

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Copy No.

2

VOLUME 2

Page

8 MATERIALS-HANDLING

Page

VOLUME 2

LIST OF TABLES

VOLUME 2

No. LIST OF FIGURES Page

8 MATERIALS-HANDLING

No.	LIST OF TABLES	Page

LIST OF FIGURES

LIST OF FIGURES (Continued) Page

SECTION 8

MATERIALS-HANDLING

8.1 INTRODUCTION

This section of the report describes the various materials-handling systems employed in the different areas of the Hat Creek Project, i.e. mining, powerplant, waste, and ash disposal. These systems receive and deliver the different materials to their respective destinations, bearing in mind the environmental impacts, the operational requirements, the characteristics of the material, and costs as well as the safety and reliability of all components. The systems, described in Section 8.2 "Coal-handling", and Section 8.3 "Waste-handling" (including "Ash-handling"), have been developed by the following groups:

(1) Mining and Waste-handling - Cominco-Monenco Joint Venture, Simon-Carves of Canada Ltd., and B.C. Hydro Mining Department;

(2) Powerplant Coal-handling and Ash-handling - Integ-Ebasco/B.C. Hydro Thermal Engineering Department.

The different materials to be handled, and their destination points, are noted below:

Coal

With a range of heating values, above cut-off grade, i.e. +9.3 MJ/kg (d.c.b.), containing varying amounts of waste materials such as clays and carbonaceous shales not removed separately in the mining process: This material will be delivered to the stockpile for blending prior to delivery to either the powerplant silos or the powerplant storage area.

Low-sulphur Coal

Better-quality coal from D-zone of the deposit, with lower sulphur content, will be delivered to the stockpile areas for blending, or direct to either the powerplant silos or to the powerplant storage area.

Low-grade Coal

Generally below cut-off grade, i.e. 7 MJ/kg - 9.3 MJ/kg (d.c.b.), containing large amounts of waste materials: This material will be delivered to a dry beneficiation plant for up-grading, with acceptable material being delivered to the stockpiles for blending with the coals described above. Reject material with high ash content will be routed to the waste dumps. Provision can readily be made to divert the reject material to alternative uses if and when these develop.

Waste

The run-of-mine wastes can generally be classified as: (1) construction-grade material, i.e. sands and gravels for construction of retaining embankments; (2) general mine waste, i.e. clays, shales. The materials will be delivered to Houth Meadows, and in later years to Medicine Creek, for construction of embankments and dumps. Some waste materials will also be required for other construction requirements, i.e. road construction.

Ash

Removed from the powerplant as fly-ash and bottom-ash: This will be delivered to a dry-ash disposal area in Medicine Creek.

With this range of materials, there will be many different handling problems which the design has taken into account. Their handling properties will be greatly affected by Summer and Winter conditions; moisture content could also pose problems. The gradual development of the mine and the phased installation of fixed equipment, such as conveyors and crushers, will allow the design of mining equipment to be modified as experience proves necessary.

8.2 COAL-HANDLING

8.2.1 Coal-handling System Requirements

The requirements of the Coal-handling System are:

(1) To provide a reliable supply of coal to the powerplant silos at a consistent quality, as defined in Section 5.2.1, "Powerplant Requirements", and at the quantities shown in Table 5-6;

(2) To supply the required daily tonnage of performance coal, based upon an 18 out of 24-hour silo-filling sequence at full load conditions;

(3) To provide a reliable and readily available supply of higher quality lower-sulphur coal to meet the Meterological Control System or plant upset conditions;

(4) To handle the clays in the coal, bearing in mind the climatic conditions;

(5) To allow for the considerable variation in the run-of-mine coal quality.

8.2.2 Coal-handling - Design Criteria

The coal-handling systems have been designed to the following criteria:

(1) A bulk density for conveyor capacity calculations of 800 kg/m^3;

(2) A maximum slope for conveyors of 14°; for capacity calculations, a surcharge angle of 25° for materials, and a 35° troughing angle for conveyor idlers; all conveyors designed to start-up under full load conditions.

(3) The Conveyor Equipment Manufacturers' Association Handbook for belt load, friction factors, power calculations, and so forth.

The individual conveyor capacities are discussed in the relevant sections.

8.2.2.1 Design Features

Many important design features have been incorporated into the system to handle the variety of materials and to provide both safety and reliability.

Further test work on the materials must be carried out before detailed engineering of these systems can proceed. This is discussed further in Section 8.5.

All chutes, transfer hoppers, and surge bins will be designed with slopes to reduce the incidence of material sticking to the walls. Conveyor transfer and loading points will be installed on the horizontal wherever possible, and impact rollers will be incorporated for conveyor belt protection. Skirtboards will also be fitted to conveyor transfer points to provide for safe and effective load control. This is especially important on inclined conveyors carrying lump material. Magnets and metal detectors will be installed to protect equipment from damage by tramp metal.

Dust-control measures will include water sprays for dust suppression and/or dust collection systems at transfer points. Open conveyors will be fitted with dust covers where necessary and practical.

Fire protection systems will consist generally of automatic sprinkler systems in transfer houses and galleries, and fire hydrants in other areas. The Fire Protection System for the coal-handling in the powerplant is part of a comprehensive Fire Protection System planned for the entire powerplant. This greater system has not been engineered in any detail, but it will include, as a minimum, detection and sprinkler-deluge devices for enclosed galleries and underground conveyors with fire hydrants to protect open conveyors and stockpiles.

All stockpiles will be designed to minimize spontaneous combustion. This will be achieved by compacting dead piles and ensuring that live piles are consumed within a short time, usually two weeks.

Comprehensive control schemes will be installed for both the mine and powerplant conveying systems. At the interface of the two areas, special care has been taken to integrate the systems.

The control schemes incorporate certain safety features and devices. These are designed to ensure safe and reliable operation of the conveyor system and early detection of conditions potentially damaging to the conveyors or associated equipment, or causing excessive spillage of material. These include:

(1) Sequential starting and stopping of all pieces of equipment forming one line of transportation;

(2) Chute-plugging switches which detect the blockage of a chute and stop the system;

(3) Safety cords along the conveyors which allow manual stopping of the conveyor line in emergencies;

(4) Side travel switches which stop the conveyor in case of excessive off-centre movement of the belt;

(5) Low-speed switches which stop the conveyor in case the speed drops below normal. These also prevent the start-up of the preceding conveyor until the conveyor on which the switch is installed has reached its normal speed;

(6) Belt-tensioning devices and controls to ensure that the correct operating tension has been reached before loading the conveyor;

(7) Overload protection devices for motors and conveyor belts; this feature will shut down upstream equipment at a predetermined set-point to minimize overload hazards;

(8) Holdbacks to prevent an inclined conveyor from running backwards under load;

(9) Torque-limiting devices to prevent over-tensioning of the belt during start-up.

8.2.3 Project Coal Facilities - Basic Description

The proposed project coal system can be divided into the three operational areas shown on Figure 8-1.

Operation 1: Mining, processing, blending, and storing, adjacent to the mine;

Operation 2: Reclaiming, loading, and delivery to the powerplant;

Operation 3: Receiving, storing, and handling at the powerplant.

Operations 1 and 2 fall under the jurisdiction of the mine. Operation 3 is under the jurisdiction of the powerplant.

The three operations are described in detail below. In the detailed engineering phase, the engineering specifications for all handling equipment will be correlated to permit standardization of major components where practical.

8.2.4 Coal-handling - Mining and Powerplant

8.2.4.1 Coal-handling - Mining

As noted in Section 8.2.3, the mine Coal-handling System consists of two separate operations. The operations and their main components are described below. The total quantity of performance coal to be delivered to the powerplant is 331 x 10^6 t, with a peak annual requirement of 11.64 x 10^6 t in Year 13. Figure 8-2 shows the system flow diagram. The main control room for the mine Materials-handling System will be located in the Coal Surge Bin House.

(1) Mining, Processing, Blending, and Storing (Operation 1)

1. Truck Dump Stations

Three dump stations are proposed for the full capacity mining operations over the life of the mine. Located adjacent to the inclined in-pit conveyors at the Northern end of the proposed mine, the stations, designed to handle all run-of-mine material, will be built in sequence as the pit deepens. The first, near the surface, will be installed during the pre-production phase; the second in Year 8, approximately half-way down the incline; and the third in Year 20, at the bottom of the incline. Two dump stations will normally operate at any one time.

As shown in Figure 8-3, a dump station will consist of a series of separate dump pockets, each designed to handle a separate material. This section deals only with coal; low-grade coal and waste materials are described in Sections 8.2.6 and 8.3, respectively.

Coal loaded into 77-t rear dump haulage trucks by 10.7 m^3 hydraulic shovels will be delivered at a peak rate of 3,200 t/h to coal pockets (M1, M2) at one of the dump stations.

Each coal pocket, with a capacity of 300 t, will receive coal from rear dump trucks. A 600 mm square grizzly, covering the hopper, screens off oversize petrified wood and boulders, which are disposed of by front-end loader and truck. At the planned production rates, two pockets will be required. Initially, these will be installed at the first dump station. However, as the mine deepens and truck haulage distances increase, they will be relocated to the second and third stations, as required. A third pocket (M9) will be required when the second dump station becomes operational. This will allow delivery of coal to two dump stations, giving additional flexibility.

2. Primary Crushing

Run-of-mine coal will be fed from each dump-hopper at 1,600 t/h maximum by an apron feeder, which discharges the coal to a roller screen working in conjunction with a Siebra crusher. The roller screen will allow coal at -200 mm to pass through, while larger pieces are crushed by an overhead crushing mechanism. Uncrushable material, such as petrified wood, lifts this mechanism and passes through to a short rejects-conveyor, which discharges the material to a stockpile for disposal by a front-end loader and truck. Coal at -200 mm will be discharged to a transfer conveyor and delivered to the mine Coal Conveyor. Table 8-1 gives details of the conveyors in this area.

3. Mine Coal Conveyor

The mine Coal Conveyor (M8A), with 1,400 mm-wide steel cord belt and designed to handle 3,200 t/h, receives coal from the transfer conveyors and delivers it via the Conveyor Drive and Transfer House to the secondary screening and crushing plant. The conveyor initially will consist of one flight, with two more (M9A, M10A) being installed in series to follow the mine development sequence as noted above. Table 8-2 gives details of the conveyors. In the event of a breakdown of this system, a back-up is provided by use of the Low-grade Coal Conveyor described in Section 8.2.6. As shown in Figure 8-4, a bypass conveyor (C1) located in the Conveyor Drive and Transfer House will be used to deliver coal from the Low-grade Coal Conveyor (M8D) to the Coal Transfer Conveyor (C2) and to the Screening and Crushing Plant.

4. Secondary Screening and Crushing

Coal received from the mine Coal Conveyor (M8A), or from the Bypass Conveyor (C1), is discharged to a 1,400 mm-wide transfer conveyor (C2) and delivered to a set of four surge bins, as shown on Figure 8-4. Table 8-3 gives details of the conveyors in this area. A rotating chute will distribute the coal feed equally into the bins. Reclaim from each of the 150 t-capacity bins at 1,000 t/h will be by apron feeders (C3A, 3B, 3C, 3D). Each feeder will discharge to a screen-feed conveyor (C4A, 4B, 4C, 4D) delivering to the Screening and Crushing Plant. The coal will be discharged from each delivery conveyor to a two-deck inclined vibrating screen (C5A, 5B, 5C, 5D). The top decks of these screens will be fitted with 50 mm square woven wire surfaces to classify by size at 50 mm nominal. The lower deck will be fitted with mild steel plate and function as a carrying deck.

Screen overflow will be discharged to an impact-type crusher (C6A, 6B, 6C, 6D), sized to handle up to 350 t/h for reduction to -50 mm. Screen underflow will be carried forward to blend with the crusher product. The -50 mm coal will gravitate to a two-way chute for diversion to either of two parallel conveyors (C7A, 7B). One of these two conveyors will also receive coal from the low-grade coal facilities described in Section 8.2.6.

These conveyors transfer to a second pair of conveyors (C8A, 8B), which then deliver the coal to the Sampling House. A further transfer of the coal to a third pair of conveyors (C9A, 9B) will occur in this house. Either of this third pair of conveyors will deliver the coal to the Stockpile Conveyor (C10) in the Blending/Storage Yard, or to the Reclaim/Bypass Conveyor (C12) if the coal is to be delivered directly to the powerplant.

The provision of four separate screening and crushing lines and a second conveyor line to the Blending Stockpile Area ensures maximum reliability.

5. Blending/Storage Piles

The purpose of the Blending/Storage System is two-fold: (1) to smooth out the short-term variations in the quality of coal received from the mine; and (2) to provide a surge capacity in the flow of coal from the mine in case of breakdown of the Overland Conveyor to the powerplant or powerplant problems.

The crushed coal, at -50 mm, from the crushing plant or low-grade coal beneficiation plant, will vary in quality from the cut-off grade of 9.3 MJ/kg (dry basis) to about 22.0 MJ/kg. The average quality of the coal to be delivered to the powerplant will be 18 MJ/kg. Therefore, the blending system selected must be capable of producing performance coal. The quality of coal delivered to, and reclaimed from, the blending piles will be continuously monitored, as described in Section 8.2.8. This will allow the mine coal production schedule to be adjusted to ensure the delivery of performance coal to the powerplant.

Several factors were considered in selecting the size and location of the blending/storage piles, and a study to determine their location was carried out. The study considered the powerplant storage requirements to eliminate unnecessary duplication of facilities, provide a reliable supply, furnish operational control, ensure efficiency of operation, and meet environmental standards.

Earlier studies by the Cominco-Monenco Joint Venture on stockpile size to suit the predicted blending requirement for the varying qualities of coal and on mine dust, were also taken into consideration. The location selected for the piles is adjacent to the mine mouth in an area that has room for expansion.

The selected layout of the facility, as shown on Figure 8-5, comprises two regular blending piles, each with a maximum 300,000 t capacity. Each pile will be equal to about one week's supply of performance coal for full-load operation of the powerplant. Normally, one pile is being built while the other is being reclaimed.

The operating size of the blending piles will vary according to the long-term and short-term coal forecasts of the powerplant. These forecasts enable the mine to schedule coal production and blending pile construction accordingly. Spontaneous combustion hazards will also be reduced by such planning and scheduling. The size of the piles will also vary in the early years of operation as the powerplant construction proceeds to full production from the four generating units, i.e. from 500 MW to 2,000 MW. Quality control of the coal in the piles during these periods is assured by varying the pile construction technique, as given in Section 7.6.3.

The size and configuration of the piles is influenced by the space limitations of the site and the selection of 300,000 t as the size of a blending pile. Other factors include the size and availability of the required equipment, and the stockpile efficiency, which is determined by the ratio of pile length to width. The selection of the blending method is influenced by the method in which a pile is deposited and reclaimed. The Windrow Method, as shown on Figure 8-5, has been selected over the Chevron Method, as the most suitable for Hat Creek. The Windrow Method gives better blending efficiency by reducing particle segregation and reduces dusting potential.

The system will use a slewable, luffing, rail-mounted stacker receiving coal at 3,200 t/h from the Central Stacking Conveyor (C11), 1,400 mm wide, via a travelling-belt tripper. The luffing boom will deposit the coal by the Windrow Method. Normally, the stacker constructs a pile of 100 windrows, but is designed to construct a pile of 200 windrows. The stacker has its travelling speed controlled by a weigh scale, and is in other respects automatically controlled. This enables windrows of uniform cross-section to be built, which allows maximum use of the storage space and gives better assurance of quality. Table 8-3 gives details of the conveyors in the area.

The stacker, after building one pile, slews through 180° and is able to begin building the other pile with a minimum of delay.

Major specifications of the stacker are:

Capacity	- 3,200 t/h
Boom length	- 55 m
Slewing arc	- 200° minimum
Lifting height	- 18 m
Travelling speed	- 3 to 30 m/min.

Normally, all coal arriving from the mine would pass through the Blending/Storage System after crushing. However, as described in Section 8.2.7, a bypass will be provided to allow direct delivery to the powerplant of low-sulphur coal to meet MCS conditions or to replenish low-sulphur storage stockpiles at the powerplant.

As recommended in the CMJV Dust Study, certain features have been incorporated into the Blending/Storage System. As shown on Figure 8-5, the piles will be specially contoured and oriented to minimize dusting potential, considering the prevailing wind directions, i.e. in a North-East - South-West direction. A specially constructed berm along the Southern edge of the piles will provide a windbreak. A dust-suppression system using water guns will also be installed. The stacker discharge boom will be equipped with a telescopic chute to reduce dusting in the stacking operation, and all transfer points on the system will be equipped with dust-suppression equipment.

Normally, because of the short residence time, the blending/storage piles would not be compacted. Provision would be made for compacting if this should prove necessary to prevent spontaneous combustion.

(2) Reclaiming, Loading, and Delivery (Operation 2)

1. Reclaiming and Loading

To ensure the delivery of performance coal in the desired quantities to the powerplant, close co-ordination will be required between powerplant and mine operations. The powerplant requirements for coal of performance quality will be advised in advance, according to short-term and long-term electricity production schedules. Accurate forecasting of coal requirements is necessary to enable the mine to schedule production of the required quantity and quality of coal to the Blending System.

The Reclaiming System consists of a single rail-mounted bucketwheel reclaimer with a reversible shuttle conveyor. The reclaimer is also equipped with a moving rake which moves the coal down the face of the pile to the bucketwheel moving across the face at the foot of the pile. The rake aids in the blending of the coal and allows for a safer operation, i.e. it does not allow undercutting of the pile. As shown on Figure 8-5, the reclaimer reclaims blended coal from one pile, feeding it to the Reclaim/Bypass Conveyor (C12) on the South side of the Blending/ Storage Yard. After reclaiming one pile, the reclaimer travels back to the Eastern side of the yard, where a transporter car transfers it to the other pile which the stacker has built. After reconnection of the power supply, the Shuttle Conveyor will be repositioned and its direction of travel reversed.

Reclaiming operations will then recommence, with the reclaimed coal being fed to the other 1,400 mm-wide reclaim conveyor (C13) on the Northern side of the yard, which will deliver it to a collecting conveyor (C14) 1,400 mm wide feeding the Overland Coal Conveyor. Stacking operations will then resume to rebuild the first pile.

The reclaimer has a maximum capacity of 3,000 t/h. Normal flow to the powerplant is 2,500 t/h, based upon filling the powerplant silos for 18 out of 24 hours. When necessary, the reclaim/ delivery facilities could be operated at up to 3,000 t/h to simultaneously fill the powerplant silos and replenish the dead stockpile at the powerplant after prolonged outage of the Reclaim/Delivery System.

The most important features of the reclaimer are:

- track mounted with reversible operation;

- bridge span between tracks - 51 m;

- number of bucketwheels - 1;

- capacity - variable from 3,000 t/h to 500 t/h.

An emergency back-up system is also provided. This consists of a portable conveyor supplied with coal from front-end loaders and dozers, which will be supplied from mine operations. The coal is delivered to the Reclaim Conveyor via a hopper.

2. Overland Conveyor

A single conveyor in four flights (C15, C16, C17, and powerplant Receiving Conveyor 1) carries coal from the reclaiming area to a Main Transfer House at the powerplant. This conveyor, with steel cord belt, normally operates at a capacity not exceeding 2,500 t/h, based upon an 18-hour silo-filling sequence. The maximum design capacity of the Overland Conveyor, however, is 3,000 t/h to replenish powerplant stockpiles as well as deliver 2,500 t/h to the powerplant silos. Tables 8-3 and 8-7 give details of the conveyor flights and Figure 8-6 shows the layout.

The conveyor is mounted near ground level, with cut and fill sections to suit the land contours. Adequate clearance is provided to permit clean-up of spillage. It passes underneath the project access road in one location. A 5 m-wide road allowance is included alongside the conveyor for inspection and maintenance.

The conveyor is covered to prevent dusting and, in certain areas, a totally enclosed gallery may be used, e.g. where deep snowdrifts can occur. An allowance for such enclosures has been included in the capital cost of the conveyor.

A study of the reliability of a number of overland conveyors has been carried out. However many overland conveyors are provided, there is still a risk that the coal supply may be interrupted. Therefore, as an insurance, a minimum supply of 14 days' coal at continuous full-load conditions will be stored in the powerplant storage yard. This storage facility is described in Section 8.2.4.2. This amount of storage is sufficient to maintain operation of the powerplant for the longest predictable major breakdown, i.e. the complete replacement of one conveyor belt.

Based on this reasoning, a single overland conveyor has been selected. The provision of four flights, with a change in direction occurring at the end of the first and third flights, allows a conservative route for the conveyor line to be chosen. This results in shallower inclines for the conveyor and access roads compared with a direct route, minimizes contact with the highway, and reduces belt tensions to allow selection of proven belts giving better assurance of reliability.

8.2.4.2 Coal-handling System - Powerplant (Operation 3)

General

The powerplant Coal-handling System includes:

(1) A facility for receiving the discharge from the Overland Conveyor System;

(2) A Silo-filling System to deliver coal to the silos above the pulverizers from the Overland Conveyor or from powerplant storage;

(3) Powerplant storage and reclaiming facilities.

Powerplant coal-related design data, including coal requirements, are assembled in the Station Design Manual (SDM) compiled by the powerplant consultant, Integ-Ebasco.

In 18 hours, the Silo-filling System would provide the coal required by four units at full load for 24 hours.

Summary of Components

The main components of the powerplant Coal-handling System, in addition to the Receiving Conveyor (1) from the powerplant perimeter, are:

(1) The Main Transfer House, including a 600 t surge bin and crushers for frozen coal;

(2) Conveyors 4A and 4B from Main Transfer House to Surge Bins 1 and 2;

(3) Surge Bins 1 and 2 in the Auxiliary Bay;

(4) Feeders and conveyors for transfer from Surge Bins 1 and 2 to silo conveyors;

(5) Silo conveyors;

(6) Silos;

(7) Stocking-out Reclaiming Conveyor 18;

(8) Stacker-reclaimer and live storage facility;

(9) Dead-storage facility, mobile equipment, and emergency reclaim facilities;

(10) Powerplant coal-handling control facilities;

(11) Powerplant coal-handling sampling/testing facilities.

Figure 8-1 shows the powerplant Coal-handling System diagrammatically as part of the overall project coal system.

Major features of the coal-handling layout are shown on the plot plan of the powerplant, Figure 8-8.

Figure 8-9 shows the detailed coal-handling diagram for the powerplant.

Table 8-7 lists the conveyors and belt-feeders for the powerplant Coal-handling System.

Description of Components

A description of the powerplant Coal-handling System follows:

1. Final Flight of the Overland Conveyor

The Receiving Conveyor (1) at the powerplant is an extension of the Overland Conveyor and operates as part of that complete system. This conveyor (1) is a single, covered belt, above ground, and running North/South on the East side of the cooling towers. Should the Overland Conveyor System be unavailable for any reason, it does not preclude operating the remainder of the powerplant Coal-handling System or prevent the supply of coal to the silos, because the powerplant has storage under its direct control, as described in items 9 and 10.

Capacities of the Overland Conveyor and Receiving Conveyor (1) are:

Normal maximum	2,500 t/h
Peak capacity	3,000 t/h

Normally, the Overland Conveyor will empty before being stopped. Should the coal on the Overland Conveyor have to be dumped, the silo in the Transfer House is used. The Overland Conveyor can hold about 900 t. Excess coal will be dumped via the Excess Discharge Conveyor (17) to the ground.

2. <u>Main Transfer House</u>

This Transfer House is the main coal receiving and distribution point for the powerplant. It will contain a 600 t surge bin and transfer conveyors for normal delivery of coal to the powerplant silos or, when desirable or necessary, to the powerplant Storage System.

Two 100%-capacity frozen lump crushers, with variable-speed inlet feeders, are included for recrushing frozen coal reclaimed from the storage areas in Winter, if and when necessary. Screens may be included ahead of the crushers after a full evaluation of crusher alternatives. Normally, the crushers are bypassed. Protective devices such as metal detectors are provided.

The Transfer House is heated and includes dust-control and fire-protection facilities.

3. <u>Powerhouse Conveyors</u> (4A and 4B)

Two 2,500 t/h inclined Powerhouse Conveyors (4A and 4B), housed in a common enclosed and heated gallery, carry coal from the Main Transfer House to the Surge Bins (1 and 2) in the Auxiliary Bay of the Powerhouse. These conveyors enter the Powerhouse between Boilers 1 and 2. Normally, one conveyor operates and the other is on standby.

4. <u>Powerhouse Surge Bins and Transfer Conveyors</u>

Surge Bins (1 and 2), each of 100 t capacity, are located respectively between Boilers 1 and 2 and Boilers 3 and 4 in the Auxiliary Bay. Surge Bin 1 is fed directly from the Powerhouse Conveyors (4A and 4B). Surge Bin 2 is fed from the Powerhouse Conveyors (4A or 4B) by two 2,500 t/h Transfer Conveyors (5A and 5B). Normally, the surge bins are fed by either of the inclined Powerhouse Conveyors (4A or 4B) in conjunction with either of Transfer Conveyors (5A or 5B).

5. Powerhouse Surge Bin Outlet Feeders and Conveyors

Discharge from the Powerhouse Surge Bins (1 and 2) is by variable-speed discharge feeders and manually-operated gates. The feeders supplying the adjacent silo conveyors feed direct. Those supplying the outer silo conveyors feed to 400 t/h Intermediate Conveyors (6A, 6B, 7A, and 7B). For each conveyor there is a standby of equal capacity.

6. Silo Conveyors

Over the row of four silos on each side of each boiler, a single silo-filling conveyor (10A/B, 11 A/B, 12 A/B, and 13 A/B) of capacity 400 t/h each delivers coal to a travelling tripper, which fills the silos.

Simultaneous filling of all rows of silos, so that daily coal demand for full-load boiler operations can be completed in 18 hours out of a 24-hour period, is tentatively planned as the operating mode. However, the system design is flexible and allows continuous filling with varying boiler loads.

Individual silos are filled on a "layering" basis.

The silo-filling operation is automated to a reasonable degree, but is under constant supervision from the coal-handling control panel, from which the filling rate can be manually adjusted.

Key signals (e.g. low silo-level alarms) are repeated in the boiler control panels.

7. Silos

Eight silos, four on each side, are provided for each boiler. Each silo feeds one pulverizer. The silos each hold up to eight hours' capacity for one pulverizer at full load with performance coal. Normally, seven mills carry full load. Silos are of circular construction, with conical bottoms of stainless steel with a 78° slope. Manual gates are fitted at each silo outlet, and provision is made for emptying the silo contents in an emergency.

The silo gates, downpipes, feeders, and emergency emptying chutes are part of the boiler.

8. Stocking-out Conveyor

A single 2,500 t/h conveyor (18) feeds from the Main Transfer House to the live storage area. It discharges to the stacker/reclaimer.

Conveyor (18) is tentatively of the open type. A study would be made of enclosing this conveyor along with the live storage pile (see below).

9. Stacker/Reclaimer - Live Storage

The base scheme includes a live storage pile of up to 2½ days' supply at full load (about 100,000 t) in two sections. This ensures that the powerplant has performance coal and low-sulphur coal directly and promptly reclaimable to assure continuity of power production at all times, including short interruptions in the coal supply from the mine.

Lower-sulphur coal is stored at one end of the live pile in readiness for coal switching for the MCS.

A travelling, rail-mounted stacker/reclaimer stacks coal at up to 2,500 t/h on the live storage piles adjacent to the track.

The live storage piles are reclaimed regularly to avoid spontaneous combustion.

Reclaim from the uncompacted live storage piles is by the bucketwheel on the stacker/reclaimer. Alternatively, a bottom-reclaim system with ploughs may be used. Reclaim capacity is 2,500 t/h.

The live storage pile may be roofed so that, for a reasonable period, the powerplant could directly reclaim dry coal regardless of climatic conditions.

10. Dead Storage

Adjacent to the live storage area, a compacted dead storage pile of approximately 30 days' capacity at full station load could be built. This would allow the powerplant to be self-sufficient for a reasonable period if a major interruption in coal supply from the mine were to occur. The dead storage would be compacted to avoid spontaneous combustion. This storage would be built by mobile equipment taking coal from the live storage area. A minimum of 14 days' supply in dead storage is proposed.

Reclaim would be by mobile equipment to the live storage reclaimer. Emergency reclaim hoppers and conveyors are also included.

The powerplant Coal-handling Plant is designed so that live or dead storage can be rebuilt following heavy usage, while also receiving coal and filling silos at the normal rate of 2,500 t/h. Accordingly, the Supply System (operation 2) will have a maximum capacity of 3,000 t/h.

Part of the dead storage area would be stocked with lower-sulphur coal required for MCS operation.

It is anticipated that, in addition to giving the powerplant operators an assured supply of coal at all times and rapid retrieval of lower-sulphur coal, the live and dead storage facilities may also be used to ease temporary operating problems which may arise from difficult coal quality or other operational factors.

11. Other Powerplant Coal-handling System Features

Many items of detailed engineering related to the coal system will be performed in the final design stage, particularly after the major boiler and coal-handling equipment is ordered.

Particularly important are:

(1) The basic control and instrumentation scheme, including the necessary sampling and testing facilities;

(2) Environmental protection (e.g. dust control, noise control).

8.2.5 Coal System Operation

General

Detailed operating regimes for the components of the project coal system can only be finalized when engineering has advanced into the detailed stage. However, the basic operational concepts are:

(1) Power production for the next period (say one month) will be planned ahead;

(2) The coal requirements will be determined and communicated to both powerplant coal operators and to the mining operation;

(3) Mine production will be scheduled to construct one blending pile, while coal from the other pile is reclaimed and delivered to the powerplant;

(4) When a new blending pile is complete and the other pile is reclaimed, the stacker and reclaimer are interchanged and the process repeated;

(5) In normal operation, coal deliveries will be balanced to powerplant consumption;

(6) Sampling and quality control facilities in operations 1 and 2 will monitor delivered quality. The powerplant will also sample quality of coal delivered to the silos;

(7) The blending piles act as a surge between the mine and the Overland Conveyor, and the powerplant stockpiles provide surge capacity between the Overland Conveyor and the silos. This allows reasonable flexibility to maintain efficient operations in all areas despite temporary imbalance.

Planning has recognized that there may be short periods of emergency when the quantity and quality of the supply of coal to the powerplant does not meet the requirements of the powerplant. Table 8-9 lists some of these possible situations and typical corresponding corrective actions. Strategically placed stockpiles are integrated with the mining and powerplant operations. This provides a means of dealing with emergencies without affecting electricity production.

The overriding concept in operation 3, the powerplant coal operation, is to ensure reliability of power production, with coal of adequate quantity and quality available at all times.

Coal-handling Control System

The instrumentation and control of the powerplant system will be centralized on a separate panel located in the main control room of Generating Units 1 and 2 in the powerhouse. Further consideration will be given to the location of a separate panel in the Main Transfer House for remote control of the storage facilities. A programmable logic controller will be used for the Coal-handling System because of its flexibility and suitability for program changes. This may be integrated with the powerplant process control computer. Program changes will be available to suit various layering techniques required for the mixing of coal in the silos and to change silo-filling programs when handling free-flowing or sticky coals.

The objectives of the Control System are:

(1) To provide an automatic Silo-filling System in which the rate of fill and silo levels can be varied to meet predetermined powerplant coal demand, short-term adjustments, handling ability of the coal, and availability of the coal-handling equipment;

(2) To provide manual selections and indications so that the system can be operated manually;

(3) To provide operational protection of the Coal-handling System.

Silo feed rates are based on unit load, and silo operating levels are adjusted to suit the flow characteristics of the coal and silo-filling requirements. The readout from the belt scale on the third flight of the Overland Conveyor is available in the powerplant control room to assist the operators in setting the loading sequence.

Intermittent operation of the mine Reclaim System and overland conveyors will be avoided.

Coal delivered from the mine is normally directed to the boiler house silos. When handling free-flowing coal after the silos are filled, the powerplant operators may divert the coal to their stockpiles or instruct the mine to stop delivery.

The powerplant Coal-handling System comprises several sub-systems, each connected in independent series, as described in items 1 to 6 below.

The coal-delivery sub-system from the mine, the powerplant, and normal and emergency reclaim systems, discharge coal to the Main Transfer House Surge Bin. Each one is connected in independent series, with plugged chute controls in their discharge chutes located above the surge bin. The bin in the Main Transfer House provides surge capacity for the above systems.

Surge Bins 1 and 2 in the Auxiliary Bay provide surge capacity for the Powerhouse conveyors fed from the Main Transfer House.

An independent system for each row of four silos delivers coal from the Surge Bins (1 and 2) to the silos.

The powerplant stockout sub-system is interlocked with the overland conveyors by a plugged chute detector below the variable splitter in the Main Transfer House.

All sub-systems interlocked in series are provided with timers for sequential starting. Initiation of any stop control on any conveyor in series automatically stops all conveyors upstream of the conveyors on which the stop is made.

1. Overland Conveyors to Surge Bins in Main Transfer House

Receiving Conveyor (1) is part of the Overland Conveyor System, which is interlocked with the high-level chute control in the Main Transfer House. Initiation of any stop of the controls on the overland system or of the variable splitter in the Transfer House switches off all equipment back to the reclaimer at the blending piles.

2. Surge Bin in Main Transfer House

Activation of the high level control automatically speeds up the discharge feeders and/or starts up the Excess Discharge Belt 17. Activation of the plugged chute controls in the chutes feeding the surge bin stops the Overland Conveyor System, the Reclaim System from live storage (Conveyor 18), and the Emergency Reclaim System (Conveyors 19A and 19B). Operation of the low-level control stops the Discharge Feeders.

3. Delivery from Main Transfer House Surge Bin to Surge Bins 1 and 2

Feeders (2A and 2B), the frozen coal crushers, Powerhouse Conveyors (4A and 4B), Transfer Conveyors (5A and 5B), form two independent sub-systems operating in parallel and receiving signals from the high-level control in the Powerhouse Surge Bins (1 and 2).

The variable splitter at the discharge of Powerhouse Conveyors (4A and 4B) automatically adjusts to equalize the loads in the Powerhouse Surge Bins (1 and 2). Indications from the silo load cell determine when the silo for any pair of units is nearly full. When this occurs, the feed from the Transfer House automatically reduces by 50% and, after a delay, the splitter positions the bypass gate to deliver all coal to the surge bin serving the silos not yet filled.

4. Delivery System from Surge Bins 1 and 2 to Silos

The low-level controls of the surge bins are interlocked with their discharge feeders in order to maintain an operational layer of coal on the feeders.

Limit switches confine operating limits and indicate positions for tripper or shuttle conveyors over each silo. Movement of trippers (or shuttles) is under automatic control, with manual override.

All systems, after manual initiation, are automatically controlled.

5. <u>Reversible Stock-out from the Main Transfer House to the Live Stockpile and Reclaim for Normal Reclaim</u>

(1) Stock-out mode:

The Stocking-out Conveyors (16 and 18) operate with the Bucketwheel Conveyor (20); the reversing drives are blocked out and the system interlocked with the Overland Conveyor via the plugged chute control below the splitter in the Main Transfer House.

(2) Reclaim mode:

The bucketwheel, the Stocking-out Conveyors (16 and 18) operating in reverse, and Bucketwheel Conveyor (20) are connected in series and interlocked with the plugged chute and system controls.

6. <u>Emergency Reclaim from Storage</u>

The Dual Conveyor Feeders (14A/B, 14C/D, and 14E/F) below the emergency reclaim hoppers are connected in series with the Dual Emergency Conveyors (15A/B and 19A/B). Both systems are interlocked with the plugged chute control in the Main Transfer House Surge Bin.

<u>General</u>

All silos, distribution bins, and the Main Transfer House Surge Bin are mounted on load cells, and each is equipped with high-level and low-level controls or alarms. Indications of the amount of coal in each bin and silo is shown in the control room.

All feeders from the surge bin and distribution bins have variable-speed drives automatically controlled, but manually adjustable from the control room. Low-level controls switch off the feeders.

The variable splitters in the Main Transfer House and above Powerhouse Surge Bin 1 are motorized with position indicators, and are manually adjustable from the control room.

All flop gates are motorized and may be manually positioned from the control room. In addition, the loads in the Surge Bins (1 and 2) will automatically adjust the variable flop gate splitter into which the Powerhouse Conveyors (4A and 4B) discharge. Limit switches indicate the position of the gate.

All chutes are equipped with plugged chute detectors. All conveyor belts are equipped with:

(1) Belt misalignment switches (two at each head end and two at each tail end);

(2) Emergency pull-cord trip switches on both sides of the conveyor;

(3) Speed switches.

Details of the control of crushers, belt scales, magnetic separators, and metal detectors are not included in this preliminary description.

8.2.6 Low-grade Coal Facilities

The low-grade coal facilities are designed primarily to beneficiate the low-grade coal, i.e. coal between 7.0 and 9.3 MJ/kg. However, coal which is above 9.3 MJ/kg can also be routed through the facility for beneficiating when problems are encountered in making target quality. The facility also allows flexibility in the selective mining process by handling coal which contains excessive amounts of waste materials. Also, should the secondary screening and crushing plant be required to handle low-sulphur coal, or should it be out of commission, the low-grade coal facility can be modified to handle normal-grade coal at a reduced rate.

The estimated quantity of low-grade coal to be handled over the life of the mine is 21.7×10^6 t. Details of the low-grade coal beneficiation study appear in Simon-Carves' report, dated August 1979. Further testing on a pilot-plant scale is required to confirm the feasibility and design parameters for low-grade coal beneficiation before final design. Figures 8-2 and 8-4 show the layout of the facility, and Tables 8-2 and 8-4 give details of the conveyors.

The low-grade coal is delivered to the low-grade coal truck dump pocket (M4). The truck dump pocket, also capable of accepting waste material or coal, is fitted with a grizzly having 600 mm square openings.

Coal reclaimed from the pocket by a reciprocating push feeder discharges to a cascading vibrating grizzly with 200 mm square openings. The grizzly overflow discharges to an impact-type crusher for size reduction to -200 mm. Grizzly underflow, together with the crusher product, gravitates to a 1,400 mm-wide transfer conveyor for transport to the Low-grade Coal Conveyor, also 1,400 mm wide, which feeds it to the Low-grade Coal Conveyor (M8D), terminating at the drive and Transfer House. A transfer conveyor (LG1) then delivers it to the low-grade coal bins.

The Low-grade Coal System is designed for 1,000 t/h. However, the Conveying System is designed to handle up to 5,000 t/h to allow greater flexibility by providing a back-up system for both coal and waste systems. If coal is being handled, a bypass conveyor (C1) in the Drive and Transfer House allows the coal to be diverted to the Transfer Conveyor (C2) feeding the coal bins. Waste-handling on this system is described in Section 8.3.

Low-grade coal is discharged to one of the two low-grade coal bins by means of a reciprocating chute. The reclaiming of low-grade coal at 500 t/h from each bin is by apron feeder (LG2A, 2B). Each apron feeder feeds to a low-grade coal screen feed conveyor (LG3A, 3B) delivering to the Screening and Crushing House for low-grade coal. Each conveyor then discharges its product to an inclined three-deck vibrating screen (LG4A, 4B). The top deck is fitted with a 50 mm square opening woven wire deck, while the middle deck is fitted with a rod deck having 13 mm spacings. The bottom deck is blanked off with mild steel plate and acts as a carrying deck. The screen can be upgraded to a capacity of 1,000 t/h by blanking off the middle deck when the system has to handle regular grades of coal as described in Section 8.2.4.1.

The +50 mm oversized material carried on the top deck is discharged to an impact-type crusher (LG5A, 5B) for reduction to -50 mm. Material sized 50 x 13 mm passes via a chute to join the crusher product. A portion of this product is directed to a bulk density meter (LG6A, 6B) for ash monitoring. The ash value determines to which conveyor the +13 mm low-grade coal is discharged. Should a low ash-reading indicate the +13 mm fraction as acceptable for inclusion in the blended product for the powerplant, the fraction gravitates to the Coal Conveyor. Conversely, a high ash-reading causes the flop gate in the two-way chute to automatically divert the +13 mm coal to the Reject Conveyor.

The -13 mm low-grade coal carried on the lower deck is similarly sampled on a bulk density meter (LG6C, 6D) to determine ash. A two-way chute and flop gate diverts this product either to the Reject Conveyor (LG1) or to the Product Conveyor (C7B), depending on the measured ash.

The Reject Conveyor terminates at a transfer house, where the product is discharged to a second reject conveyor (LG8). This conveyor delivers it to a transfer house, where the product is fed to a two-way chute for routing either to the Houth Meadows Waste Dump, to Medicine Creek in later years, or for other uses.

8.2.7 Low-sulphur Coal

The Coal-handling System will be required to convey low-sulphur coal from the mine direct to the powerplant, bypassing the blending piles.

The low-sulphur coal will be required to meet MCS conditions, or to replenish the low-sulphur portion of the stockpiles at the powerplant, whenever these become depleted.

Low-sulphur coal at a peak rate of 3,000 t/h will be routed from the mine face to the mine Coal Conveyor (M8A), which delivers the coal to the Screening and Crushing Plant. The low-sulphur coal will be crushed to -50 mm, and routed through the Sampling and Transfer House via Product Conveyors C7A and C8A to Product Conveyor C9A. Product Conveyors C7B, 8B, and 9B provide back-up to this delivery line and would also handle normal-grade coal production, if required, from the low-grade coal facility to the blending piles.

The low-sulphur coal transfers to the Reclaim/Bypass Conveyor C12, bypassing the blending piles, and then via the Overland Conveyor System to the 600 t surge bin. The powerplant Coal-handling System will then route the low-sulphur coal to the silos or to the storage areas.

8.2.8 Coal Sampling

Throughout the coal-mining and handling operations described in the preceding sections, coal-sampling is used to monitor the quality of the coal. This is necessary to ensure the supply of coal of consistent quality to the powerplant, efficient use of the resource, and to allow efficient control in all areas of the project. Sulphur content, as well as HHV/ash, is analysed to assist in maintaining a mean sulphur level in powerplant fuel which provides SO_2 emission levels within the predicted range. The coal-sampling techniques employed in the mine area are described in Section 7.5, "Fuel Quality Control". Sampling of coal in the materials-handling system takes place in each of the coal-handling operations described in Section 8.2.3. The locations, as shown on Figure 8-1, are: (1) before blending; (2) after reclaiming in the mine operations; and (3) before silo-filling in the powerplant.

Standard and special analyses of the samples from the various stages will be carried out in on-site laboratories. The results will be used to monitor the operation. The measurement of sulphur in the coal will be also carried out by standard methods. However, a sulphur monitor with rapid readout is being developed and is expected to be installed for testing in the near future. This type of device would be included in the Sampling System in the detailed engineering phase, should it prove effective.

The installations are described below.

8.2.8.1 Belt-Sampling - Mine

Automatic samplers, one on each of two conveyor belts (C8A, 8B), are installed in the Sample and Transfer House located between the Screening and Crushing Plant and the Blending/Storage Yard. These samplers monitor the quality of coal going into the blending/ storage piles. A second installation located in the Sampling and Transfer House on the Overland Conveyor monitors the quality of coal being delivered to the powerplant.

Each of these installations is interlocked with weigh scales, which allows the samples to be taken at predetermined intervals. This also enables the weighted average quality of coal in the blending piles, or shipped to the powerplant, to be determined.

The information provides feedback to mine operations to check predictions and to adjust the mining schedule if required.

The variation in values obtained from these two installations checks the efficiency of the blending operations. In addition, provision is made for the installation of continuous ash monitors, which give a rapid check of the ash content of the coal. Because of the linear relationship between ash and heating value, a quick check on the heating value of the coal is therefore possible. This readout can then be integrated to show the aggregate value of the coal in the stockpile or of the coal shipped to the powerplant in a given period. The heating value of the coal being delivered to the powerplant is automatically relayed to the powerplant.

8.2.8.2 Belt-Sampling - Powerplant

Sampling installations similar to those described in 8.2.8.1 are employed. At the powerplant, coal-sampling will be carried out for plant operation, for plant performance assessment, and for monitoring in relation to stack emission data.

8.3 WASTE-HANDLING

8.3.1 Waste-handling System Requirements

This section describes the material-handling equipment and methods required to transport waste materials from the pit to their respective disposal areas and to construct the waste dumps to meet the requirements of the mining plan and production schedule presented in Section 5. The schedule shows that the total volume of waste to be handled over the life of the mine is 426.8 x 10 bank m^3. It is planned to dispose of 418 x 10^6 bank m^3 in the waste dumps; the remainder will be used for road construction, etc. The peak year for waste production will be Year 11, when 18.25 x 10^6 bank m^3 will be handled.

8.3.2 Design Criteria

The basic design criteria described in Section 8.2.2, "Coal-handling - Design Criteria", will apply to the waste-handling and conveying systems. The bulk density for conveyor and equipment capacity calculations is 1,600 kg/m^3 for the waste materials. For other waste material parameters refer to Section 5.2.5. For ash-handling, the bulk density of loose ash is about 800 kg/m^3 and of compacted ash about 1,280 kg/m^3.

8.3.2.1 Design Features

The design features noted in Section 8.2.2, "Coal-handling - Design Criteria", are also incorporated in the design of the Waste-handling System. The clay-handling system is carefully designed to account for the volumes of wet and sticky materials. Features of this system minimize the handling of the material by eliminating surge hoppers and storage bins, reducing the number of transfer points, employing vertical drops at transfer points where possible, and avoiding two-way chutes, etc.

8.3.3 Waste-handling System - Description

The project Waste-handling System can be divided into two separate areas, as shown on Figure 8-1. They are:

1. Mine Waste Disposal

Mine waste disposal consists of an in-pit handling system and a dump construction system to handle the mine waste materials. The latter will consist of two identical systems initially installed in the Houth Meadows Dump, with one of them being relocated to the Medicine Creek Dump in Year 15.

2. Ash Disposal

This consists of two identical systems to handle the fly-ash and bottom-ash materials from the powerplant for disposal in Medicine Creek.

Ultimately, Medicine Creek will receive both mine waste and ash. Careful scheduling of the dumps' construction will ensure that each system can operate effectively without affecting the other.

These separate and independent operations are described below.

8.3.4 Waste-handling - In-Pit

The In-pit Waste-handling System shown on Figure 8-3 consists of the following:

1. Truck Dump Stations

As described in Section 8.2.4.1, "Coal-handling - Mining", a total of three dump stations are installed. Waste materials, loaded on 154 t rear dump haulage trucks by the 14.5 m^3 hydraulic shovels, are delivered at a peak rate of 5,000 t/h to the designated dump pockets at the dump stations. In-seam waste materials are handled by the 77-t trucks and 10.7 m^3 shovels, as required during mining operations. Each dump pocket is designed to handle up to 2,500 t/h. The number of dump pockets installed at each of the dump stations depends upon the quantities of the different materials to be delivered to each dump station. Each pocket holds three truckloads of material and has a 600 mm square grizzly installed to screen off oversize material.

As shown on Figure 8-2, there are four pockets installed at the first dump station, two for construction materials (M3 and M4) and two for general waste/clay materials (M5 and M6). The second station has two pockets for construction waste (M11 and M12), one for general waste, and the third has one for construction materials (M13). The pocket at the third dump station will be relocated from the first station. Each of these pockets is identical in design, to allow for the handling of all materials.

Separate dump pockets (M7 and M14), as shown on Figure 8-3, with a capacity of one truckload of material, are installed at the first and second dump stations, to handle wet clay. This pocket is located directly over the general waste/clay conveyor, allowing the transfer of material to the conveyor through an apron feeder, and eliminating handling through a crusher. A grizzly will screen off oversize materials, which will be disposed of by front-end loader and truck.

Further testing on this wet clay is required before final design of the Clay-handling System.

2. Primary Crushing

Waste materials are reclaimed from each dump pocket by a hydraulic reciprocating feeder at a peak rate of 2,500 t/h. Two feeders are required at any time to handle the peak tonnage of a given waste material, i.e. construction grade or general waste/clay. The feeder delivers the waste to a vibrating screen, which removes the -200 mm material and feeds it to a 1,400 mm-wide transfer conveyor below. The oversize material is crushed to -200 mm by an impact crusher, then passes to the Transfer Conveyor for delivery to the appropriate mine waste conveyor.

A preliminary selection of an impact crusher has been made. This crusher has the ability to handle the run-of-mine materials in Hat Creek, although further tests are required, especially for the clay materials. The crusher can be fitted with heated impact surfaces which would release wet clay. This feature can be easily retrofitted if necessary. Other types of crushers studied are discussed in the Simon-Carves' report 1979.

Although the crushers are designated for specific materials, their ability to handle other materials allows added flexibility in the system - for example, coal and low-grade coal routed through the impact crusher and delivered to the coal belt by use of a two-way chute on the Transfer Conveyor. Figure 8-3 shows this arrangement.

3. Waste Conveyors

The In-pit Waste Conveying System is designed to handle the two types of waste. Two conveyors, each with 1,400 mm-wide steel cord belt, will be installed. The first, designated "Waste Conveyor" (M8B), handles only construction-grade materials; and the other, designated "Waste/Clay" (M8C), handles general mine waste/clay and is equipped to handle wet clays. Table 8-5 gives details of the conveyors.

The Waste Conveyor, ultimately three flights long (M8B, M9B, and M10B) to follow the mine development, receives the construction-grade material from the transfer conveyors and transports it at up to 5,000 t/h to the Drive and Transfer House. Here the material is routed to a pair of waste bins, adjacent to the low-grade coal bins, by a 1,400 mm-wide transfer conveyor (W1). A two-way chute ensures equal distribution of material into the bins. A pair of apron feeders (W2A and 2B) discharge the waste from the bins to a 1,400 mm-wide transfer conveyor (W3), which feeds it to one of two overland conveyors (H01 and H02). A two-way chute determines which conveyor carries the material to Houth Meadows, or, in later years, to Medicine Creek. The waste bins incorporate truck-loading facilities for emergency use and surge if the dump waste-conveying systems are inoperative, and also provide a supply of construction-grade materials for road building or other uses.

The Waste/Clay Conveyor is only two flights long (M8C and M9C). A study of material distribution indicates that a third flight is not required. The general waste materials received from the Transfer Conveyor, or, in the case of wet clay from the apron feeder at dump pocket M7 or M13, are delivered to the surface, bypassing the Drive and Transfer House, and are delivered to either one of the overland waste conveyors to Houth Meadows. A moving-head pulley on the Waste/Clay Conveyor allows selection of the appropriate conveyor to Houth Meadows.

Mine conveyors (M8D and M9D), as shown on Figure 8-3, provide some flexibility and back-up in emergencies. These conveyors, primarily handling low-grade coal, as described in Section 8.2.6, also handle construction-grade material as well as coal when required.

8.3.5 Waste-handling - Houth Meadows

The Waste-handling System must be able to dispose of both types of waste materials to suit the method and sequence of construction of the dumps.

The retaining embankment must be constructed using only sand and gravel. The section of the dump upstream from the embankment will be used to dispose of the general waste/clay materials.

The dumps will be constructed in 35-m lifts with a system which consists of conveyors and spreaders. Two systems, each building a 35-m lift, will be installed initially at Houth Meadows, with one system being relocated to Medicine Creek in Year 15.

Each system will be installed at the upstream end of the dump and will progress downstream to the retaining embankment. After each lift is completed, the system will be dismantled and reassembled at the upstream end of the dump to begin another lift. A pictorial layout of the Houth Meadows Waste-handling System is shown on Figure 8-7; the components are described below. Section 5.5 describes the method of construction and the development sequence of the dump and its retaining embankment.

1. Conveyors

The initial development of Houth Meadows Waste Dump will be carried out by two independent systems. Each of the two systems consists of three types of conveyor, i.e. permanent overland, transfer, and shiftable. Table 8-5 lists the conveyors. Because of the capacity and high belt tensions, the conveyors are equipped with a steel-cord belt.

Although similar in design and construction, the 1,400 mm-wide conveyors have special features necessary for their particular function. For example, the Shiftable Conveyor is complete with shifting rails to facilitate moving the conveyor line on the dump. Drive stations on the transfer and shiftable conveyors are mounted on pontoons for easy moving.

Material from the mine, i.e. construction waste or waste/clay, is delivered to one of the two overland permanent conveyors (H01 or H02), either from the waste bins or direct from the mine in the case of clay materials. The material is then fed to the Transfer Conveyor (HT1 and HT2), and then to the Shiftable Conveyor (HS1 and HS2) via a short portable conveyor. A travelling-belt tripper transfers the material to the spreader.

The position of the system on the dump determines which material is required, i.e. construction waste for embankment construction or general waste for other areas.

2. Spreaders

Each of the two systems incorporates a crawler-mounted spreader to place the waste materials.

The specifications are as follows:

Length of loading boom	- 40 m
Length of discharge boom	- 40 m
Belt width	- 1,400 mm
Belt speed	- 4.5 m/s
Discharge height	- 18 m
Capacity	- 5,000 t/h.

As shown in Figure 8-7, the spreader, receiving material via the belt tripper, dumps the waste first in a 20-m lift below and ahead of the Shiftable Conveyor, and then in a 15-m lift above and behind the conveyor. The spreader and Shiftable Conveyor, after completing a cycle, are moved 50 m down the dump towards the retaining embankment to begin another pass. A dozer provides the necessary back-up to the spreader for levelling and clean-up. One of the spreaders will also be relocated to Medicine Creek in Year 15. A front-end loader and 32-t truck fleet are used to deliver waste to areas beyond the reach of the spreader.

8.3.6 Waste-handling - Medicine Creek

It is planned to begin using Medicine Creek for waste disposal in Year 15. One of the two conveyor and spreader systems will be relocated from Houth Meadows; additional overland conveyors will be required. The route selected for the Overland Conveyors (M01 to M05) from the mine mouth will be parallel to the Overland Coal Conveyor C15, and will continue to a transfer station at the Northern edge of Medicine Creek. The system of Transfer Conveyors (MT1 and MT2) and Shiftable Conveyor (MS1) and spreader extends from this Transfer House into the dump area.

The method of operation of the system and dump development sequence will be the same as for Houth Meadows. Table 8-6 gives details of the conveyors.

The Ash-handling System, as described in Section 8.3.6, will be in operation in mid-Medicine Creek at all times. Delivery of waste materials to Medicine Creek will be scheduled to ensure that the disposal of ash will not be affected. Figure 5-20 shows the dump with the waste and ash interface.

8.3.7 Ash-handling - Medicine Creek

Close attention has been paid to the design of the Hat Creek Ash-handling and Disposal System due to the large quantities involved, because reliability of the entire system is of the utmost importance for continuous power production. The system caters for variations in ash production under all conditions of the specified operating regime, such as the unusual amounts of bottom-ash which could form at times.

For ash systems, environmental impacts are particularly prominent, and these have been addressed, together with mitigation measures. The adoption of a "dry" disposal scheme reduces the quantity of water required from the Thompson River and uses the storage area more effectively than "wet" ash disposal.

Safe working conditions are vital and will be prescribed during the construction, operation, and maintenance of the system. Economics of operation, including manpower requirements, have been studied to minimize costs.

Provision for loading fly-ash or bottom-ash for sale has not been included, but can be incorporated if and when needed. Provision for possible recovery of fly-ash or bottom-ash from the disposal site is not included.

8.3.7.1 Bottom-ash

The Bottom-ash Removal System is shown on Figures 8-10 and 8-11.

A continuous removal system using a submerged drag-bar conveyor (98) moves the ash from beneath each boiler and discharges it to a cross-belt conveyor (99) and thence to one of two collecting belt conveyors (100A and 100B) which service all four boilers.

The final position and arrangement of this equipment will be established when the boiler is designed.

The Drag-bar Conveyor (98) is driven through a motor-gearbox combination to a round-link-type chain and sprocket assembly at a fixed or variable speed related to boiler load. The design incorporates within the boiler hoppers quenching water sprays, which cool and break up the ash. The hoppers are fitted with shut-off gates.

The ash is further cooled in the Drag-bar Conveyor Trough to an acceptable temperature for handling by the belt conveyors. The water temperature in the trough is controlled by a heat exchanger and recirculating-pump cooling system. A surge tank is incorporated to absorb the excess water during removal cycles of rejects from the coal pulverizers, which are sluiced intermittently to the Drag-bar Conveyor Trough.

The bottom-ash and pulverizer rejects move up the inclined section of the conveyor, allowing the quenching water to drain off and eliminating the need for dewatering bins.

Provision for an ash crusher, between the Drag-bar Conveyor (98) and the Unit Cross-belt Conveyor (99), would be included, should it be established that this would reduce compacting effort at the disposal site.

During normal operation only one of two collecting belt conveyors (100A and 100B) will operate; start up and transfer to the standby conveyor is automatic.

Capacities are based on bottom-ash at 40% water content. Unit cross-belt capacity includes 20% surge capacity and collecting belt conveyors include 10% surge capacity.

8.3.7.2 Fly-ash

The Fly-ash Removal System is shown on Figure 8-12.

Reliability at high elevations necessitates a pressure system for fly-ash removal.

Fly-ash is released from collecting hoppers by air lock valves and is pneumatically conveyed in pipes. Fly-ash discharges to one of two storage silos whose volume depends upon the selection of either an intermittent or a continuous removal system.

Each silo is equipped with two conditioner/unloaders to discharge fly-ash in a dampened state to the transport conveyors (101A and 101B).

8.3.7.3 Economiser and Airheater Ash

Ash collected from the economiser hoppers and air pre-heater hoppers is transported by the Fly-ash Pressure System to the two storage silos.

As a possibility exists that large pieces of ash may form in the economiser from agglomeration, there will be provision to fit the hoppers with grizzlies to prevent blockage.

8.3.7.4 Transportation of Ash to Disposal Area

Two single-flight belt conveyors (101A and 101B) transport both bottom-ash and fly-ash from the powerplant to the North side of the disposal area in mid-Medicine Creek Valley. This system is shown on Figures 8-10, 8-12, and 8-13.

To minimise dust problems, dewatered bottom-ash is deposited over the fly-ash on the conveyors, although it is necessary periodically to load bottom-ash and fly-ash on separate conveyors when building drainage courses within the ash disposal pile.

As they are downhill conveyors, the loaded ash transportation conveyors will feed power back to the plant. A reliable braking system is provided.

Typically, the Ash Transportation System will handle about 10,000 t/d of ash from four units operating at full load when burning performance coal.

One transportation conveyor runs continuously, carrying bottom-ash, and the other conveyor is on standby. For five hours in each shift, fly-ash is discharged from the storage silos to the Transportation Conveyor upstream of the bottom-ash loading point, allowing bottom-ash to cover the fly-ash. During the time that only bottom-ash is sent to the disposal area, this is spread and compacted in the drainage layer for the succeeding mixture of fly-ash and bottom-ash.

The control station for the ash-handling systems is located in the South end of the Powerhouse at ground level, with local/remote controls for cross-belt conveyors at each unit. Bicolour signal lights operated in conjunction with the discharge gates from the fly-ash silos indicate to the operators at Medicine Creek what material is being loaded to the Transportation Conveyor, which is fitted with emergency stop controls at the discharge end.

The mid-Medicine Creek Valley disposal site will be prepared by removal of all vegetation and topsoil.

Two Shiftable Conveyors (102A and 102B) are used to deliver the ash from the transportation conveyors to the required location at the disposal site. This will initially be at the base of the reservoir dam at the East end of mid-Medicine Creek Valley, moving Westwards.

Two mobile conveyors (103A and 103B), two shiftable stackers (104A and 104B), and two rubber-tired dozers distribute the ash, which is deposited and compacted in layers of approximately 300 mm thickness. One shiftable conveyor, one mobile conveyor, and one shiftable stacker will be in service while the second of these pieces of equipment will either be on standby or will be moving to a new location on the disposal site.

Most of the ash is placed and compacted as a mixture of fly-ash and bottom-ash, but drainage courses of bottom-ash are laid at specified elevations to promote proper drainage within the pile. It is possible that during Winter months, less compaction will be achieved than during the warmer season.

8.3.7.5 Ash Disposal, Pile Reclamation, Drainage, and Stability

Reclamation of the Ash Disposal Pile will be a continuous process. Figure 8-14 shows an early stage in the development of the ash disposal area and its reclamation.

As soon as the final elevation has been reached in each section, approximately 600 mm of topsoil will be spread and seeded to prevent erosion. This will occur following Year 3, Year 6, and Year 15 of powerplant operation, and is environmentally advantageous, as reclamation of disturbed land areas reduces erosion, seepage, and fugitive dust emissions.

Removal of all vegetation and topsoil from the ash disposal area will leave a stripped surface of glacial till or other similarly impermeable surface.

Lined drainage courses are provided at the bottom and sides of the disposal area in addition to those within the pile, to prevent accumulation of water and consequential pile instability.

The finished surface of the pile is sloped a minimum of 1% to the West and South. During the initial 15 years of powerplant operation, precipitation and seepage from the make-up water reservoir will be collected behind a berm located just downstream of the ash-pile toe. This wastewater will then be pumped to a runoff holding basin sited North of the waste pile.

Rainfall runoff from the powerplant site and the associated coal storage area is collected in drainage ditches and feeds by gravity to the holding basin, where it is available for ash-dust suppression.

The lower slope of the ash disposal area is sloped 5% as shown on Figure 8-13. Ash and mine waste volumes produced may also be higher than anticipated, but the capacity of the disposal area can be increased by raising the mine waste embankment and filling the area up to the minimum slope of 1% if required, depending upon volumes required by less densely compacted ash during freezing conditions.

8.4 ELECTRICAL POWER SUPPLY

8.4.1 Mine - Coal, Waste, and Ash-handling

The mine coal and waste conveyors, Coal Screening and Crushing Scheme, Coal-blending and Reclaim System, overland conveyor intermediate drive motors, and Ash-handling System, will all be tapped off the 60 kV overhead ring main system supplied from the switchyard at the powerplant. One overhead 60 kV line runs from the switchyard down a common corridor with the Overland Conveyor to the Coal Blending Area, where it turns South down the waste conveyor route to Medicine Creek. A second 60 kV line runs down the ash conveyor route, turning West along the North edge of Medicine Creek, to link up with the first 60 kV line to complete the ring. In this way, all areas of the Materials-handling System have two independent and physically segregated alternative supplies.

8.4.2 Powerplant - Coal-handling

The powerplant Coal-handling System and the drive motors at the delivery end of the Third Overland Conveyor are supplied from the generating station 6.9 kV station auxiliary boards.

8.5 RECOMMENDATIONS FOR FUTURE TESTWORK

8.5.1 Crushing and General Characteristics of Run-of-Mine Materials and Blended Coal

Bulk samples representative of the various run-of-mine materials must be obtained for testing. However, some material samples will not be available until after mining commences, as they will derive from lower levels of the pit.

The following tests are recommended:

(1) A run-of-mine size analysis, and a size analysis for each material after crushing and handling operations, using different crushers;

(2) Tests to determine the breaking characteristics of the better coals; specifically, to obtain answers to such questions as: If the better coals are harder than the waste materials, is beneficiation by selective crushing and screening feasible? Would a Bradford Breaker reject good coal along with petrified wood and clay?

(3) Tests to identify problems connected with petrified wood to obtain answers to questions such as: Could impactor crushers allow scalping off this material after being subjected to primary crushing? Is the material intrinsically so hard that damage may result by using simpler types of crushers like the "Wing" crusher? Could a Bradford Breaker reject this material from say 200 x 50 mm raw coal at the secondary crushing stage?

(4) Tests to indicate practical methods for dealing with claystone waste, specifically in connection with moisture content, and crushing and handling characteristics when mined in conditions anticipated;

(5) Tests to determine the basic material parameters to aid in the design and selection of handling equipment and silos, such parameters to include bulk density factors, angle of repose and surcharge, flowability, and shearing.

8.5.2 Borecore Test Program

Since bulk samples can only be obtained from many areas after mining has advanced, it will be necessary to obtain data from suitable large-diameter (200 mm) drill cores. In many cases they should help to answer the above questions, subject only to final design stage confirmation.

The program must first establish the applicability and technique of the method by comparison with data from adjacent bulk trenches.

It is not anticipated that a large number of these drill cores will be required. (Their situation can be determined from existing small-diameter core results, to ensure that the complete range of materials is sampled.) Due to the thickness of the measures, each core would produce a significant sample weight.

(1) Dry tumbling tests should be performed to establish the raw coal size consist of coal zones which have not been sampled.

(2) Samples of all materials should be obtained for practical classification by crushing and handling equipment manufacturers.

8.5.3 Crushing Tests

There are no standard test procedures, since each type of crusher makes use of different characteristics. Specific requirements should be determined by consultation with each crusher manufacturer. The following types of crusher will be considered:

Bradford Breakers;
Siebra Screen/Crusher;
Impactors;
Roll Crusher;
Hammermills;
Clay Shredders.

Specific attention should be paid to the characteristics of the 200 mm x 50 mm fraction after primary breaking at 200 mm.

8.5.4 Handling Characteristics

(1) A series of 500 mm x 0 coal qualities should be tested at various surface-moisture contents between 3% and 10%. This should enable the plant designers to project chute angles for the coarser fractions;

(2) A series of 13 mm x 0 coal qualities should be similarly tested;

(3) Clay samples must be submitted to equipment manufacturers.

8.5.5 Screening Tests

Specific requirements for a detailed test program for the screening of the various materials will be determined by consultation with screen manufacturers.

The following types of screens will be considered: roller (self-cleaning type), vibrating (woven wire, rod), probability, disc., etc.

Special attention will be paid to the handling of wet fines and sticky materials.

TABLE 8-1

IN-PIT CRUSHING PLANT - TRANSFER CONVEYORS

Conveyor	Length m	Lift m	Capacity t/h	Speed	Installed hp	Year Installed
Dump Station No. 1						
Coal Transfer M1	47	6	1,600	2.5	175	-1
Coal Transfer M2	62	7	1,600	2.5	200	-1
Waste Transfer M3	47	6	2,500	2.5	300	1
Waste Transfer M4[1]	47	6	2,500	2.5	300	1
Waste/Clay Transfer M5	47	6	2,500	2.5	300	1
Waste/Clay Transfer M6	47	6	2,500	2.5	300	1
Dump Station No. 2						
Coal Transfer M1[2]	47	6	1,600	2.5	175	8
Coal Transfer M9	62	7	1,600	2.5	200	8
Waste Transfer M11[1]	47	7	2,500	2.5	300	8
Waste Transfer M12	47	6	2,500	2.5	300	8
Waste/Clay Transfer M13	47	6	2,500	2.5	300	8
Dump Station No. 3						
Coal Transfer M1[2]	47	6	1,600	2.5	175	20
Coal Transfer M9[2]	62	7	1,600	2.5	200	20
Waste Transfer M4[1,2]	47	6	2,500	2.5	300	20

[1] Handles low-grade coal
[2] Relocated from Dump Stations No. 1 and No. 2

Sources: Simon-Carves of Canada Ltd.
B.C. Hydro Thermal Division

TABLE 8-2

IN-PIT INCLINE - COAL AND WASTE CONVEYORS

Conveyor	Length m	Lift m	Capacity t/h	Installed Power hp	Year Installed
Dump Station No. 1					
Coal M8A	500	45	3,200	1,000	-1
Waste M8B	500	45	5,000	1,400	1
Waste M8C	500	45	5,000	1,400	1
Low-grade Coal M8D	500	45	5,000	1,400	1
Dump Station No. 2					
Coal M9A	400	75	3,200	1,400	8
Waste M9B	400	75	5,000	2,000	8
Waste M9C	400	75	5,000	2,000	8
Low-grade Coal M9D	400	75	5,000	2,000	8
Dump Station No. 3					
Coal M10A	600	90	5,000	1,600	20
Waste M10B	600	90	5,000	2,400	20

Note: All conveyors 1,400 mm wide and 4.5 m/s

Source: B.C. Hydro Thermal Division

TABLE 8-3

CRUSHING, STACKING RECLAIMING, AND DELIVERY CONVEYORS

Conveyor	Length m	Lift m	Capacity t/h	Speed m/s	Installed hp
Crushing					
Bypass C1[1]	26	4	3,200	4.5	150
Transfer C2	197	24	3,200	4.5	700
Screen Feed C4A	121	25	1,000	2.5	200
Screen Feed C4B	116	25	1,000	2.5	200
Screen Feed C4C	116	25	1,000	2.5	200
Screen Feed C4D	121	25	1,000	2.5	200
Product C7A	86	7	3,200	4.5	250
Product C7B	112	7	3,200	4.5	250
Product C8A	127	11	3,200	4.5	350
Product C8B	127	11	3,200	4.5	350
Product C9A	187	34	3,200	4.5	700
Product C9B	183	34	3,200	4.5	700
Stacking and Reclaiming					
Transfer C10	135	6	3,200	4.5	200
Stacking C11	670	10	3,200	4.2	600
Reclaim Bypass C12	670	10	3,000	4.2	600
Reclaim C13	670	10	3,000	4.2	600
Collecting C14	135	10	3,000	4.2	250
Delivery					
Overland C15	1,000	80	3,000	4.2	1,500
Overland C16	1,100	245	3,000	4.2	3,200
Overland C17	1,850	165	3,000	4.2	3,000

Note: All conveyors are 1,400 mm wide, except as noted
[1] This conveyor is 1,800 mm wide, 2.6 m/s
Source: Simon-Carves of Canada Ltd.
B.C. Hydro Thermal Division

TABLE 8-4

LOW-GRADE COAL - PLANT CONVEYORS

Conveyor	Length m	Lift m	Capacity t/h	Speed m/s	Installed hp
Transfer LG1	94	23	5,000	4.5	700
Screen Feed LG3A	115	25	1,000	2.5	200
Screen Feed LG3B	115	25	1,000	2.5	200
Rejects LG7	76	6	1,000	2.5	100
Rejects LG8	188	6.5	1,000	2.5	125

Note: All conveyors are 1,400 mm wide

Source: Simon-Carves of Canada Ltd.

TABLE 8-5

HOUTH MEADOWS DUMP - WASTE CONVEYORS

Conveyor	Length at Installation m	Lift m	Installed hp	Year Installed
Transfer W1	93	23	700	1
Transfer W3	75	6	300	1
Line No. 1				
EL900				
Overland H01	600	5	600	1
Transfer HT1	1,150	-	1,000	1
Shiftable HS1	900	-	1,200	1
EL970				
Overland H03	700	70	2,000	6
Transfer HT1A	1,250	-	1,000	6
Shiftable HS1A	1,700	-	1,500	6
Line No. 2				
EL935				
Overland H02	900	35	1,500	2
Transfer HT2	1,250	-	1,000	2
Shiftable HS2	1,300	-	1,500	2
EL1005				
Overland H04	700	70	2,000	9
Transfer HT2A	1,250	-	1,000	9
Transfer HT4	900	-	750	9
Shiftable HS2A	1,500	-	1,200	9
EL1040				
Overland H06	900	70	2,400	23
Transfer HT2B	850	-	750	23
Transfer HT6	850	-	1,000	23
Shiftable HS2B	1,600	-	2,000	23

Note: All conveyors on this table are 1,400 mm wide, 5,000 t/h @ 4.5 m/s

Source: B.C. Hydro Thermal Division

TABLE 8-6

MEDICINE CREEK DUMP - WASTE CONVEYORS

Conveyor	Length at Installation	Lift m	Installed hp	Year Installed
Overland M01	100	6	500	15
Overland M02	100	6	500	15
Overland M03	350	30	1,000	15
Overland M04	1,700	120	4,000	15
Overland M05	1,400	140	4,000	15
EL1060				
Transfer MT1	1,600	-145	Regenerative	15
Transfer MT2	1,000	-	1,000	15
Shiftable MS1	500	-	500	15
EL1095				
Transfer MT1A	1,200	-110	Regenerative	18
Transfer MT2A	1,400	-	1,200	18
Shiftable MS1A	500	-	500	18
EL1130				
Transfer MT1B	800	-95	Regenerative	26
Transfer MT2B	1,500	-	1,200	26
Shiftable MS1B	500	-	500	26

Note: All conveyors are 1,400 mm wide, 5,000 t/h @ 4.5 m/s

Source: B.C. Hydro Thermal Division

TABLE 8-7

POWERPLANT COAL-HANDLING CONVEYORS

Conveyor	Length m	Capacity t/h
Receiving (from Overland Conveyor) 1A and 1B	190	3,000
Belt Feeders (Main Transfer House) 2A and 2B	11	2,500
Powerhouse 4A and 4B	356	2,500
Transfer 5A and 5B	154	2,500
Intermediate 6A, 6B, 7A, and 7B	51	400
Belt Feeders (Powerhouse) 8A, 8B, 8C, 8D, 9A, 9B, 9C, and 9D	11	400
Silo 10A, 10B, 11A, 11B, 12A, 12B, 13A, and 13B	51	400
Belt Feeders (Reclaim Hoppers) 14A, 14B, 14C, 14D, 14E, and 14F	6	1,300
Emergency Reclaimer 15A and 15B	180	2,500
Stacker/Reclaimer 16	43	2,500
Excess Discharge Pile 17	15	1,200
Reclaim and Stockout 18	460	2,500
Emergency Reclaim 19A and 19B	160	2,500

Note: (1) Belt speeds shall not exceed 3.3 m/s on conveyors except the Silo Conveyors (10, 11, 12, and 13) where the belt speed shall not exceed 2.2 m/s.

(2) The total installed capacities of the motors for the powerplant Coal-handling System is approximately 3,280 kW.

TABLE 8-8

ASH-HANDLING CONVEYORS

Conveyor	Length m	Lift m	Capacity t/h	Speed m/s	Installed hp
Drag-bar Conveyor 98	21	-	90	-	-
Cross-belt Conveyor 99	48	-	127	1.0	8
Collecting-belt Conveyors 100A and 100B	310	-	453	1.5	50
Transport Conveyors 101A and 101B	2,950	-155	851	2.2	200
Shiftable Conveyors 102A and 102B	1,200	varies	851	2.2	300
Mobile Conveyors 103A and 103B	30	-	851	2.2	20
Shiftable Stackers 104A and 104B	36	-	851	2.2	50

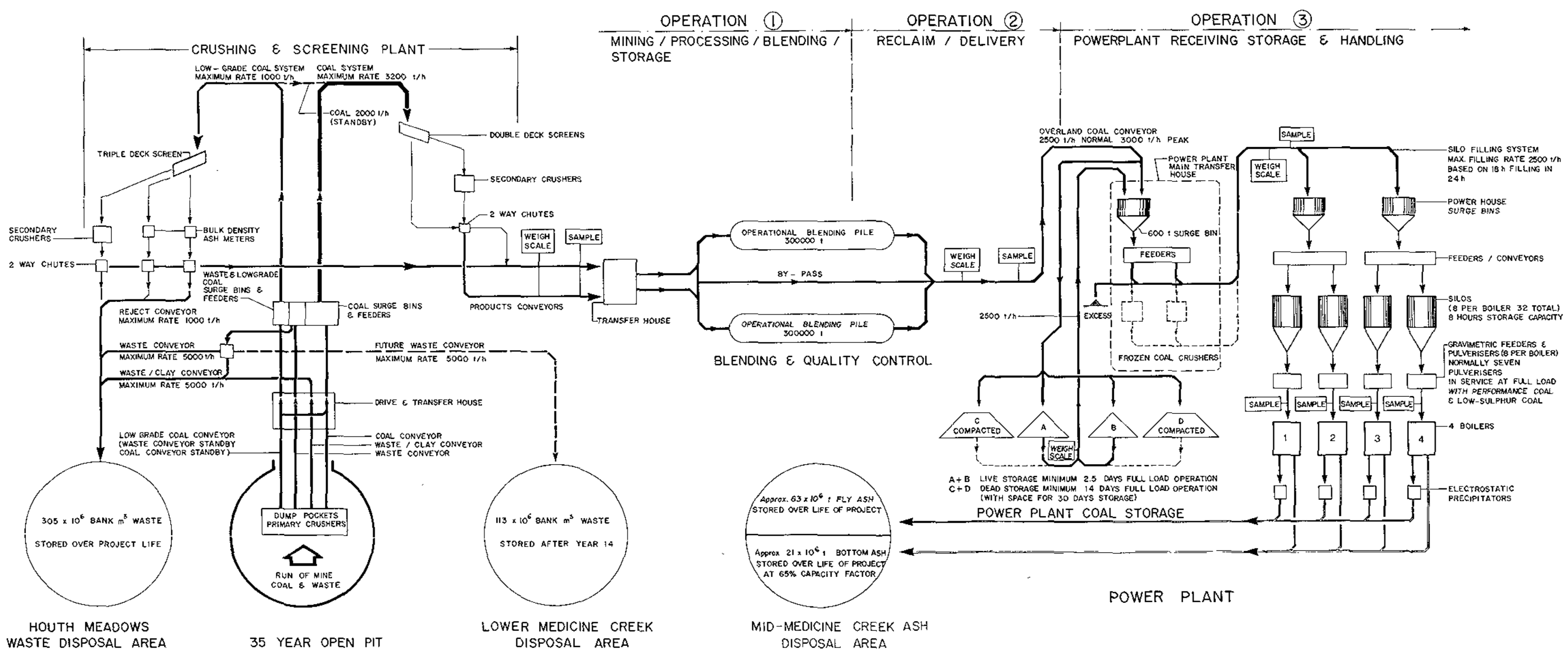
TABLE 8-9 Sheet 1 of 2

PROJECT COAL-HANDLING SYSTEM

OPERATING REGIME - EXAMPLES OF VARIOUS OPERATING CONDITIONS

Operating Condition	Mine Operation 1: Mining Processing Blending	Mine Operation 2: Reclaiming Loading Delivery	Powerplant Operation 3: Receiving and Handling to Boilers
Normal silo-filling four units at full load. (Power production maximum for current period of production.)	Mine and process coal to build blending pile at a rate of 40,000 t/d. Hourly rate varies to suit mining operation.	Reclaim, load and deliver 2,500 t/h for 18 hours out of 24 from blending pile.	Receive and fill silos for 18 hours out of 24 at a rate of 2,500 t/h.
Normal silo-filling at 70% full load. (Power production 70% for current period of production.)	Mine and process coal to build blending pile at a rate of 28,000 t/d. Hourly (or daily) rate varies to suit mining conditions.	Reclaim, load and deliver 2,500 t/h for 12½ hours out of 24 from blending pile (or 4 hours/shift).	Receive and fill silos for 12½ hours out of 24 at a rate of 2,500 t/h or (4 hours/shift).
Operations 1 and 2 lost temporarily. (Mine and blending, etc.)	Out	Out	Reclaim and fill silos for 18 hours out of 24 from storage at a rate of 2,500 t/h. (Minimum storage about 14 days at full load.)
Operation 3 lost temporarily. (All powerplant production.)	Continue to feed both blending piles until their capacity reached. Then switch, if necessary to waste material moving.	Cease delivery until powerplant calls for coal for silo-filling or storage.	Silo-filling ceases. If auxiliary power is functional silos can be filled and coal can be accepted to storage.
Mine over-producing temporarily.	Continue building blending piles to capacity.	Deliver at rate advised acceptable by powerplant.	Fill silos normally. Excess to powerplant storage.
Operation 2 under-producing temporarily. (Blending, loading and delivering.)	Continue building blending piles to capacity.	Deliver at best rate possible. Restore planned rate as soon as possible.	Fill silos continuously with mine deliveries. Reclaim necessary quantity from storage to supplement.
Operation 1 under-producing temporarily. (Mine)	Restore planned rate as soon as possible. Continue to build blending pile at best rate possible.	Deliver at planned rate until blending piles used.	Continue normally unless shortfall of delivery occurs. Reclaim necessary quantity from storage to supplement.

Operating Condition	Mine: Operation 1: Mining Processing Blending	Mine: Operation 2: Reclaiming Loading Delivery	Powerplant: Operation 3: Receiving and Handling to Boilers
Lower-sulphur coal required for brief MCS operation.	No change.	No immediate change. Plan deliveries based on revised daily instructions from powerplant.	Reclaim lower-sulphur coal from live storage immediately and start filling silos. If necessary continue from dead storage. If necessary call for delivery of lower-sulphur coal from operation 1.
Lower-sulphur coal required for lengthy MCS operation.	Mine and process high-grade coal for delivery to power-plant bypassing operation 2.	If required, can continue to build blending piles with coal routed through low-grade coal facilities at 2,000 t/h.	Receive lower-sulphur coal only from operation 1 to fill silos.
Rebuilding perfor-mance coal stockpile at powerplant after major period of non-delivery.	Mine at maximum rate to keep up with operation 2.	Deliver maximum rate powerplant can take from overland conveyor as long as powerplant can accept maximum flow.	Receive coal at maximum overland con-veyor capacity (3,000 t/h) continu-ously. Excess coal to stockpile. Rebuild live and dead storage. Re-building live storage on this basis would take about two days even with four units at full load. Rebuilding 14-day dead storage pile would take up to 17 days on this basis.
Rebuilding lower-sulphur stockpiles at powerplant while burning performance coal.	Deliver lower-sulphur coal for six hours out of 24 bypassing operation 2.	Deliver performance coal for 18 hours out of 24.	Performance coal to silos. Lower-sulphur coal to storage.

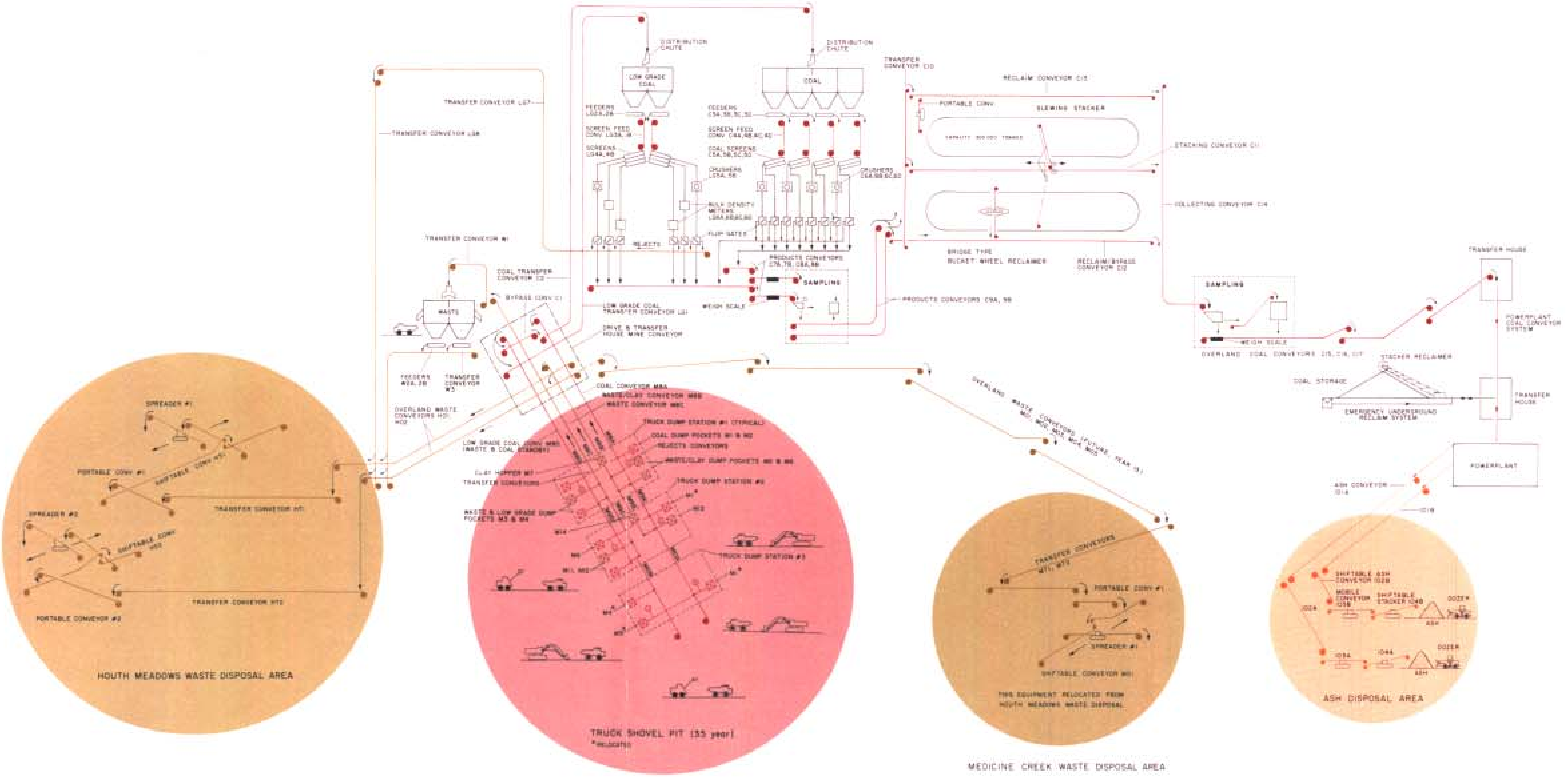


HAT CREEK PROJECT

FIGURE 8-1

Overall Project Flow Diagram

SOURCE: British Columbia Hydro and Power Authority



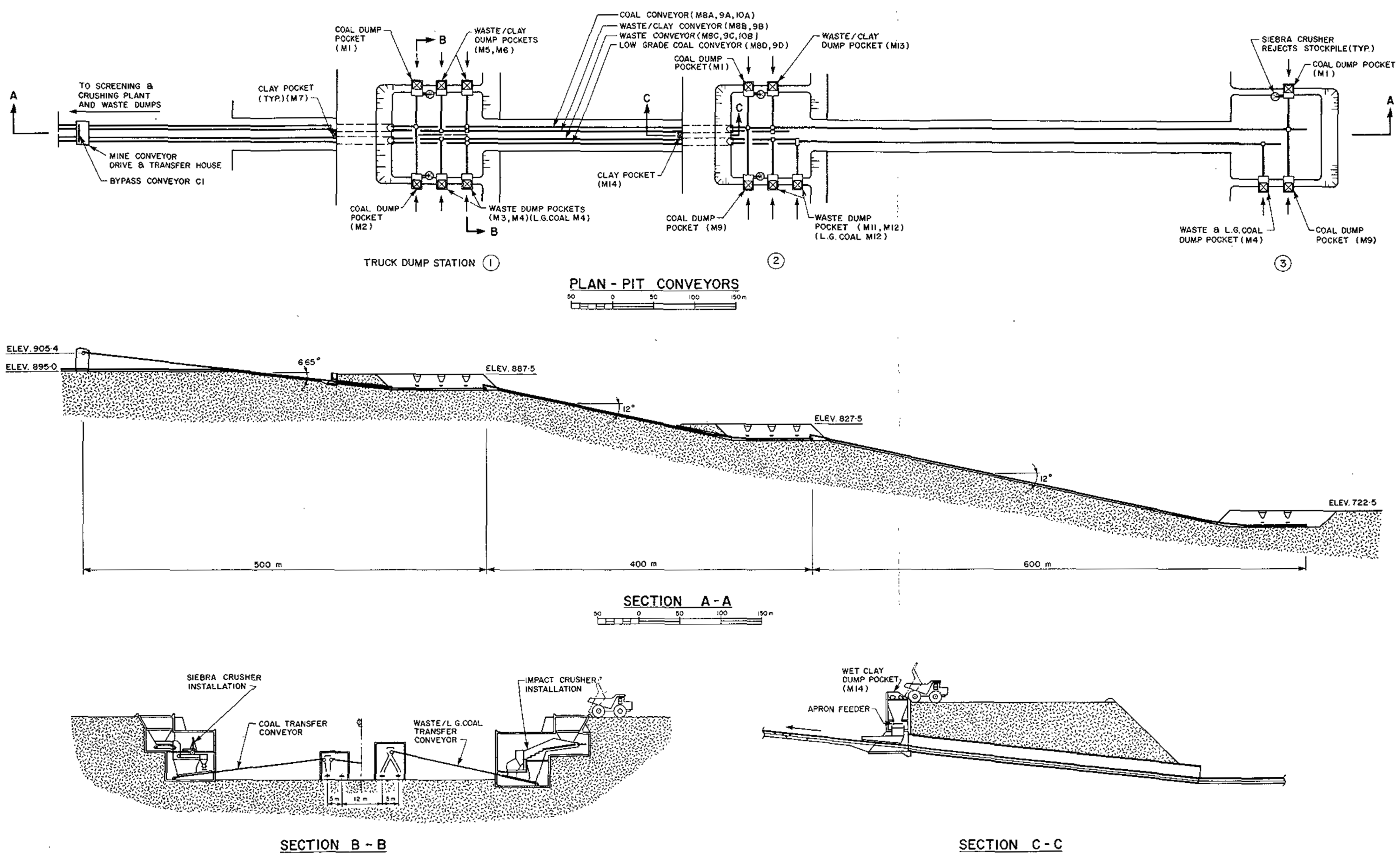
HAT CREEK PROJECT

FIGURE 8-2

MATERIALS HANDLING SYSTEM FLOW DIAGRAM

00142 2/2 1

SOURCE: Cominco — Monenco Joint Venture (5)



HAT CREEK PROJECT

FIGURE 8-3

General Arrangement Mine Conveyors and Truck Dump Stations

SOURCE: Simon-Carves of Canada Ltd.

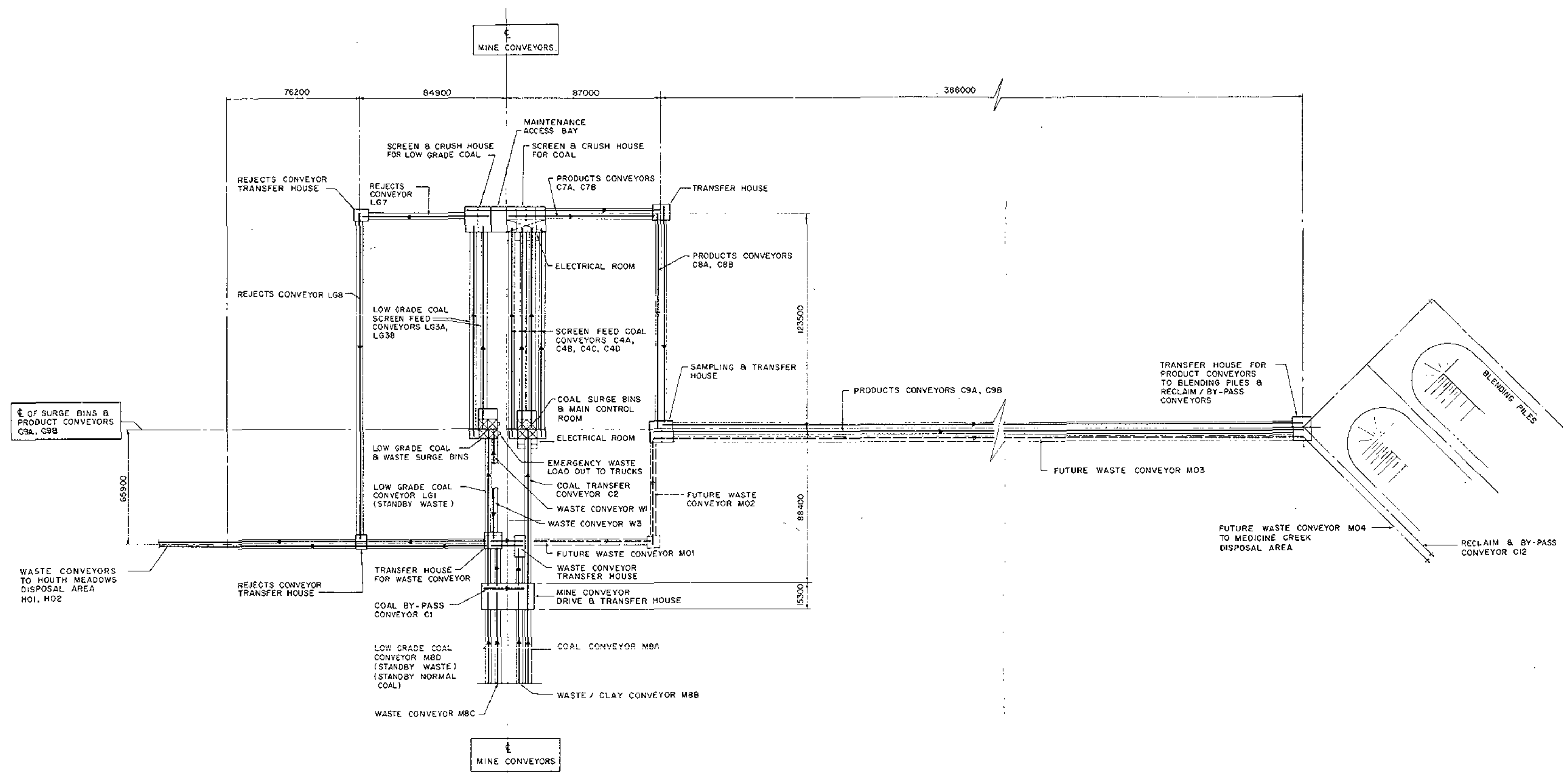


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FIGURE 8-4

General Arrangement Screening and crushing

SOURCE: Simon-Carves of Canada Ltd.

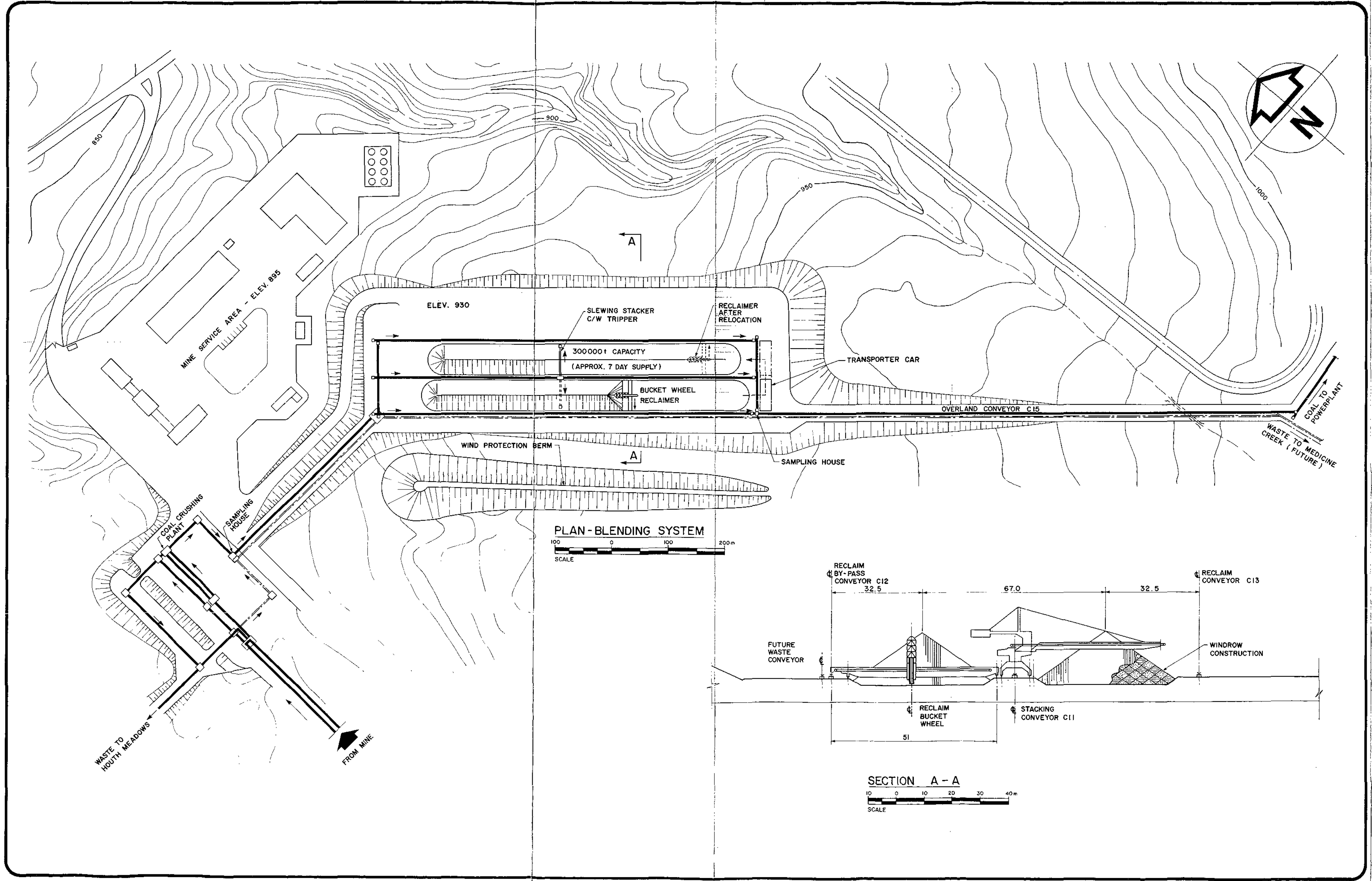

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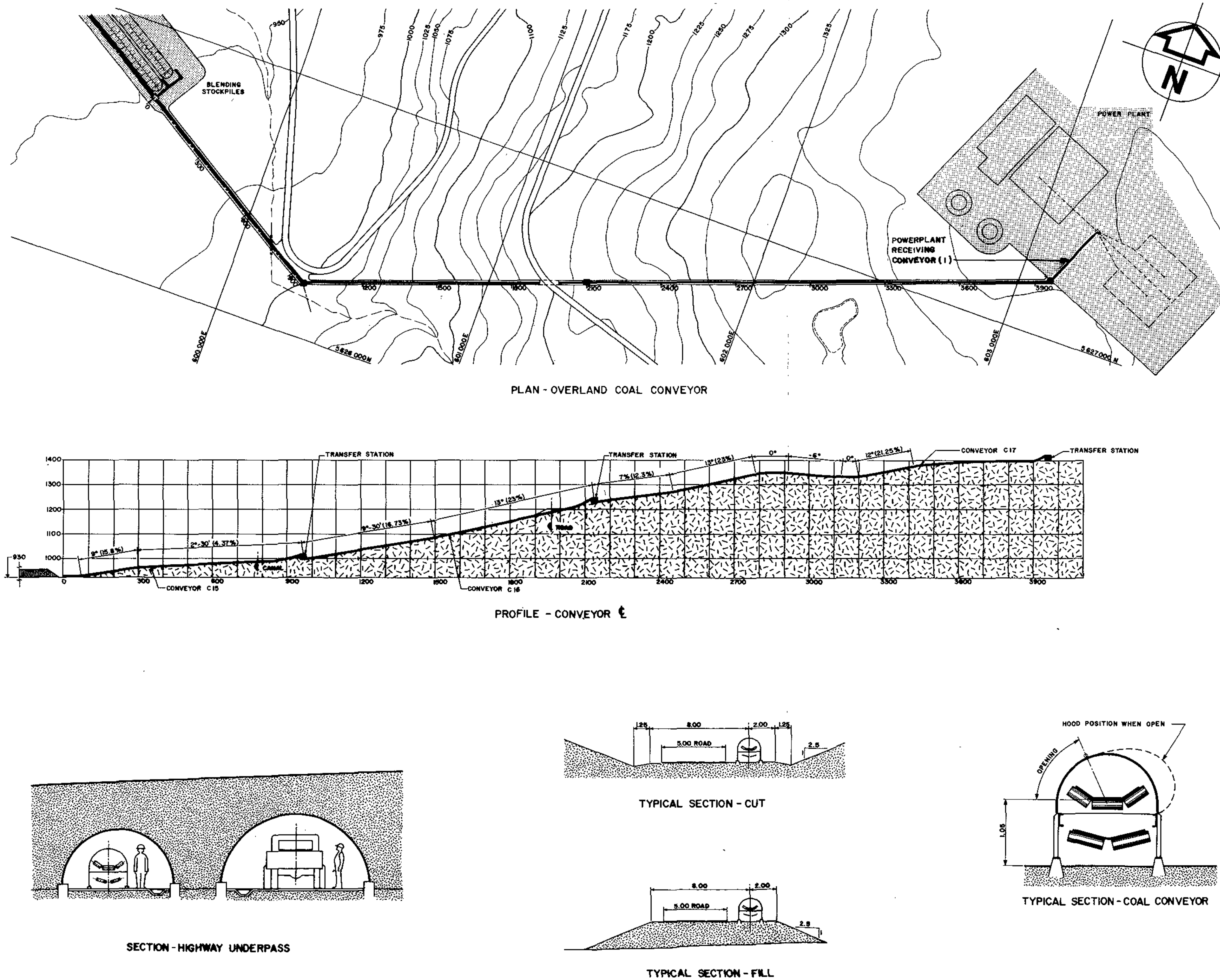
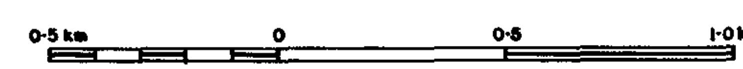
FIGURE 8-5

General Arrangement Blending / Reclaiming



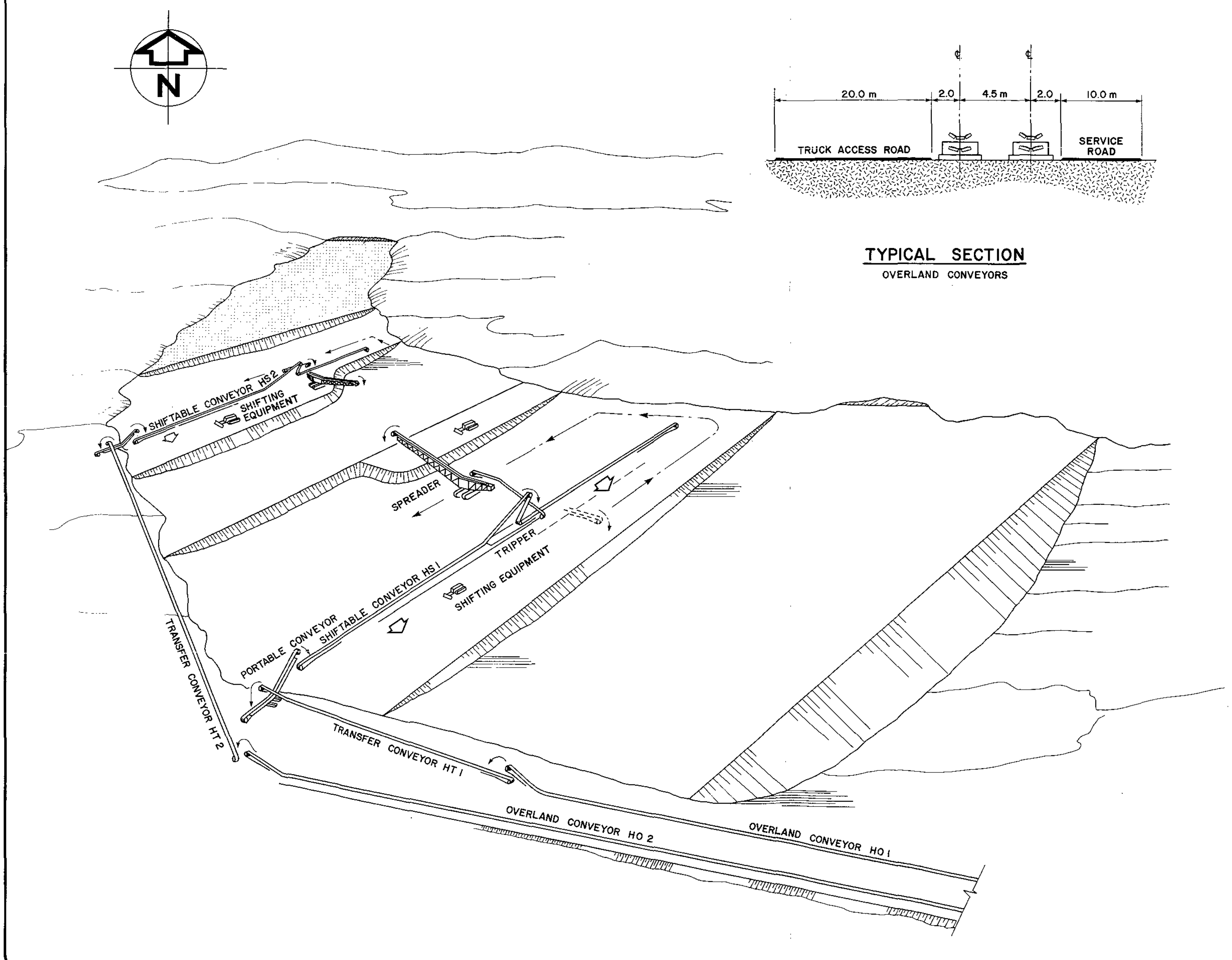


HAT CREEK PROJECT

FIGURE 8-6

General Arrangement Overland Coal Conveyor

SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 8-7

Pictorial View Waste Dump Development

SOURCE: British Columbia Hydro and Power Authority

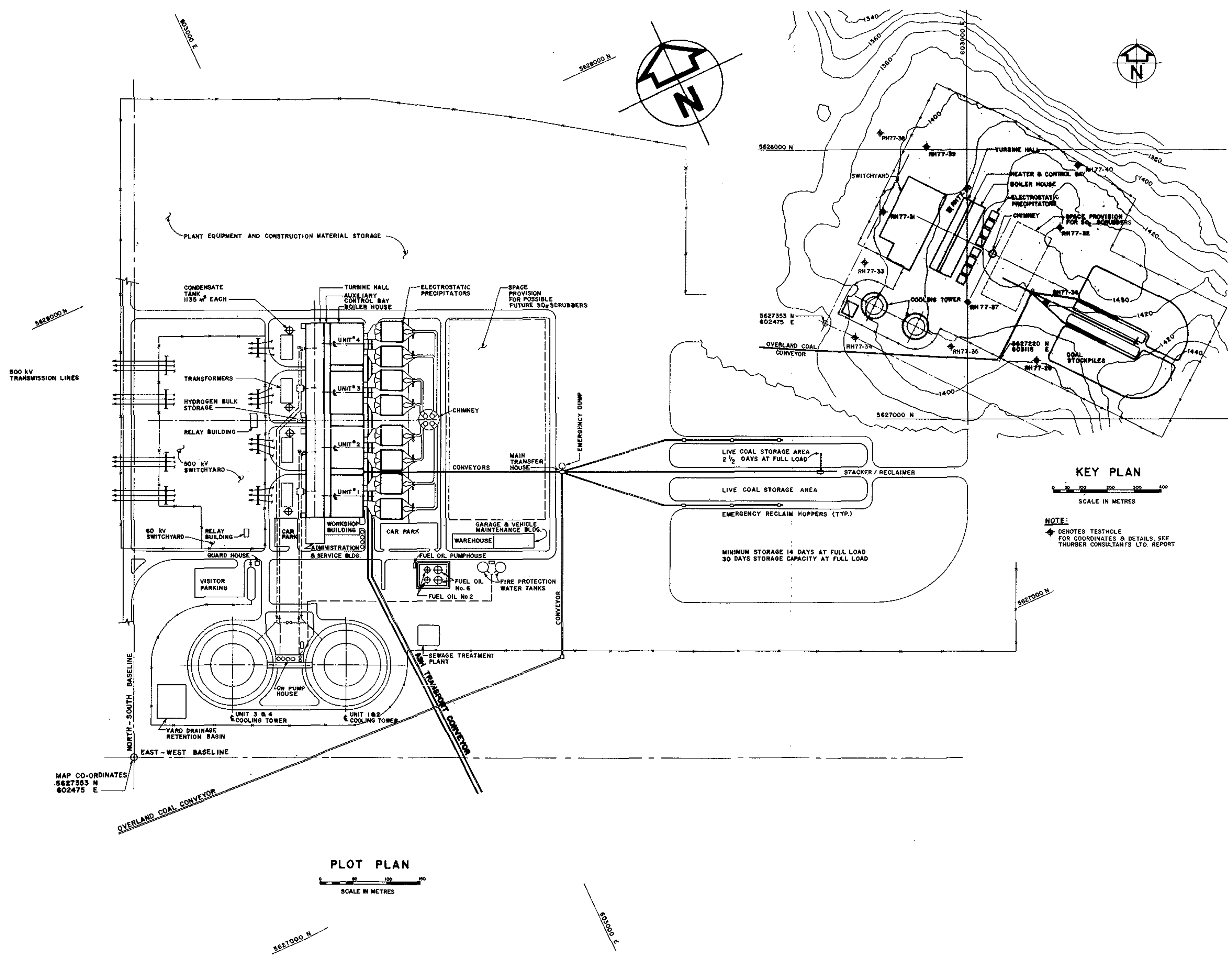


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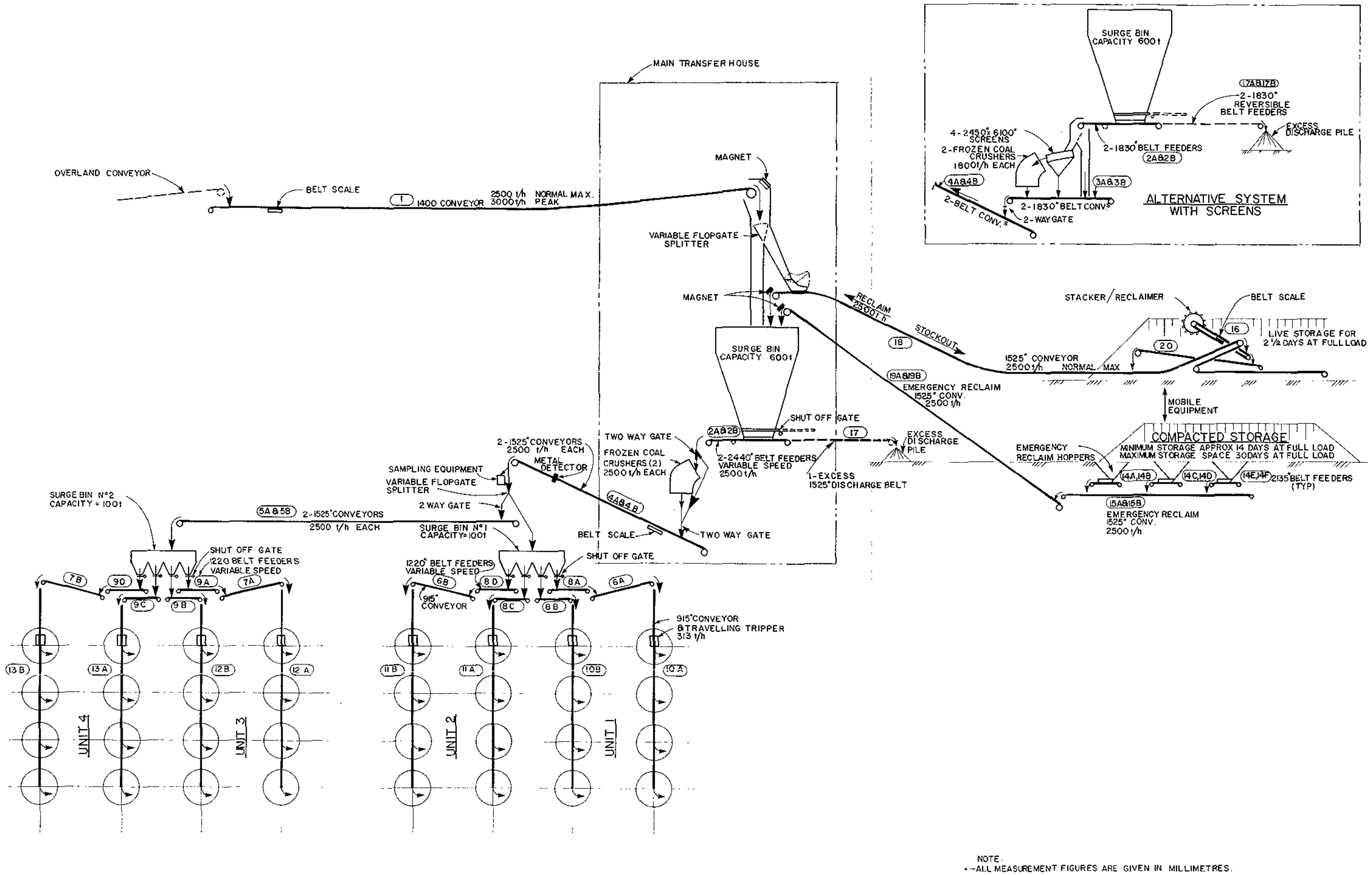
FIGURE 8-8

Powerplant — Plot Plan

SOURCE: Integ-Ebasco

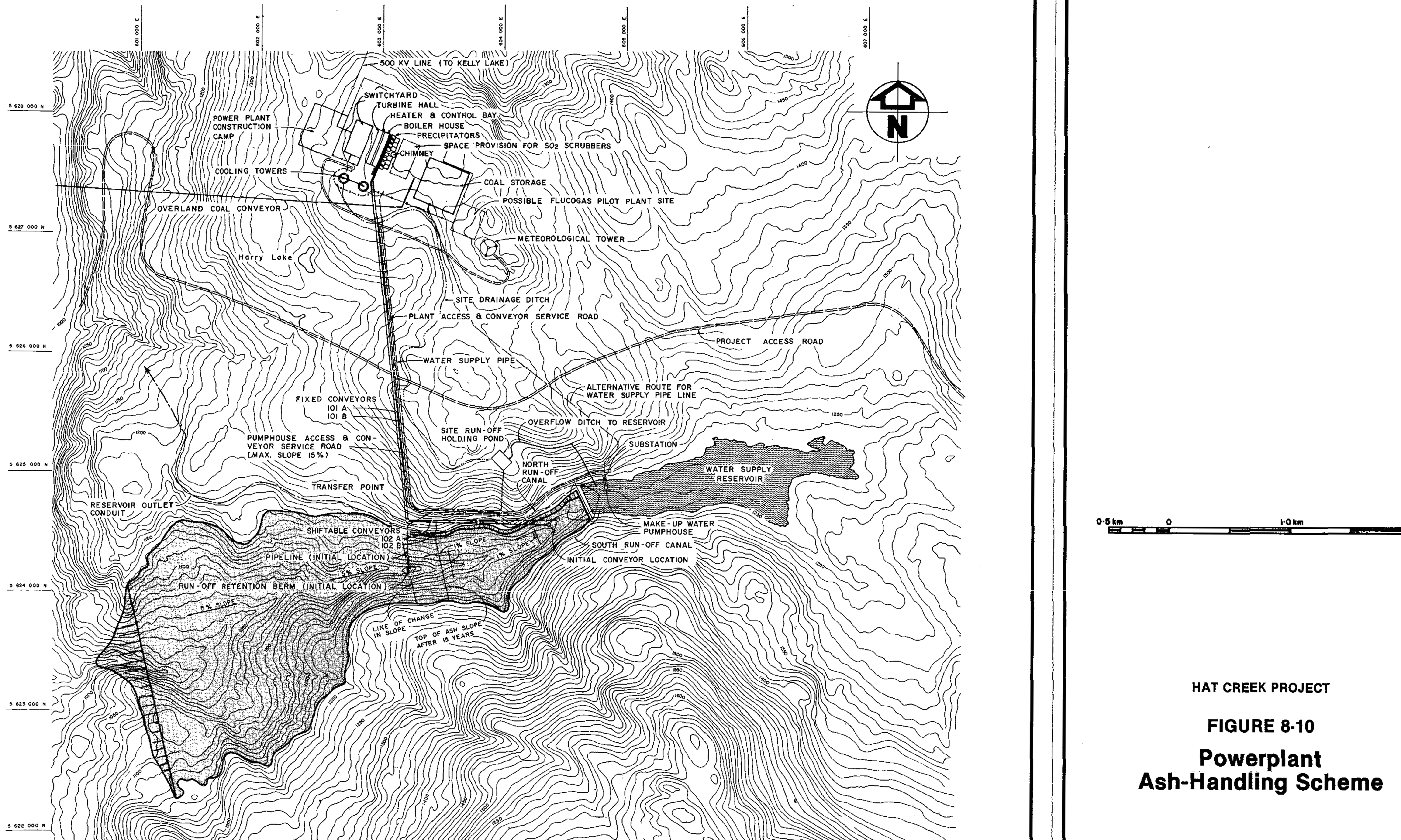


HAT CREEK PROJECT

FIGURE 8-9

Powerplant Flow Diagram of Coal System

SOURCE: Integ-Ebasco

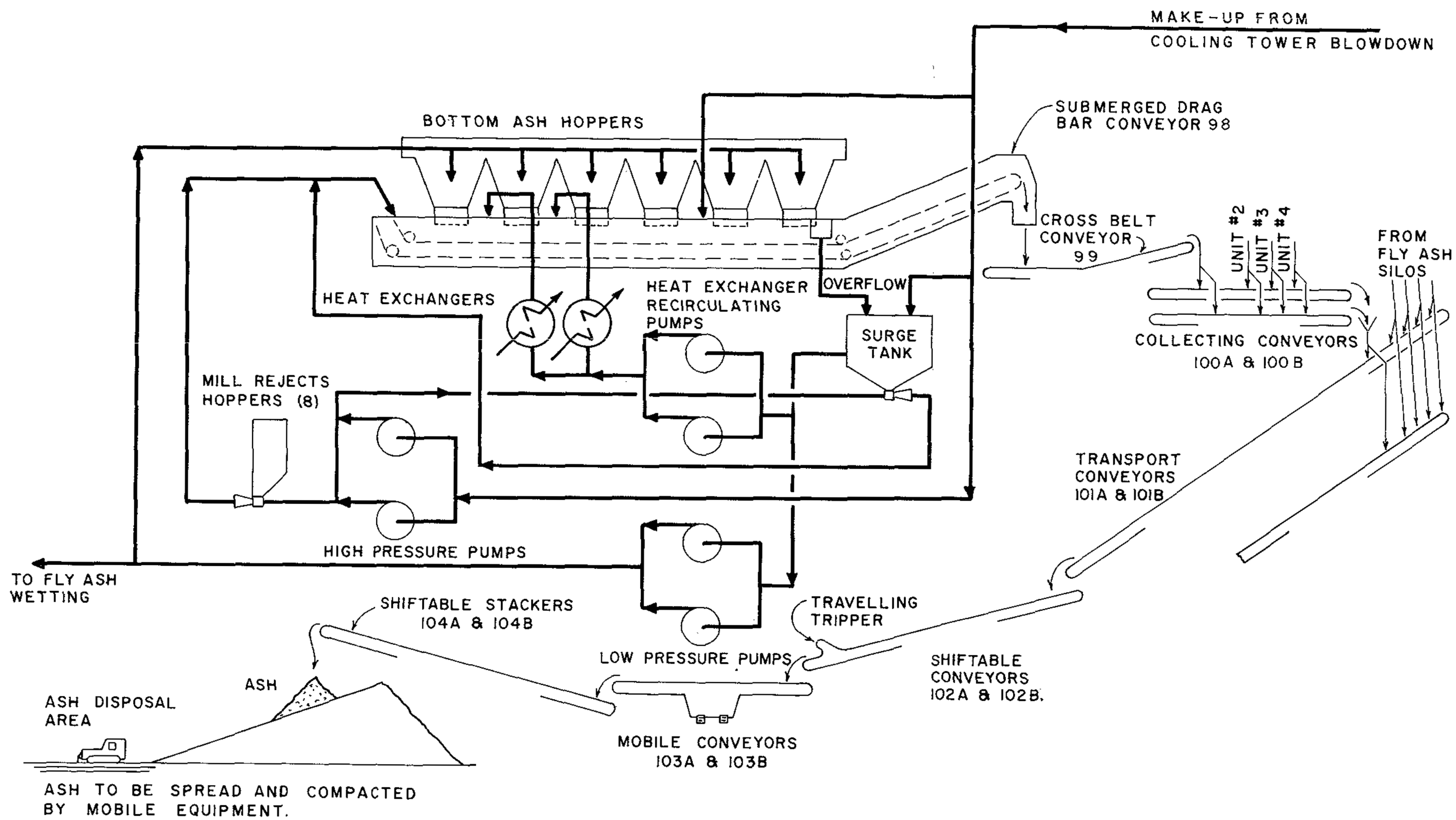


HAT CREEK PROJECT

FIGURE 8-10

Powerplant Ash-Handling Scheme

SOURCE: Integ-Ebasco

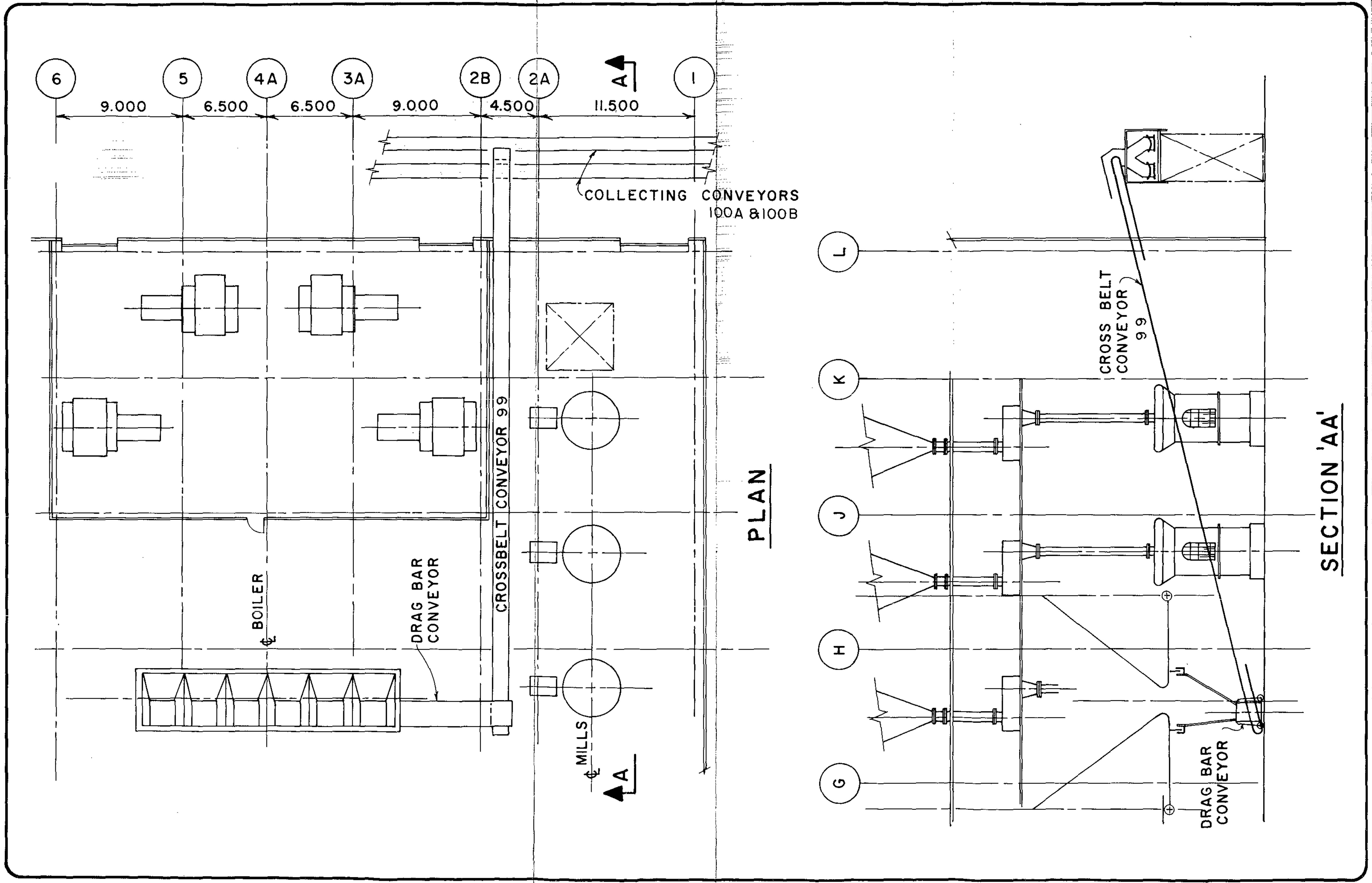


HAT CREEK PROJECT

FIGURE 8-11

POWERPLANT – FLOW DIAGRAM BOTTOM-ASH REMOVAL

SOURCE: Integ-Ebasco

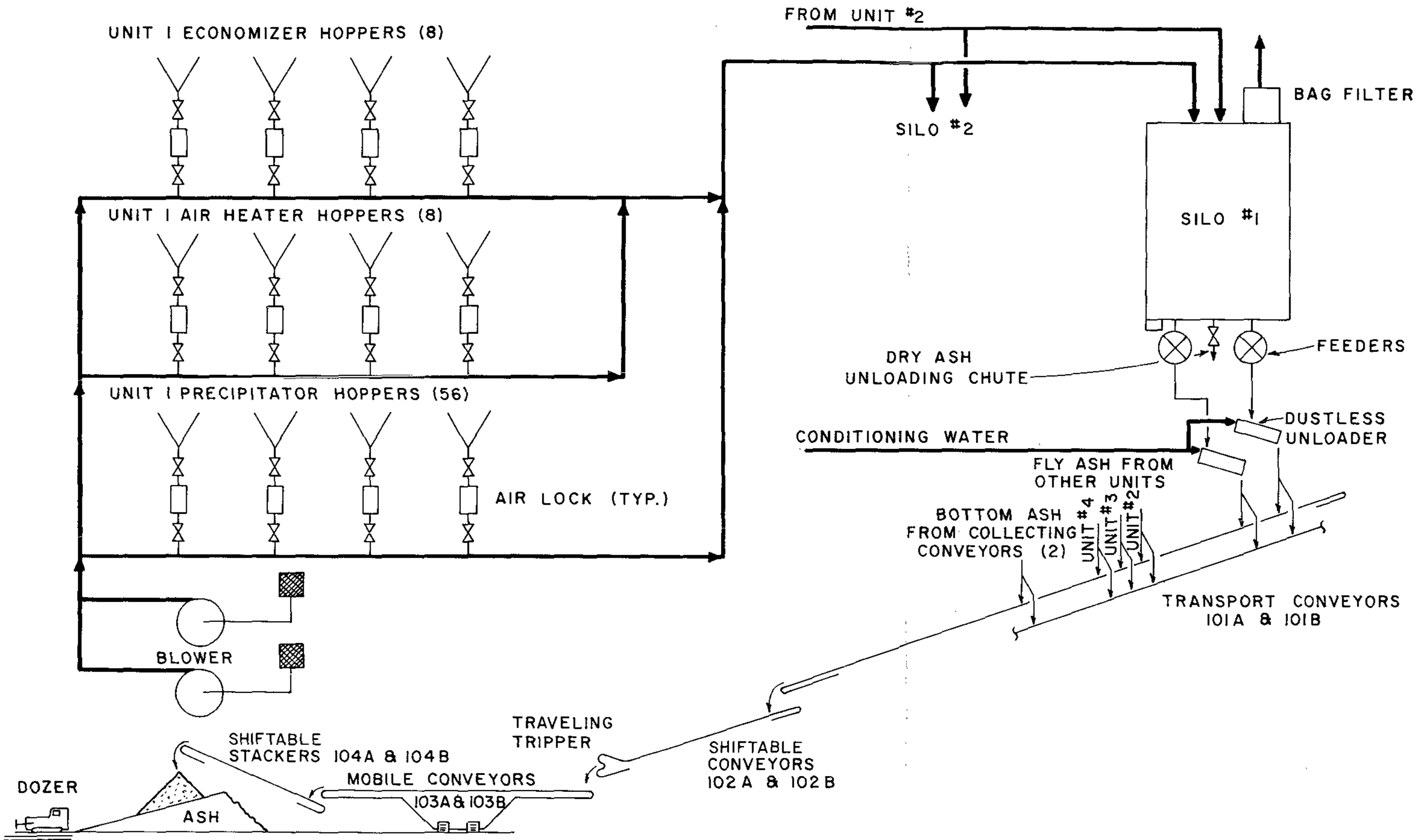


HAT CREEK PROJECT

FIGURE 8-12

Powerplant–General Arrangement Drag-bar Conveyor for Bottom-Ash

SOURCE: Integ-Ebasco

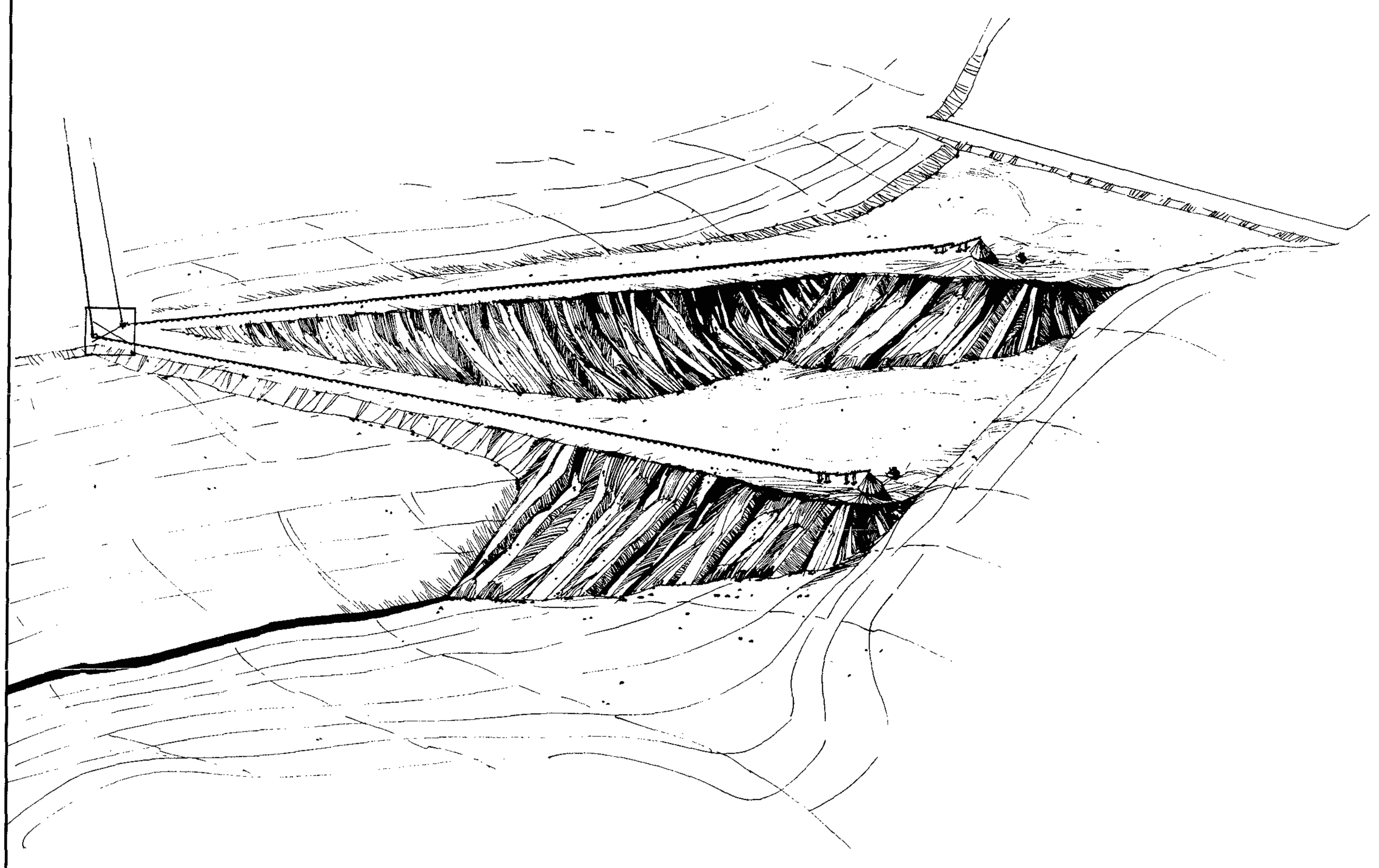


HAT CREEK PROJECT

FIGURE 8-13

Powerplant — Flow Diagram Fly-Ash Pressure System

SOURCE: Integ-Ebasco



HAT CREEK PROJECT

FIGURE 8-14

Isometric View
Powerplant - Ash Disposal System

SOURCE: Integ-Ebasco

9 EQUIPMENT

No.	LIST OF TABLES	Page

SECTION 9

EQUIPMENT SELECTION

This section discusses the selection of the equipment that will be used in the development and operation of the mine. The initial equipment is scheduled to become operational in Year -2 to commence pre-production development.

For the purposes of this study it has been assumed that any earth-moving required outside the limits of the pit for site preparation and construction purposes would be performed by contractors' equipment. The costs of this work have been provided in the construction capital cost estimates. This work could be reassigned to the mine crew and equipment with a minor revision of the schedule.

The mining equipment is divided into two groups: production equipment for loading and hauling coal and waste; and support equipment to execute the numerous other tasks required for the continuous, efficient operation of the mine.

9.1 PRODUCTION EQUIPMENT

A preliminary evaluation of possible mining equipment was conducted to determine its suitability for the proposed methods of operation. The equipment that passed this initial screening was subjected to a detailed cost and productivity analysis in the context of the mine plan and schedule.

The cost and productivity analysis was performed considering the shovels and trucks as a system for three critical mining periods:

(1) Year 6 - utilization of first dump pocket only; first high production period;

(2) Year 9 - utilization of first and second dump pockets; multiple mining areas;

(3) Year 21 - utilization of first, second, and third dump pockets; high coal production from lower benches.

These time periods include both long and short distances with uphill, downhill, and level hauls. For each year, haul road profiles were developed and truck travel times calculated. Fixed cycle times were developed for each shovel-truck combination. On level hauls, trucks were limited to a maximum speed of 40 km/h. On downhill hauls the speed limits were established to provide maximum braking capability.

9.1.1 Coal Mining

1. Shovels

The selection of the coal shovel is dictated by the decision to adopt selective mining methods. The most effective shovel for this purpose is the hydraulic excavator. Because of the variability of the deposit, the more shovels available for operation, the easier it is to maintain a consistent quality of output. The need to provide flexibility for quality control and to permit effective partings removal at 2 m was balanced against the economics of using larger equipment. The machine selected for loading coal is a hydraulic shovel with a 10.7 m^3 bottom-dump bucket equivalent to the Poclain 1000.

In addition to the scheduled coal production, these shovels have been assigned a quantity of waste partings and low-grade coal to be removed each year equivalent to 20% of the coal tonnage.

Over the project life, between two and three shovels are capable of loading the assigned quantities of coal and parting materials. To provide the necessary flexibility for producing a uniform quality of coal, and to accommodate extended periods of powerplant operation at maximum capacity, four electrically-powered hydraulic shovels will be operational, except during the initial buildup to full production and a tailing-off period in the latter years of the project.

In the peak production years, coal will be mined from as many as 15 benches. During this period, a fifth shovel has been provided for to reduce the impact of numerous shovel moves. This additional shovel would be a diesel-powered unit and would be supplied with a backhoe attachment as well as the standard shovel front. This unit provides mobility and flexibility to the operation. The backhoe attachment will also be useful in excavating sinking cuts and assisting in selective mining.

2. Trucks

Three sizes of diesel-electric haulage trucks were evaluated operating in conjunction with the 10.7 m^3 hydraulic shovel. These trucks were rated at 77 t, 91 t, and 109 t.

The economic analysis performed for the three selected critical periods showed a marginal cost advantage for the 77-t truck over the 91-t truck, with the 109-t truck ranked third. In reviewing the fleet size developed in the analysis, a better balance of trucks to shovels was obtained with the 77-t trucks, which were confirmed as the most suitable coal truck.

During the peak production years the number of 77-t trucks required ranges from nine to 11. The principal specifications of the truck are: 77 m^3 coal box; 1,000 hp engine; 24.00 x 49 tires, and a 23:1 gear ratio.

9.1.2 Waste Removal

There are two principal types of waste materials to be moved: consolidated and unconsolidated. The consolidated materials are typified by the claystones and siltstones of the Medicine Creek and Coldwater formations. Glacial till, and the sands and gravels on the East side of the pit are classified as unconsolidated materials, along with large quantities of bentonitic slide material on the West side.

1. Shovels

The two different categories of waste material to be mined present very different problems. The consolidated waste is a saturated, soft, cohesive material, which, when frozen, will form a rock-like crust a metre or more deep during an extended cold weather period. When blasted, frozen clay breaks into chunky pieces that are not compatible with conveyor transportation. An alternative approach to blasting the claystones and siltstones would be to blast the material prior to freezing, using crater blasting techniques, a method that has proven effective in tar sands. However, because of the high moisture content of the clays, the effectiveness of this approach is questionable until operational testing can be done.

Because of the nature of the claystones, it was concluded that the most effective method of excavating this material would be to use hydraulic shovels. A Demag 241 with a 14.5 m^3 bucket is the only production unit in this size range currently available.

The application of this unit would require the assistance of a D-9 ripper to handle the frozen toe. Special attention will also be required in operational planning to maintain continuous operation during the Winter months to prevent the face freezing. It will also be necessary to prevent traffic travelling on top of material to be mined during the Winter, because of the greater depth of the frozen layer that this causes.

The unconsolidated materials present less serious excavating problems. This material can be excavated with either a standard mining cable shovel or a hydraulic shovel. A cost and productivity analysis was conducted to compare a 16.8 m^3 cable shovel with the 14.5 m^3 hydraulic shovel. The results showed marginal cost savings using the 16.8 m^3 shovel for loading unconsolidated waste. The study also demonstrated that additional equipment scheduling problems would be introduced with a mixed shovel fleet, causing an increase in the number and length of shovel moves. The hydraulic shovels have the additional advantages of being lighter in weight, exerting less ground-bearing pressure, and capable of travelling approximately twice as fast as the cable shovels.

These factors outweigh the insignificant cost savings of the cable shovel and resulted in the 14.5 m^3 hydraulic shovel being selected for loading both the consolidated and unconsolidated waste materials.

2. Trucks

Three sizes of diesel-electric haulage trucks being loaded by 14.5 m^3 hydraulic shovels were evaluated for waste haulage. These trucks were 109 t, 136 t, and 154 t. Other truck sizes were eliminated in a preliminary evaluation.

The economic analysis was performed using the same three critical periods identified above. The 154-t truck showed the lowest unit production costs and was selected for waste haulage. The principal specifications for the truck are: 90 m^3 rock box; 1,600 hp engine; 36.00 x 51 tires, and a 28.85:1 gear ratio.

Consideration was given to standardizing trucks for coal and waste, but the requirements for selective mining of coal dictate a smaller unit than is economically justified for moving larger quantities of waste over significantly different haul road profiles.

9.1.3 Operation at Maximum Capacity Rating

Under exceptional conditions the powerplant could be required, and be able, to operate at its full rated capacity for an extended period of up to six months. The plan presented in this report ensures that sufficient equipment is provided to meet the normal operational requirements established by the forecast operating regime. The purchase of additional equipment to cope with an event that is unlikely to occur is not justified where contingency plans can be implemented. The mining operation, as planned, has considerable flexibility to meet a number of widely varying conditions that can be used to meet emergency requirements for additional coal production.

It is assumed that any extended period of powerplant operation at maximum capacity rating will span the Winter months. This assumption is supported by the fact that maximum power demands occur in this period, transmission lines from more distant hydro-electric projects are exposed to greater hazards in the Winter, and the thermal powerplant at Hat Creek has extended maintenance scheduled for each unit during the Summer. It is also assumed that extended operation will not be required in successive years. Should the latter assumption prove wrong, additional equipment could be purchased. The additional equipment, primarily trucks, can usually be obtained with a three to six-month lead time.

The mining contingency plan only provides for mining additional coal during the emergency period. Existing plans provide for at least six months' coal to be uncovered at all times. In many time periods waste removal is even further advanced to facilitate a level production schedule. Thus it is not considered necessary to increase waste removal to cope with the emergency.

To meet the additional coal production requirement, adequate shovel-loading capability has already been provided to allow flexibility for coal production scheduling. The conveying systems are designed to handle peak hourly requirements. On an annual basis this means that the coal conveyor has a capacity 40% above the peak annual tonnage required. This does not include the use of the low-grade coal conveyor as a back-up facility. It is concluded that adequate conveying capacity exists.

The principal area for contingency planning is in the assignment of trucks. The re-assignment of one or two trucks from waste removal to coal production should be sufficient to provide the additional tonnage required. The larger 154-t waste trucks can be loaded by the 10.7 m^3 coal shovels. Although this would not use the trucks at maximum efficiency, the performance would be acceptable under emergency conditions.

Higher productivity is expected from trucks during the Winter months than during the remainder of the year. Experience in other operations indicates substantial productivity improvement in the Winter, primarily due to improved haul road conditions. This improvement is expected to be even more pronounced with the soft, weak materials at Hat Creek and will minimize the loss of waste production.

A further back-up system for coal production is the use of the smaller 32-t trucks loaded either by 10.7 m^3 shovel or by front-end loader. These trucks are available for use during the Winter months because of the limited road construction activity at this time. As a final back-up, additional shovels and trucks can be redirected from waste removal to coal production, and any shortfall in waste can be made up through the use of contractors.

9.2 SUPPORT EQUIPMENT

The support equipment required for the mine can be broken down into four principal categories: road construction and maintenance; mine support; material-handling support; and general service equipment. There is a considerable overlap in the application of specific types of equipment to the different categories. Table 9-1 summarizes the total requirements of the major equipment types and their applications.

9.2.1 Road Construction and Maintenance

Because of the weakness of the materials at Hat Creek, considerable road construction will be required on virtually every bench to ensure that the production trucks and service equipment can operate efficiently. Truck haul roads will generally be constructed 25 m wide, with a one metre base topped with 20 cm of crushed gravel. It is estimated that half the roads to be constructed in a given year will require major construction. The remaining roads will be constructed on more stable gravel, till, and coal requiring only top dressing.

A major road building program will commence in the pre-production development period and continue throughout the life of the mine. Roads on the waste dump will be built intermittently; most of this work will be required immediately prior to the relocation of conveyor belts. The road construction requirements, shown in Table 9-2, decrease as permanent haulage roads are developed. The permanent roads above the present valley bottom will be operational in about Year 15, and below the valley bottom in Year 20.

Major haul road construction is restricted to the months of the year when the ground is not frozen, to avoid the problems that Winter-built roads have due to trapped excess moisture and poor compaction. It is planned to take advantage of the frozen conditions to gain access to the more unstable areas, using roads with only a minimum of surface topping.

The construction method for roads uses dozers to cut and fill where required and rough-level the foundation. Scrapers, aided by push dozers, would subsequently pick up run-of-mine gravels - mainly from the East side of the valley - in order to build up the lower portion of the sub-base. The upper portion of the sub-base, including the running surface, would be crushed gravel or, when available, baked zone materials. The gravel crusher would be located on a gravel bench and fed by front-end loaders. It would convey crushed material to a pile for pick-up by scrapers or by front-end loaders for truck transport. Graders and vibrating compactors would complete the final surface. Provision has been made for water trucks for both the construction phase described above and for subsequent road maintenance practice.

From pre-production to Year 5, the bulk of road construction activity will be handled by 705 hp tandem-powered 16 m^3 struck capacity scrapers assisted by D-9 push-cat. During pre-production, the scrapers will remove an estimated 2.2 million m^3 of waste materials, the majority of which will be used to construct roads. From Year 6 onwards, the haul distances required for road construction lengthen beyond the economic limits for scraper use, and the 32-t trucks loaded by front-end loaders, assisted by D-9 dozers, become the primary equipment for this function. From Year 12 on, the number of working benches decreases and with it the amount of road construction in each year.

The bench pioneering operation is intermittent and involves cut and fill work at designated bench elevations. This work will provide a sufficient number of ramps and level operating space for subsequent shovel/truck operations. The principal equipment used for bench pioneering will be scrapers and D-9 dozers.

Caterpillar 16-G graders or equivalent will be used in the construction and maintenance of the haul roads. Grader requirements were established based on each grader maintaining 3 km of road per shift. A smaller, 14-G grader has been provided for in the estimates to maintain the narrower service roads around the property.

Provision has been made in the estimates for the purchase and operation of the major road construction equipment and for all aspects of road maintenance, including snow removal, dust and ice control, gravel crushing, and ditch maintenance.

9.2.2 Mine Support

1. Drilling

It is anticipated that the majority of the coal and waste materials can be excavated using hydraulic shovels without blasting.

However, exploration drilling has identified bands of cemented conglomerate that may require fragmentation to permit efficient loading operations. In addition, during the bulk sample program, an isolated plug of fused material was uncovered in the burn zone material that had to be blasted. No further plugs of this type have been identified in the drill holes, but it is reasonable to expect that more, similar plugs exist. An allowance has been made to drill and lightly blast 10% of the waste materials to cover these events.

Two drills have been included in the equipment list for the drilling of blast holes: a truck-mounted auger drill and a tracked percussion drill, complete with a 17 m^3/s air compressor.

The principal production drilling would be performed by the auger drill, which would require only one shift a day, five days a week, to perform this task. The auger drill would also be used for close-spaced drilling for coal quality control. The percussion drill would be used in irregular terrain and in drilling material too hard for the auger.

Because of the expected low volume of drilling and blasting, the purchase of an AN-FO mix truck is not justified. Pre-packaged explosives transported by a 5-t stake-body truck would be used.

2. Shovel Support

Rubber-tired dozers will be supplied for clean-up operations around the shovels and for pit floor maintenance. Each Cat 824B rubber-tired dozer will service up to three operating shovels. During peak production years, three of these machines will be required.

Assistance in selective mining and excavating frozen waste materials will be provided by D-9 rippers. To assist selective mining of coal, the dozers will remove bands of coal or waste that the shovel has difficulty in digging cleanly. The support to the waste shovels will be: to rip frozen material on the crest of the bench to prevent unsafe overhangs developing; and also to rip frozen toe material to ensure the excavation of flat benches.

3. Pit Clean-up

Pit clean-up will be provided using front-end loaders and 32-t trucks. The clean-up function includes removal of boulders and petrified wood rejected by the shovels and removal of accumulations of saturated clay from bench toes or bladed-off haul roads.

9.2.3 Material-handling Support

1. Waste Dumps

Each waste stacker system will be provided with two D-8's for maintaining a uniformly sloping dump surface, general dump maintenance, and shifting the waste conveyors. For the conveyor-shifting operation, additional dozers will be required either from the other stacker or the mine. The dozers assigned to the waste dumps will normally remain there because of the travel distance to other parts of the mine. Other duties for the dozers include any necessary land clearing and snow removal.

Periodically, conveyor pads must be constructed prior to relocation of a conveyor and areas of the dump that are inaccessible to the stackers must be filled. After Year 16, the filling of these areas is required on a continuous basis. This work will be performed using front-end loaders to load 32-t trucks with waste deposited by the stacker in a location as close as possible to where it is required.

2. Secondary Crushing and Blending Area

A 5.4 m^3 front-end loader will be assigned to the secondary crushing and blending area for clean-up and stockpile maintenance. Stockpile maintenance includes any movement of coal required to permit the efficient operation of the stacker and reclaimer and also to dig out any areas where spontaneous combustion develops.

Emergency reclaim capability, when the reclaimer is unable to operate, will be provided through a portable skid-mounted conveyor fed by front-end loaders. The loader assigned to the area would be augmented by equipment from the mine.

3. Conveyor Line Clean-up

Spillage under the conveyors will be removed using a tracked dozer loader equipped with a rake attachment. Two of these machines will be required to cover the full length of the installed conveyors.

Routine clean-up around the truck dump stations will be handled by the rubber-tired dozers assigned to shovel support. Removal of oversize material from the crushers will be handled periodically by front-end loaders and 32-t trucks.

9.2.4 Service Equipment

A fleet of emergency vehicles and service equipment will be required to maintain the mobile equipment fleet and the conveyor belt systems:

		Number Required
(1)	Emergency vehicles:	
	1. Fire trucks	2
	2. Ambulances	2
(2)	Cranes:	
	1. 70 t	1
	2. 50 t	1
	3. 15 t	3
(3)	Recovery vehicles:	
	1. Low-boy tractor trailer	1
	2. Hi-boy trailer	1
(4)	Mechanical maintenance vehicles:	
	1. 5-t mechanic's truck	4
	2. Tire truck	1
	3. 22,730-L fuel truck	2
	4. Lube truck	2
	5. 3-t Hiab truck	2
	6. 17-m^3 compressor	1
	7. Steam cleaner	3
	8. 50-kW portable generator	1
(5)	Fuelling station	1
(6)	Electrical maintenance:	
	1. Line truck	1
	2. 1-t electrician's van	2
	3. Light plants	6

		Number Required
(7)	Warehousing equipment:	
	1. 2,700-kg forklift	2
	2. 4,500-kg forklift	1
(8)	Personnel transportation:	
	1. 1-t four-wheel drive trucks	8
	2. 1-t pick-ups	26
	3. 3/4-t pick-ups	26
	4. Management cars	7
(9)	Pit buses:	
	1. 24-passenger	5
	2. 10-passenger	2

Miscellaneous other equipment required includes: conveyor belt vulcanizers, belt reelers, and cable reelers.

TABLE 9-1

MINE EQUIPMENT LIST

Equipment Type and Job Application

Item		Description	Fleet Size
Production Equipment:			
10.7 m^3 Shovel			
	Years:	- pre-production	1
		- 3	3
		- 5 to 23	5
		- 24 to 35	4
Job application:		- Loading coal and low-grade coal	
		- Loading waste partings	
14.5 m^3 Shovel			
	Years:	- pre-production	1
		- 3	3
		- 5 to 18	4
		- 23 to 35	2
Job application:		- Loading consolidated waste	
		- Loading unconsolidated waste	
77-t Truck			
	Years:	- pre-production	3
		- 1	6
		- 5 to 7	10
		- 13 to 18	11
		- 25 to 35	8
Job application:		- Hauling coal and low-grade coal	
		- Hauling waste partings	

Item	Description	Fleet Size
154-t Truck		
Years:	- pre-production	3
	- 1	6
	- 5	12
	- 10 to 14	14
	- 25 to 35	8
Job application:	- Hauling consolidated waste	
	- Hauling unconsolidated waste	
Support Equipment:		
D-9 Dozers		
Years:	- pre-production to 3	4
	- 5 to 22	5
	- 23 to 35	3
Job application:	- Road construction and maintenance	
	- Bench pioneering	
	- Scraper assistance	
	- Front-end loader assistance	
	- Shovel assistance:	
	- partings removal	
	- ripping hard and frozen materials	
	- pit clean-up	
D-8 Dozers		
Years:	- pre-production to 5	7
	- 6 to 10	8
	- 10 to 35	7
Job application:	- Road construction and maintenance	
	- Front-end loader assistance	
	- Bench pioneering and land clearing	
	- Snow control	
	- Dump activities:	
	- spreader assistance	
	- levelling of spoil	
	- dump maintenance	
	- conveyor shifting	

Item	Description	Fleet Size
D-7 Dozers		
Years:	- pre-production to 5	1
	- 5 to 26	2
	- 27 to 35	1
Job application:	- Road construction and maintenance	
	- Pit clean-up	
32-t Truck		
Years:	- pre-production	3
	- 4 to 35	6
Job application:	- Road construction and maintenance	
	- Conveyor pad and causeway construction	
	- Pit clean-up	
Wheel Dozer		
Years:	- pre-production	1
	- 1 to 4	2
	- 5 to 20	3
Job application:	- Shovel and bench clean-up	
	- Road maintenance	
Scraper - 16 m^3		
Years:	- pre-production to 7	6
	- 8 to 10	4
	- 11 to 35	2
Job application:	- Road construction and maintenance	
	- Topsoil removal and replacement	
	- Bench pioneering	
9.6 m^3 Front-end Loader		
Years:	- Mine life	2
Job application:	- Road construction and maintenance	
	- Shovel back-up	
	- Pit clean-up	
	- Emergency reclaim of coal stockpiles	

Item	Description	Fleet Size
5.4 m^3 Front-end Loader		
Years:	- pre-production to 3	2
	- 4 to 22	3
Job application:	- Road construction and maintenance	
	- Pit clean-up	
	- Oversize reject handling	
	- Dump activities	
	- Bench pioneering	
16G Grader		
Years:	- pre-production	3
	- 3	5
	- 4 to 12	6
	- 13 to 22	5
	- 23 to 35	3
Job application:	- Road construction and maintenance	
	- Grading on dumps	
14G Grader		
Years:	- Mine life	1
Job application:	- Road construction and maintenance	
	- Snow control	
Backhoe		
Years:	- Mine life	1
Job application:	- Construction and maintenance of ditches	
	- Construction and maintenance of dewatering sumps	
Gradall		
Years:	- Mine life	1
Job application:	- Road maintenance	
	- Ditch and culvert maintenance	

Item	Description	Fleet Size
Drills		
Type:	Auger - Mine life	1
	Percussion - Mine life	1
Job application:	- Exploration drilling - Close-spaced drilling - Burn zone materials drilling - Drilling of oversized boulders	
Additional Equipment:		
Water truck		3
Mobile crusher		1
Wheeled compactor		1
Towed compactor		1
Diesel pumps		10
CaCl spreader		4
Low-boy tractor/trailer		1
Hi-boy tractor/trailer		1

TABLE 9-2

ROADS SCHEDULED FOR CONSTRUCTION

(kilometres)

Year	In-pit Roads	Dump Conveyor Pads	Total
-2	4		4
-1	8	3	11
1	9	4	13
2	14		14
3	21		21
4	29		29
5	32		32
6	32		32
7	31	4	35
8	32	5	37
9	32		32
10	34		34
11	34		34
12	32		32
13	32	2	34
14	30		30
15	29		29
16	27		27
17	26	2	28
18	25		25
19	23		23
20	23	5	28
21	22		22
22	22		22
23	22		22
24	22		22
25	22	2	24
26	20		20
27	20		20
28	20		20
29	21		21
30	21		21
31	21		21
32	21		21
33	21		21
34	11		11
35	10		10
			882

10 MINE SUPPORT FACILITIES

Page

LIST OF TABLES

LIST OF FIGURES

SECTION 10

MINE SUPPORT FACILITIES

10.1 INTRODUCTION

This section describes the facilities required to support mining operations at Hat Creek. These support facilities include an administrative centre; maintenance, service, and emergency facilities; and utilities supply and distribution. In all cases, consideration has been given to functionality, safety, fire protection, and provision for expansion.

The Mine Services Area would be just North-East of the pit, covering an area of approximately 40 ha. Included within the services area are buildings for administrative and personnel requirements, equipment repair shops, storage, laboratories, and open areas for parking, additional storage, and equipment erection. Initial development of the services area would commence in the middle of Year 4, concurrently with erection of some major mining equipment, and nine months prior to "breaking ground" in the actual pit.

The two utility services costed in this study, water and power supply, are discussed in Sections 10.3.1 and 10.3.2.

The overall recommended mining operation does not require large quantities of water; at full mine development, the maximum average daily requirement would be 2,855 m^3, which would be sufficient for drinking water, fire protection, irrigation, and dust control. Water would be supplied from the powerplant supply line from the Thompson River, the pit rim reservoir on Hat Creek, and drainage from the mine area.

The electrical power supply and distribution system at the mine is designed to accept power from the 60 kV busbar at the proposed Hat Creek Generating Station, and to distribute it wherever required at voltages varying from 60 kV to 120 v. Construction power requirements would be met from an existing 60 kV line adjacent to Highway 12, North of the mine development. A maximum of eight portable sub-stations would be strategically located in the mine area. Operation of equipment is the largest power requirement of the project, the major equipment functions comprising mining in the pit, coal conveying, crushing and blending, and spreading operations at the waste dumps. Additional power demands include pit dewatering, mine area lighting, and complete electrical servicing of the buildings and facilities in the Mine Services Area.

10.2 MINE SERVICES AREA

10.2.1 Location

The proposed location of the Mine Services Area, to the North of the pit, was chosen for its proximity to the pit access road, out-of-pit conveying system, and Coal Blending Area. An additional consideration in choosing this location is that it does not affect the mining of future coal reserves in the Upper Hat Creek Valley. Geological investigations to date have indicated that no coal reserves are present beneath the proposed Mine Services Area. Furthermore, the proposed area for development is well drained, sufficiently level to preclude any major problems in site preparation, and large enough (40 ha) to accommodate the proposed service area, with ample room for expansion, during the anticipated 35-year life of the mine.

10.2.2 Facilities Required

The following structures and facilities comprise the Mine Services Area for the Hat Creek Mine:

Administration Building;
Maintenance Complex;
Mine Services Building;
Field Maintenance Centre;
Rubber Repair Shop;
Laboratories;
Lubrication Storage Building;
Fuel Storage and Dispensing Area;
Mine Dry;
Storage Areas.

The general layout of these facilities is shown on Figure 10-1.

10.2.2.1 The Administration Building

The proposed Administration Building, located in landscaped surroundings on the Western edge of the Mine Services Area, is close to the entrance to the proposed project access road and is also easily accessible from all other facilities in the area (see Figure 10-1). The proposed building is a two-storey structure with a total floor area of 1,770 m^2, containing 50 office spaces as well as adequate storage area, service facilities, and a conference room (see Figure 10-2).

The ground floor has been arranged to provide offices for senior staff members and service and administrative departments such as accounting, data processing, payroll, personnel, and purchasing. Engineering departments such as mine planning and geology would be located on the upper floor with adequate office space and drafting area.

10.2.2.2 The Maintenance Complex

This structure, of dimensions 189.0 m x 50.0 m and 0.945 ha area, is centrally located in the Mine Services Area, providing easy access to the repair and service shops for the in-pit vehicles, as well as to the other support facilities such as the Rubber Repair Shop, Administration Building, and the Mine Dry (see Figure 10-1). The building layout is shown on Figure 10-3 and consists of the following work and storage areas.

8 Haul truck repair bays	10.5 x 18.5 m
4 Tractor repair bays	10.5 x 18.5 m
8 Auto repair bays	5.25 x 9.25 m
2 Steam-clean bay/wash down bays	10.5 x 15.0 m
Welding and fabricating shop	1,350 m^2
Machine shop	850 m^2
Electrical repair shop	280 m^2
Radio and instrument repair shop	60 m^2
Hydraulic repair shop	200 m^2
Warehouse areas	
- palletized	2,000 m^2
- small piece (shelved)	385 m^2
- flammable goods store	100 m^2
- tool crib	70 m^2

Planning area	300 m^2
Fire Truck Storage Bay	6.5 x 12.0 m
Ambulance Storage Bay	6.5 x 12.0 m
First Aid Centre	78 m^2
Training Centre	150 m^2
Mechanical service rooms	110 m^2
Battery Room	25 m^2

The internal layout of the maintenance complex was designed to the following assumed criteria:

(1) maintenance, planning, and supervisory office areas should be central and have easy access to all parts of the complex;

(2) the vehicle repair bays should be within easy reach of warehouse storage of spare parts and materials; and

(3) the vehicle repair bays should have easy access to ancillary repair and service areas such as the machine shop, welding and fabricating shop, etc.

The number of repair bays was determined by using anticipated mechanical availabilities for the various types of equipment.

All work areas in the complex will be supplied with compressed air, water, power, and oxyacetylene from bulk storage facilities. General service equipment such as welding machines, powered hand tools, welding screens, bench and floor-mounted grinders, and drill presses will also be supplied as required. In areas where fumes and dust will be a problem, such as the welding shop and steam bay, fume hoods and/or extractor fans should be provided.

Each shop area will include a foreman's office in addition to specialized equipment.

The Truck Repair Shop

This will be equipped with a 30 t overhead electric travelling crane fitted with a 5 t auxiliary crane, controlled from floor level, to accommodate work on the 77 t and 154 t mine trucks. Other equipment will include 100 t hydraulic jacks, a lubrication rack complete with hose reels to dispense all necessary fluids and greases in the service bays, and waste-oil disposal tanks from which the waste oil is pumped to a holding tank in the fuel storage and dispensing area (see Figure 10-1). A 10 m-wide concrete apron will be provided adjacent to the entrances to this and the following two shops.

The Tractor Repair Shop

This will be equipped with a 15 t overhead electric travelling crane with floor-level controls. A lubrication rack and a waste-oil disposal system, as described earlier, will be supplied in the service bays.

The Auto Repair Shop

This will be equipped with a 10 t overhead electric crane, plus floor-mounted hydraulic vehicle lifts of 500 kg capacity.

The Welding/Fabricating Shop

This will be equipped with a 25 t crane, as earlier described, as well as 2 t jib-type cranes. The specialized equipment in this shop should include a universal iron worker, plate rolls, profile cutter, air-arcing equipment, blacksmith furnace, and welding booths.

The Machine Shop

This will be equipped with a 12 t overhead electric crane, plus 2 t jib cranes to service various machine tools such as 750, 430, and 275 mm lathes, 150 mm horizontal boring mill, 1,400 mm x 280 mm milling machine, 2,000 mm radial arm drill, 550 mm stroke shaping machine, key-seater, surface plate, 75 t and 300 t hydraulic presses, cleaning tanks, and precision gauges. A strip-down and assembly area will be designated.

The Electrical Repair Shop

This will be equipped with a 12 t overhead crane and other related equipment such as testing equipment, cleaning tanks, and drying oven.

The Warehouse

This will be equipped with 2,250 kg and 4,500 kg capacity forklift trucks in the covered warehouse area to handle the loading, unloading and storing of palletized parts. Handling of larger and heavier components will be by mobile yard cranes as required, either in outside yards or coal storage buildings. Small parts will be stored in bins.

Necessary office space for warehouse personnel and inventory records is included.

10.2.2.3 The Mine Services Building

This building, with its ancillary storage yard, is located in the Eastern section of the Mine Services Area (see Figure 10-4) and covers an area of approximately 1.8 ha. It is easily accessible for "in-pit" and maintenance building requirements, as well as delivery of raw materials and spare parts via the mine access road.

The following services are included in this facility:

Sheet Metal and Pipefitters' Shop	215 m^2
Carpenters' Shop	245 m^2
Painters' Shop	95 m^2
Supervisory office space	85 m^2
Winter and/or night storage for vehicles (will also serve as extra workspace for carpenters and pipefitters when required)	400 m^2
Material Storage Area and tool crib	50 m^2
Personnel areas - punch in/out area, washrooms, locker space, lunchroom, and janitorial room	200 m^2
Open storage yard	1.6 ha
Covered plywood and timber storage	100 m^2

The various shop areas in the building will be equipped with necessary tools and equipment as follows:

The Sheet Metal and Pipefitters' Shop

This will be equipped with a sheet metal shear and former, spot welder, bench grinder, drill press, pipe threader, bandsaw, overhead monorail and hoist, and miscellaneous handtools.

The Carpenters' Shop

This will be equipped with a bench saw, radial arm saw, drill, planer, and necessary handtools.

The Painters' Shop

This will be equipped with portable spray painting equipment.

An outside parking area, equipped with electrical plug-in receptacles, will be provided to accommodate overnight parking of miscellaneous service vehicles.

10.2.2.4 The Field Maintenance Centre

Most of the work carried out by the crews operating from this building will be in the field, and therefore no major equipment need be supplied in the building. Bench-mounted grinders and a pedestal drill press will be installed, along with a welding machine and oxyacetylene equipment.

Service trucks equipped with welding machines, oxyacetylene equipment, compressors, lifting equipment, and handtools will provide the services required for the field work. Line trucks will be available for maintenance of the mine electrical power system.

Any other equipment required for major overhauls or major repairs will be supplied from the maintenance complex or Mine Services Centre.

10.2.2.5 The Rubber Repair Shop

This building houses three separate service and repair shops for tires (225 m^2), conveyor belting (300 m^2), and trailing cables (225 m^2), and is located in the Southern section of the maintenance and service area within easy access of the pit and main conveyor routes (see Figure 10-4).

The Tire Shop

Mine production vehicles requiring tire service will have easy access to a heated concrete apron in front of the tire repair building, where tires will be mounted using portable hydraulic jacks and a 13,500 kg capacity forklift, complete with tire manipulator attachment. Minor repairs only should be done in-shop while major repairs should be carried out off-site by a suitable contractor.

The Conveyor Belting Shop

The Conveyor Belt Repair Shop will be equipped to handle belt splicing and belt repairs, both in the shop and in the field, using portable vulcanizing equipment suitable for use with steel cord or fabric ply belting. The reels of belt will be handled by a mobile crane in the storage area and by an overhead monorail at the powered belt reelers in the shop area. The shop will be equipped with all necessary handtools such as knives and belt cutters. Power supply to this building at 600 v would be used directly for the vulcanizing equipment, while a step-down to 120 v would be necessary for lighting, heating, ventilation, power tools, etc. A cold-storage building will be provided for proper storage of conveyor belt splicing and repair materials.

The Cable Shop

This area will be provided for repairing and testing mine cables. Typical equipment in the shop includes powered reels, repair benches, test panels, overhead cranes, and necessary handtools.

10.2.2.6 Laboratories

The following laboratory facilities will be provided:

- an assay/environmental analytical laboratory in the Mine Services Area (see Figure 10-1);

- an environmental services complex situated on a parcel of presently cultivated land to the South-East of the pit near the confluence of Hat Creek and Medicine Creek (see Figure 10-2).

The Assay/Environmental Analytical Laboratory

Located adjacent to the Administration Building, this facility consists of:

- a general office area, conference room and reception area (300 m^2);

- a "wet" laboratory area to be used by both environmental and assay staff for sample analysis;

- a dry-coal laboratory, used solely for coal sample analysis (150 m^2);

- a core-sample handling area (150 m^2);

- a sample storage area (75 m^2);

- equipment storage (50 m^2).

A central enclosed walkway will separate the wet laboratory and office area from the dirtier, coal-handling facilities; precautions will be taken to prevent dust emissions from the latter area.

Core storage sheds will be provided in the vicinity of the Coal Blending Area, with only sufficient core for analytical testing being transported to the laboratory. A shed will also be provided for logging and splitting cores.

Adjacent parking space will be provided for staff work vehicles and vehicles delivering samples.

The Environmental Services Complex

This group of structures will be built on presently developed agricultural land (proposed as potential nursery area) and consist of:

- a "Lord & Burnam"-type greenhouse (aluminum and glass construction), heated, ventilated, and equipped with necessary lighting (105 m^2);

- a "Quonset"-type greenhouse (50 m^2);

- a service building containing reception area, office and drafting areas, and sample preparation room (150 m^2);

- a reclamation/agricultural machinery shed (300 m^2);

- bulk fertilizer storage (8 m^2).

10.2.2.7 The Lubricant Storage Building

As shown in Figure 10-1, this building is located adjacent to the North-East corner of the maintenance complex to provide good access for both delivery vehicles and mine service vehicles. The building should be heated and insulated and house bulk storage tanks for the various lubricating oils and greases required for the mine mobile equipment. The types and quantities of stored materials are as follows:

Motor oil	30 kL
Hydraulic oil	30 kL
Transmission fluid	30 kL
Gear oil	30 kL
Chassis grease	9,000 kg
Track grease	9,000 kg

The various lubricants will be pumped on demand in heated underground piping to the maintenance complex and then to dispensing racks in service bays in the truck, tractor, and auto repair shops.

The storage tanks of mine lube trucks will be replenished at an external loading station forming part of the building. A portable lube island in the mine complements this facility and will be located conveniently close to the main haulage route for quick servicing of trucks whenever necessary. Less-mobile equipment such as tracked equipment will be serviced on location by the lube truck.

10.2.2.8 The Fuel Storage and Dispensing Area

A fuel tank farm with diesel fuel and gasoline dispensing pumps will be located in the Eastern edge of the Mine Services Area, close to the main mine haulage access road and approximately 100 m from the maintenance complex. Access will be provided around the tank farm for the mine fuel trucks, other service vehicles, pick-up trucks, and the fuel supplier's tank truck. A safety berm will be constructed around the farm area to contain any spillage of fuel.

The tank farm will contain diesel fuel, gasoline, mixed anti-freeze, and waste oil storage tanks of 364, 90, 55, and 45 kL, respectively, these amounts of fuel being equal to approximately one week's predicted consumption rates.

Metered bulk loading and unloading facilities for use by the mine fuel trucks and fuel supplier's trucks, as well as dispensing pumps for the general mine use, will be supplied in this area.

An in-pit fuel facility is also provided for the use of the haulage truck fleet and is located in the same area as the in-pit lube island. Other mobile equipment such as tracked vehicles will be fuelled by the mine fuel trucks. The fuel island, receiving its supplies from the supplier's tank truck as required, will have portable horizontally-mounted diesel fuel tanks of approximately 45 kL total capacity, provided with trays to contain accidental spillage, and will be complete with metered loading and unloading facilities.

10.2.2.9 The Mine Dry

The Mine Dry will be divided into four basic areas dirty lockers, clean lockers, washing area, and mine supervision and marshalling area. Provision will be made for installation of 700 "clean" lockers and 700 "dirty" lockers so as to allow a locker of each type for every person on the mine labour force (see Figure 10-5).

Janitorial space and mechanical equipment rooms will also be provided.

Separate dry areas will be provided for male staff members as well as female work crew members.

The heating and ventilating systems will be designed to accommodate the hot and humid conditions expected in a washroom area. The ventilation system will be designed so that the air is directed through the locker spaces to facilitate drying of clothes, and then into the building. The building will be constructed of non-combustible materials, and provision of a full complement of hydrants and fire extinguishers is recommended, rather than installation of a sprinkler system.

10.2.2.10 Storage Areas

The location and size of various storage facilities in the mine service area have been based on consideration of their expected use, access from the work areas in which the stored materials are to be used, the ease of moving materials within the storage area, the need to monitor incoming and outgoing materials, and the possible need for future expansion.

The main storage areas are shown on Figure 10-1 and are described below.

Yard storage

An area of approximately 3.6 ha will be fenced and manned for materials control. Large parts requiring covered storage will be housed in an unheated building constructed of light metal. The area will be arranged to allow easy access for service vehicles and yard cranes.

Shop area storage

Individual storage areas are recommended for the parts and materials used in the various shop areas. The area recommended (approximately 2.7 ha) is centred mainly around the Mine Services Building and Rubber Repair Shop, where a large area is required for storage of tires, rolls of belting, cable reels, and lumber.

A separate fenced area adjacent to the Tire Repair Shop will be provided to permit the secure storage of small vehicle tires. Various rubber repair materials may be stored in a refrigerated building within this fenced area.

Construction storage area

The construction storage, or "laydown", area will be used initially for the storage of materials and equipment during the construction period, and thereafter for the storage of supplies for the pit and mine service area.

10.2.3 Structural Description of Buildings

Buildings will be constructed of light structural steel frame over reinforced concrete floors and footings. Where necessary, structures will be designed for extra loading imposed by travelling cranes, overhead office space, heating and ventilating units.

External walls will generally be built of insulated metal cladding, as specified and detailed by Toby, Russell, Buckwell and Partners, architectural consultants to B.C. Hydro. Sections of the wall adjacent to vehicle access roads or doors into the building will be constructed of reinforced concrete to doorhead height, so as to minimize damage in case vehicles collide.

Internal dividing walls will be built of concrete blockwork or moveable prefabricated panels.

Roofs will be constructed of insulated metal cladding and will be sufficiently sloped to facilitate proper drainage. The exterior cladding is insulated to conserve heat in Winter and keep the shops cool in Summer. The thickness of insulation is designed to accommodate the large variations in temperature expected in the Hat Creek Valley. As with the wall cladding, the roofing is specified and detailed by the architectural consultant.

Because of their very rough usage in heavy-duty repair shops, shop floors will generally be constructed of reinforced concrete, using only a hard abrasion-resistant aggregate. In particularly hard-worked areas, a chemically-hardened wearing surface will be applied to minimize surface break-up and "dusting". To permit a good workable floor, special consideration will be given to producing a well-compacted, capable, load-bearing sub-grade. As drainage of shop areas is of prime importance, care should be taken in pre-designing all floor openings and trenches for services such as water, electrical cables, and exhaust piping, so as to prevent future disruption of drainage systems.

10.2.4 Services to Buildings

10.2.4.1 Water Supply

Water will be provided to all buildings in the Mine Services Area by a buried pipe reticulation system.

10.2.4.2 Electrical Distribution

Each of the maintenance, office, and service buildings will receive incoming power supply at 6.9 kv transmitted via underground cable. A MVA 6.9 kv to 600/347 v sub-station will be located adjacent to each building. The large-size and high Winter heating load of the maintenance complex will necessitate three 1 MVA sub-stations, one to be located centrally and one at each end of the building. Power from these sub-stations will be distributed within the buildings at 600/347 v with 600 v receptacles provided throughout. Transformers of 600/120 v and panel boards complete with single-phase 120 v circuitry and receptacles will be provided where necessary. The type and number of power outlets will be designated according to the layout and designed use of each building, and an adequate number of spare panel board circuits will be made available to allow for future growth of building power load.

In workshop areas such as the machine shop and welding/fabricating shop, a plug-in three-phase 600/347 v bus duct will be used. This bus duct will be suspended from the ceiling structure of the building and provides a high degree of flexibility in future equipment locations.

10.2.4.3 Heating, Ventilating, and Air Conditioning

Electrical heating units are planned throughout, and air conditioning systems will be provided for the Administration Building, as well as the office areas in the mine dry and maintenance complex. Ventilating systems will be provided throughout all buildings, with special attention given to hazardous areas such as the Paint Shop, Woodworking Shop, and Rubber Repair Shop.

10.2.4.4 Lighting

Lighting in maintenance and service facilities is designed to standards recommended by the Illuminating Engineering Society. Specific recommendations for lighting fixtures and intensity are as follows:

(1) In any inside work area of high-ceiling bays with a ceiling height greater than 4.0 m, 1,000 w high-intensity discharge lamps should be spaced so as to provide even light intensity;

(2) Office areas should be lit with 4-tube, 1.2 m fluorescent luminaires;

(3) Where required, outdoor lighting should be provided by 1,000 w high-intensity discharge lamps. The fixtures should be suspended from buildings themselves, where possible, or mounted on poles or towers;

(4) Emergency lighting fixtures should be provided in stairways and passageways of all buildings. Self-contained twin lamp battery packs will be used;

(5) Lighting in hazardous areas such as the Rubber Repair Shop, Paint Shop, and lubricant storage area should be provided, using suitably enclosed and sealed 3-tube, 1.2 m fluorescent units.

10.2.4.5 Sewage Disposal

Sanitary effluent from the mine service complex will be pre-treated and discharged to a holding pond, and will be seasonally used for dust control in the mine. Allowance has been made for a package wastewater treatment plant capable of handling 140 m^3/d of effluent at 400 ppm BOD_5.

If necessary, the treated effluent could be disinfected prior to being sprayed on roads in the mine area; however, no provision has been made for this at present.

Water from equipment washdown facilities will be discharged to stormwater drains via a grease and grit trap, from where it will flow to sedimentation lagoons for primary treatment prior to discharge. The integration of these facilities into the overall mine drainage and water supply systems is discussed further in Section 6.

10.2.4.6 Fire Protection Systems

A fire protection system will be provided to protect capital investment in buildings and equipment and to minimize the danger of fire causing a major shutdown of coal supply to the generating station.

Permanent automatic systems will be installed in high-risk areas. Where there is "medium" risk, a permanent water supply will be provided for hoses or fire trucks. There will be no permanent installations in low-risk areas; spot fires will be extinguished by fire trucks supplied by water tankers.

10.2.4.6.1 The Mine Services Area

This is a relatively high-risk area within the mine development, and insurance underwriters' standards have been used to define fire protection requirements. Automatic fire detection systems will be installed in all buildings, with a central alarm panel in the proposed fire truck bay.

Automatic sprinkler systems are proposed for all maintenance workshops and service buildings other than the Administration Building and Mine Dry Building, which would be relatively low-risk areas, constructed of non-combustible materials.

All buildings will have a standard complement of fire extinguishers, as well as 40 mm standpipes complete with run-out hoses situated to provide coverage of the building floor plan.

Fire hydrants will be located within the service area, so as to further protect buildings and the open yard storage areas around the perimeter.

The water supply system to the service area will maintain a reserve of 1,000 m^3 of water for firefighting, and will be capable of supplying 95 L/s at 415 kPa at connections to buildings.

Two four-wheel drive fire trucks will be based at the Mine Services Area which will be used as required throughout the entire mine development area. These trucks will carry about 2,000 L of water, fire pumps, hose reels, and ladders.

10.2.4.6.2 Coal Handling Systems

The Coal Conveyor from the pit to the Coal Blending Area and the Overland Conveyor to the generating station will be under the surveillance of an automatic fire detector system which will identify the location of a fire and prevent its spread by stopping the conveyor.

Conveyor transfer stations and enclosed sections of the Overland Conveyor will be protected by automatic sprinkler systems fed from a buried water main adjacent to the conveyorway. Exposed sections of conveyor will be protected by fire trucks which will gain access by a proposed service road, and will draw water from fire hydrants adjacent to the conveyor. In the Coal Blending Area buried water mains between the stockpiles will supply water to fire hydrants. Service roads will be provided between stockpiles to allow access for fire trucks and crews.

10.2.4.6.3 The Open Pit

Within the open pit, water tankers and fire trucks will be used to extinguish spot fires. A permanent water supply will be available from the water main at the North conveyor incline; a hydrant will be provided at each bench. A further allowance has been made for 10,000 m of 75 mm aluminum pipe should an in-pit water supply main be required during mining operations.

10.2.4.6.4 Waste Conveyors

These systems will be surveyed by an automatic fire detection system similar to that for the coal conveyors. Permanent fire protection systems will not be installed on the conveyorways to the waste dumps and water tankers; fire trucks will be used instead.

10.2.4.6.5 Mine Equipment and Vehicles

All mine vehicles will be equipped with portable extinguishers for vehicle safety. Supervisors' and safety officers' vehicles will carry larger units for emergency use on mine equipment.

10.2.4.7 Surface Drainage

Because of the need for ample drainage in the Mine Services Area, a comprehensive system of ditches and culverts is recommended to cope with the estimated 10-year flood. This will be the stormwater collection system.

A 750 mm corrugated steel pipe installed with catch basins will be routed through the centre of the maintenance and service area to receive runoff from the central yard area, as well as the roof drainage from the Administration Building, Laboratory, Mine Dry, and Maintenance Complex.

Storage areas, parking areas, general landscaped areas, and service shops should be drained by means of a network of ditches channelling the runoff to the water treatment area. To facilitate this drainage, general areas should be sloped at 0.5% to perimeter ditches or catch basins, and buildings would be drained via 300 mm buried corrugated pipes to the perimeter ditches or central 750 mm pipe.

All of the runoff from this area is planned to be collected at the West end of the service area, from where it would be carried by culverts and ditches to the sedimentation lagoons. Sewage effluent and leachate from coal-handling plant areas will be collected in a separate drain system and fed to the leachate storage lagoon.

10.2.4.8 Security

It is intended that a 2 m high maximum security barrier will enclose the Mine Services Area, the Coal Blending Area, and the construction storage area. This barrier will be a heavy duty mesh fence topped with a barbed-wire strung overhead section.

Controlled access to this area will be provided at the following four points around its perimeter:

(1) A main entrance from the public access road will lead into the Mine Services Area and the entrance to the pit and blending areas. A guardhouse will be provided at this entrance to maintain a 24-hour full security check of incoming and outgoing traffic. The entrance will be large enough for two opposite flows of traffic, with the guardhouse located centrally between the traffic lanes;

(2) A second entrance from the main access road to the coal-blending yard is planned, but a permanent guardhouse will not be provided. It is intended that this gate be locked and opened as required by selected mine personnel;

(3) A gate will be provided in the vicinity of the main conveyor transfer point and adjacent to the Houth Meadows Waste Conveyor for the use of the conveyor service crews with their vehicles.

(4) A similar gate will be provided adjacent to the Medicine Creek Waste Conveyor.

North of the project area, security fencing will be strung across Hat Creek Valley adjacent to Highway 12 to discourage public entry to the congested area in the valley bottom between the blending area and the Houth Meadows Retaining Embankment. A lockable gate will be provided across the existing Hat Creek road.

Security fencing will also be provided around the larger, fixed electrical sub-stations. Security and safety barriers for other electrical components, moveable or fixed, within the mine area are included in the design and construction of those components. A further run of security fencing would surround the explosives magazine. In total, approximately 7.5 km of security fencing is to be installed.

As the mine will operate on a 24-hour basis, general lighting for the security fence need not be provided, but future illumination of selected areas should be considered by the mine security personnel.

It is proposed to enclose the entire mine project area within a low-security barrier to keep out livestock, as well as to provide a visual deterrent to the public. This barrier would be a three-strand barbed-wire fence strung between 1.5 m high metal posts at 4 m intervals. At the commencement of mining operations, 24 km of this type of fencing should be erected around the 35-year limits of the pit and the Houth Meadows Waste Dump, and up both sides of the main coal conveyor. In about Year 16, a further 14.5 km of ranch fencing should be erected around the 35-year Medicine Creek Waste Dump and Waste Conveyor. A fenced corridor should be provided through the project area for the diverted access road to the Upper Hat Creek area. Notices should be posted at suitable intervals around this fenceline, warning people against trespass for the sake of their own safety.

10.2.5 Construction Period Requirements

Temporary facilities for personnel, materials, and equipment will be required during construction of the permanent mine support facilities. These temporary facilities are outlined below.

10.2.5.1 Construction Schedule

According to this schedule (Section 12), all mine support facilities will be ready for use when required for the general mining operations. The schedule takes into account engineering and purchasing, anticipated delivery periods, and assembly of equipment and materials.

10.2.5.2 Construction Period Facilities

Construction period facilities would include a temporary camp to house the construction work force; the camp does not form part of this study and is not discussed further. It is assumed that the supply of the necessary construction buildings, shops, etc., would be the responsibility of the respective contractors during construction of the various structures and erection of the initial mining and conveying equipment. It is assumed that temporary office and warehousing facilities would be erected to house the construction management team and B.C. Hydro's construction and management group during the construction period.

10.2.5.3 Mine Equipment Erection Area

A mine equipment erection area of 12 ha is provided at the Northern end of the proposed mine perimeter, close to the conveyor ramp, the Mine Services Area, and serviced by access roads for the movement of materials and equipment. The area is large enough for the simultaneous erection of two mining shovels, and has sufficient storage space for miscellaneous components of other equipment awaiting erection, as well as for various small buildings, offices, and other facilities.

The area will be well drained and supplied with water and power.

10.3 UTILITIES

10.3.1 Water Supply

10.3.1.1 Introduction

This section of the report presents the estimated water supply requirements for the mine development and describes the layout of the proposed supply and distribution systems. Consideration was given to the integration of the mine supply with parallel systems proposed for the thermal generating station and the construction camps.

10.3.1.2 Water Requirements

The mine and adjacent service complex will require a permanent water supply during the years of operation of the mine. The mine water supply system must provide for the estimated demand from the following areas and services:

Potable Water for the Mine Services Area	- Administration Building - Mine Dry - laboratories - maintenance buildings
Fire Protection Systems	- buildings - in-pit coal conveyor - overland coal conveyor - Coal Blending Area
Washdown Water	- vehicles and equipment
Irrigation Waters	- mine service area lawns and landscaping - revegetation nursery
Dust Control	- roads - Coal Blending Area - low-grade coal stockpile
Temporary Construction Supply	- during construction of the Mine Services Area

A preliminary estimate of the water requirements of the mine at full development is shown on Table 10-1. Fire requirements are not shown as they do not form a day-by-day consumptive demand. Fire protection for buildings in the Mine Services Area would require the provision of 1,000 m^3 of reservoir capacity and a flow of 95 L/s at a residual pressure of 415 kPa at risers (M & M Consultants 1978). At the Coal Blending Area allowance has been made for a flow of 30 L/s at 415 kPa for fire control at the stockpiles.

Water quality standards for potable and irrigation water supply are presented by Beak (1978). These criteria were utilized in the selection of the source for these systems. The quality of dust control water is, however, not critical, and the supply requirements may be satisfied by recycling wastewater from the mine operation.

10.3.1.3 Water Sources

Four alternative sources of supply were considered and are discussed in detail in the CMJV Feasibility Report Vol. IV.

- Water from the proposed generating station supply which will be taken from the Thompson River;
- Surface water from the proposed Hat Creek Diversion Canal located to the East of the mine;
- Groundwater from a well sunk to the North-East of the Mine Services Area;
- Recycled mine wastewater.

The major factors in the selection of a suitable source are water availability, cost of treatment, and supply. The locations of ancillary plant and mine facilities were considered as well, in order to provide an integrated system and to avoid duplication.

10.3.1.4 Proposed Water System

10.3.1.4.1 Source of Supply

Water for the mine development will be supplied from the following sources:

Potable Water and Fire Protection

Water for these uses will come initially from an integrated groundwater supply system with the project temporary construction supply. When the permanent generating station supply from the Thompson River comes on line, the construction supply pipeline from Hat Creek Valley to the powerplant will become the main supply line for potable water in the Mine Services Area. This water will require minimum treatment prior to use. Fire protection requirements will also be supplied by this line. A backup supply could also be provided by the use of treated surface water from Hat Creek.

Irrigation

Water for irrigation will be supplied from the Pit Rim Reservoir to the South of the open pit.

Dust Control

These requirements would be satisfied by recycling wastewater and pit dewatering flow.

The proposed water supply system is shown on the mine drainage and water supply flow chart, Figure 6-3 in Section 6.

10.3.1.4.2 Mine Services Area

The Mine Services Area would be fed from a reservoir located near the proposed construction camp. A booster pumphouse at the Eastern perimeter would increase line pressure to 700 kPa using two electric pumps of total capacity 100 L/s. A gasoline-driven fire pump of 100 L/s capacity would also be installed to provide water for fire control in the case of power failure. A water main approximately 1,700 m x 200 mm would provide a water supply for the Mine Services Area, provide potable water to buildings, supply fixed sprinkler fire protection systems in buildings, and fire hydrants in open yard storage.

10.3.1.4.3 Coal Blending Area

Provision has been made for a permanent dust control and fire protection system at the Coal Blending Area. Buried mains approximately 2,100 m in length beneath the stacker corridors would supply water to fire hydrants located at 100 m spacing between the coal piles. Dust control would be provided by oscillating water jets which could be mounted between the stacker rails and reclaimer units. During Summer the dust control system would be fed from the leachate holding pond as part of an evaporative disposal system for leachate. Make-up water to the system and fire reserve would be provided by the open pit fire protection supply.

10.3.1.4.4 Overland Coal Conveyor

Water for fire protection of conveyor belts and transfer stations will be supplied from a buried main following the conveyor. A service road along the conveyor will allow access by fire truck to hydrants along the conveyor route. Frost protection may be required to prevent freeze-up of this pipeline during Winter. The relatively high head loss in the pipe (i.e., 450 m over its 4 km length) may require water hammer protection and pressure-reducing valves at lower points of supply.

10.3.1.4.5 In-Pit Water Supply

Allowance has been made for the construction of a 150 mm water main on the main conveyor incline to provide a supply of water for fire protection of the conveyor and loading stations. Further provision has been made for 10,000 m of 75 mm aluminum in-pit distribution main which would be available for temporary use. The primary source of water for dust control on pit roads would stem from the reclaim water system, which recycles wastewater from the leachate holding pond. Make-up water for this system would come from pit dewatering flows or the permanent fire supply, as required.

10.3.1.4.6 Revegetation Nursery

The proposed Revegetation Nursery to the South of the mine would be remote from the Mine Services Area and would therefore have a separate water supply.

Water would be taken from the pipeline which interconnects the Pit Rim Reservoir with the Hat Creek Diversion Canal and supplied to irrigation spray systems.

Potable water requirements at the adjacent reclamation laboratory would be provided by a small package water treatment unit.

10.3.2 Mine Power Supply

10.3.2.1 Introduction

This section of the report describes the electrical power distribution system used on the basis of estimating the cost of supplying power to the pit, waste dumps, and support facilities. The network developed includes all electrical equipment required to supply power from the 60 kV busbars of the proposed Hat Creek generating station to the open pit and dump areas, and to distribute the power within these areas to the shovels, conveyors, spreaders, and to the crushing and blending equipment. The developed network also includes supply for the various service buildings and provides the construction power required during the development phase of the mine.

10.3.2.2 Electrical Loads

10.3.2.2.1 Power Shovels

The load cycle for a large electrically-powered hydraulic shovel consists of a large surge of power required to hoist and swing the load to a small load during the dump portion of the cycle.

The acceptability of such a load, or the need to compensate for the power swings generated, can only be resolved through a comprehensive study of the utility network. However, the proximity of the proposed generating station to the Hat Creek Mine and the choice of shovels equipped with rectifier inverters should preclude the need for any additional compensating equipment.

The use of rectifier inverter-equipped shovels will result in lower energy consumption due to a reduction in machine losses from approximately 10% for the normal Ward Leonard system to approximately 3% for the inverter system. In addition, shovel availability will be increased due to the reduced maintenance required by the solid-state equipment relative to rotating machinery. The reduced losses and maintenance level of the rectifier inverter shovels is expected to more than offset any additional capital costs involved.

10.3.2.2.2 Voltage Regulation

The start-up of large motors results in the imposition of a large reactive power demand upon the transmission network and, accordingly, a significant drop in voltage at the terminals of the motors. These voltage drops can be compensated for by the use of appropriate equipment comprising either synchronous or static compensators. At Hat Creek, the use of static capacitors on individual motor loads would be restricted to the pre-production years, when limited construction power is available for operation of the pit and the Houth Meadows Waste Dump.

No voltage regulation problems are foreseen at the time that the main 60 kV lines are installed to the powerhouse.

10.3.2.2.3 Estimated Annual Power Demands and Energy Consumption

Table 10-3 shows the estimated mine loads for peak consumption during Years 22 to 25, inclusive. The three types of load specified on the table are defined as follows:

Connected Load

This is the sum of all equipment and motor loads installed in the plant or site.

Typical Load

This is the load most likely to be exerted upon the power supply system during normal production periods.

Annual Average Load

Although this load never actually applied to the power system, this is an estimate of the annual energy consumption presented as a continuous load.

The estimated annual power demands and energy consumption for Years 2 to 35, inclusive, are listed on Table 10-4.

10.3.2.3 Network Design Criteria

The proposed electrical system is shown in Figure 10-7 and has been developed on the basis of the following criteria:

(1) Two mine feeders will be available from B.C. Hydro at the Hat Creek Generating Station's 60 kV busbar; each of these feeders are capable of supplying the total mining load, make-up water for the generating station, and ash disposal pumping requirements;

(2) Construction power at 60 kV will be available from the existing powerline at the junction of Highway 12 and the present Hat Creek road;

(3) Development of the major supply network to the open pit and to Houth Meadows and Medicine Creek Waste Dumps will take place over a period of 14 years. Primary distribution network throughout the complex will be rated at 60 kV, while supplies to shovels, conveyors, etc., will be rated at 6.9 kV. Auxiliary mine equipment, pumps, lighting, and air compressors, together with internal systems for auxiliary buildings, will be 600/347 v, while other building voltages will be 120 v single-phase, where desirable. Transformers will be selected to be interchangeable within standard ratings;

(4) The rating of the apparatus, transformer impedances, line and cable conductor, etc., will be sized such that the voltage fluctuation at the terminals of a shovel will not exceed +10%, -5% under the worst operating conditions. Voltage regulation of the 60 kV power supply from the Hat Creek Generating Station is not expected to exceed ±5%;

(5) The electrical power network for the mine has been designed based on the overall economic optimization of both the remote power loads from the generating station and the remote mining loads, under the control of a single operating entity;

(6) Power distribution within the pit and waste dump areas will be by portable sub-stations and power cables. The portable power cables will be generally TYPE SHD-GC, three-conductor with two ground and one ground check conductor terminated at their extremities with plug and socket-type couplers. The portable sub-stations will be of one common design and will consist of the following elements:

- A portable high voltage (60 kV) switching unit;

- A skid-mounted 4 MVA 60 kV/6.9 kV 3 PH., 60 Hz oil-filled transformer;

- A portable 6.9 kV totally enclosed switching unit complete with three feeder circuits and female cable couplers;

- A portable low-voltage 600/347 v switching and distribution unit complete with 600 v female cable couplers, as well as one 1 MVA 6.9 kV/600/347 v 3 PH., 60 Hz oil-filled transformer;

(Figure 10-8 shows a typical in-pit arrangement of portable sub-stations and cables.)

(7) Fixed loads at permanent locations will be supplied from the 60 kV powerlines via permanent sub-stations similar to the one shown in Figure 10-9; this drawing depicts a typical permanent sub-station which is designed to tap on to a 60 kV transmission line, and which can be equipped with either one or two step-down transformers and 6.9 kV switching units.

10.3.2.4 System Description

10.3.2.4.1 General

In assessing the sizing of lines, transformers, etc., recognition has been made of diversity in operation of the various items of plant. Transformer ratings have been established from consideration of both the thermal loading and voltage drop during start-up of the motor loads, and the standardization of the MVA ratings.

Utilization voltages are based on economic optimization, technical desirability, and standardization of all portable sub-stations for the highest degree of interchangeability and reliability. For the portable sub-stations, 6.9 kV was selected, as this is the preferred shovel voltage.

The initial supply network to the mine up to and including Year 13, will consist of two 60 kV lines from the sub-station at the generating station. The first parallels the coal conveyor and the Houth Meadows Waste Material Conveyor, and is routed so as to encompass the West side of the pit. The second 60 kV line runs from the vicinity of the ash disposal pond adjacent to the extreme Northern boundary of the Medicine Creek Waste Dump, then parallels the Medicine Creek Waste Material Conveyor right-of-way so as to encompass the East side of the pit.

10.3.2.4.2 Pit Area

The in-pit supply network will consist of eight portable skid-mounted 60 kV/6.9 kV sub-stations, i.e., one per shovel plus one spare. The spare unit will be used to prepare for shovel relocations. Eight low-voltage portable skid-mounted 6.9 kV/600 v sub-stations, one per shovel plus one spare, will also be provided. All in-pit cabling will be via 6.9 kV or 600 v trailing cables, depending on the equipment being served.

10.3.2.4.3 Houth Meadows Waste Dump

The Houth Meadows Supply Network will initially consist of one 8 MVA permanent 60 kV/6.9 kV sub-station to feed the overland waste conveyors. In Year 4, this sub-station will be extended to 2 x 8 MVA to accommodate the increased loading of these conveyors. With the opening of the Medicine Creek Waste Dump in Year 16, the sub-station can be reduced to 1 x 8 MVA.

Power supply to the transfer conveyors and spreaders will be provided by portable skid-mounted 60 kV/6.9 kV sub-stations supplemented by portable skid-mounted 6.9 kV/600 v sub-stations where low-voltage 600 v loads are present. The number of complete portable sub-stations varies according to the following schedule:

Years -2 to 8	3 units
Years 8 to 14	4 units
Years 14 to 21	5 units
Years 21 to 35	4 units

All distribution within the dump will be via 6.9 kV or 600 v trailing cables, depending on the equipment being served.

10.3.2.4.4 Medicine Creek Waste Dump

The Medicine Creek Supply Network will comprise two overland conveyor sub-stations, each consisting of one 8 MVA transformer rated at 60 kV/6.9 kV. A maximum of two complete portable skid-mounted 60 kV/6.9 kV. A maximum of two complete, portable skid-mounted 60 kV/6.9 kV sub-stations, each supplemented by a portable skid-mounted 6.9 kV/600 v sub-station, will be required and will be installed in Year 15. The schedule of complete portable sub-stations is as follows:

Years 15 to 17	1 unit
Year 18	2 units
Years 18 to 24	1 unit
Years 24 to 27	2 units
Years 27 to 35	1 unit

All distribution within the dump will be via 6.9 kV or 600 v trailing cables, depending on the equipment being served.

10.3.2.4.5 Mine Service Facilities

All service and office building areas are planned to be supplied by underground 6.9 kV cables from the Truck Unloading Station No. 1 permanent sub-station. Supply provisions will be made for the following facilities:

- Maintenance complex;
- Administration Building;
- Mine Dry;
- Rubber repair shops;
- Mine Services Building.

Other minor loads, such as those for the laboratory and gate house, will be supplied from an adjacent service or office building by 600 v underground cables.

10.3.2.4.6 Crushing/Blending Plant

A 2 x 8 MVA sub-station will supply a common 6.9 kV service to the crusher building, blending area, and Truck Unloading Station No. 1, to be built in Year 1. This sub-station will supply most of the power requirements of the coal-crushing and blending plant. A separate 16 MVA sub-station would be required to supply a coal-washing plant, should one be installed.

10.3.2.4.7 Reliability

The 60 kV transmission lines have been physically located to provide maximum reliability for all the major electric loads. It will be possible to operate the network as a ring main system for the pit and part of the Medicine Creek Waste Dump. However, the Houth Meadows Waste Dump 60 kV network will operate as a radial feeder.

Should failure of an 8 MVA 60 kV transformer occur, requiring several months for repair, it is intended that one of the 8 MVA transformers installed at the Truck Unloading Station No. 1 or No. 2 would be used. Since these two stations are not expected to be simultaneously operating at maximum capacity due to the yearly change in mining activity, one transformer should always be available for use as a temporary replacement elsewhere.

10.3.2.4.8 Construction Power

The erection of special construction powerlines is not required, with the exception of:

(1) A short 60 kV line connecting the most Easterly leg of the Houth Meadows Waste Dump line to the existing 60 kV circuit at the junction of Highway 12 and the Hat Creek road; and

(2) A short 60 kV line connecting Truck Unloading Station No. 1 to the line supplying the crusher sub-station.

By installation early in the construction period of these two short temporary 60 kV lines, about 2 km of 60 kV permanent transmission lines, and Hopper Station No. 1 sub-station, an adequate power supply at 6.9 kV can be realized. In general, the construction power supply will be provided by the early installation of the permanent electrical supply equipment.

TABLE 10-1

Estimated Mine Water Requirements

	Daily Average (1) m^3/day	Supply Source Construction	Supply Source Operation
Potable			
Mine Dry	140	Wells	Powerplant
Service Buildings and Laboratories	90	Wells	Powerplant
Revegetation Nursery	5	Wells	Powerplant
Total Potable	235	Wells	Powerplant
Irrigation			
Mine Services Area	120	--	Pit Rim Reservoir
Nursery	500		
Total Irrigation	620 (2)	--	Pit Rim Reservoir
Dust Control	2,000	--	Leachate Storage Lagoon
TOTAL REQUIREMENTS	2,855		
Fire Protection (Storage)	1,000 m^3	Wells	Powerplant

Notes: (1) based on Year 35
(2) Summer use only

Source: CMJV Vol. IV Section 4

TABLE 10-2

Water Quality Data
Hat Creek Project Mining Feasibility Report 1978

Water Source	Hat Creek Surface Water	Thompson River Surface Water	Hat Creek Valley Groundwater
PARAMETER (mg/L)	System Mean	System Mean	One Sample Well V78-72
CATIONS - Trace Metals			
Aluminum (Al)	< 0.010	< 0.017	-
Arsenic (As)	< 0.005	< 0.005	-
Cadmium (Cd)	< 0.005	< 0.005	-
Chromium (Cr)	< 0.010	< 0.010	-
Copper (Cu)	< 0.005	< 0.005	< 0.03
Iron (Fe)	< 0.026	< 0.022	-
Lead (Pb)	< 0.010	< 0.010	-
Mercury (Hg)	< 0.00040	< 0.00034	-
Molybdenum (Mo)	< 0.020	< 0.020	-
Selenium (Se)	< 0.003	< 0.003	-
Vanadium (V)	< 0.005	< 0.005	-
Zinc (Zn)	< 0.007	0.017	-
Manganese (Mn)	-	-	0.05
CATIONS - Alkali Earths & Metals			
Calcium (Ca)	57	11	59.8
Lithium (Li)	0.002	< 0.001	-
Magnesium (Mg)	19	2.3	59.1
Potassium (K)	4.0	0.63	24.5
Sodium (Na)	20/23	3.3	220
Strontium (Sr)	0.32	0.055	-
ANIONS - General			
Boron (B)	0.10	< 0.10	-
Chloride (Cl)	1.1	1.6	8
Fluoride (F)	0.16	0.11	-
Sulfate (SO_4)	54	7.6	392
ANIONS - Nutrients			
Total Kjeldahl Nitrogen (N)	0.19	0.08	-
Nitrate Nitrogen (NO_3-N)	< 0.05	< 0.07	-
Nitrite Nitrogen (NO_2-N)	< 0.002	< 0.002	-
Total Orthophosphate Phosphorus (P)	0.043	0.020	-
ORGANIC, NONIONIC & CALCULATED VALUES			
COD	21	21	-
TOC	9	3	-
Phenol	< 0.002	< 0.002	-
Total Hardness ($CaCO_3$)	224	38	392
Total Alkalinity ($CaCO_3$)	226	35	760
PHYSICAL DATA			
pH (units)	8.4	7.8	8.2
Specific Conductance (umhos/cm @ 25^o)	489	93	1470
True Color (Pt-Co Units)	12	9	-
Turbidity (NTU)	1.5	0.81	-
Temperature (^oC)	6.6	8.0	-
PHYSICAL DATA - Residues			
Total Residue	348	77	-
Filterable Residue	342	74	1230
Nonfilterable Residue	6	3	-
Fixed Total Residue	281	50	-
Fixed Filterable Residue	278	49	-
Fixed Nonfilterable Residue	4	2	-
BIOCHEMICAL, DISSOLVED GASES & RELATED MEASUREMENTS			
BOD	< 1	< 1	-
D.O.	11.1	11.1	-
Data Sources:	Beak, 1978	Beak, 1978	H.A. Simons, 1978

TABLE 10-3

Total Estimated Mine Load During Peak Years
Hat Creek Project Mining Feasibility Report 1978

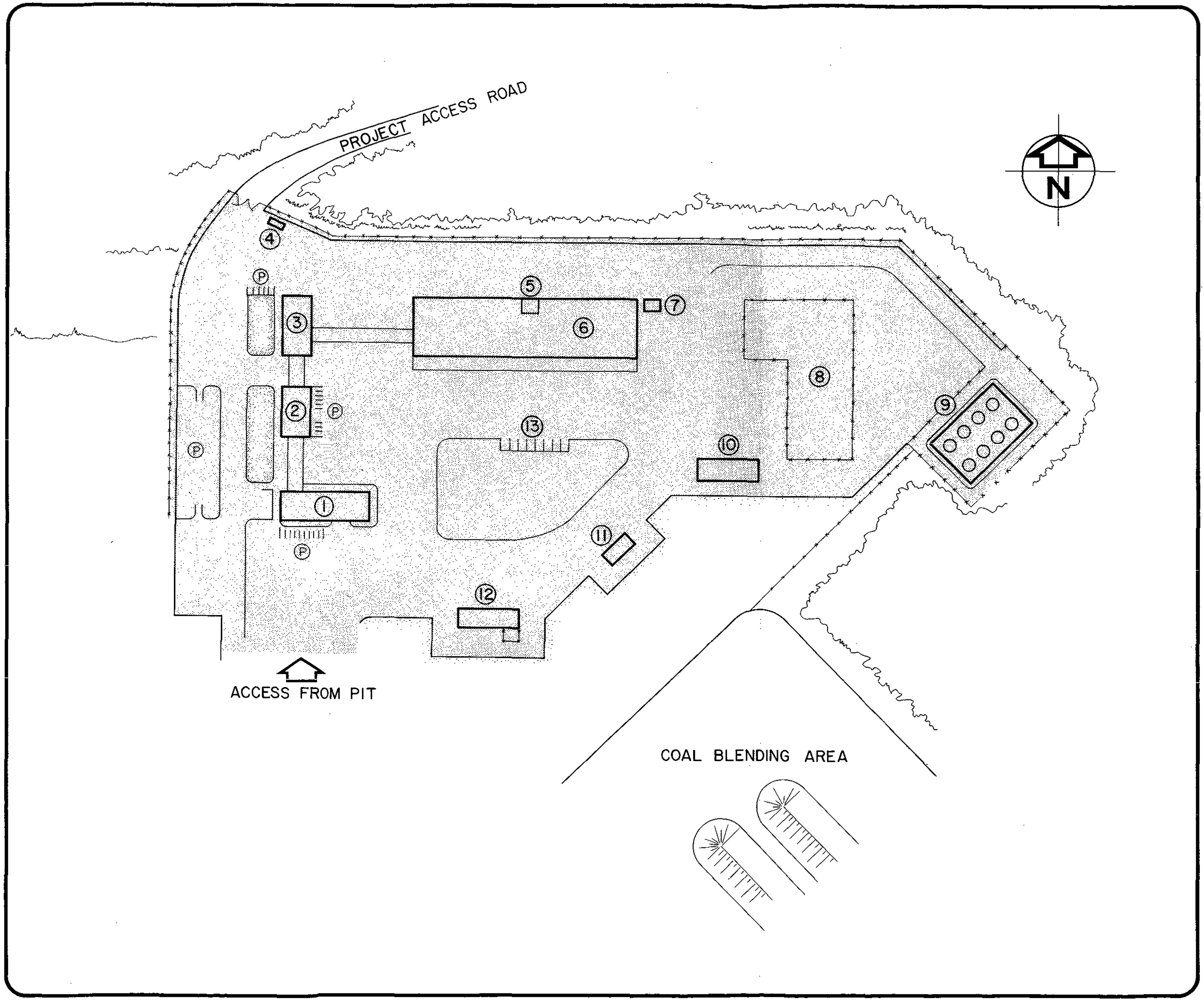
Requirement	Connected Load (kW)	Typical Load (kW)	Annual Average Load (kW)
Out-of-Pit Loads			
Conveyor Load	53,500	24,382	15,600
Maintenance Complex	2,760	1,681	1,320
Mine Dry Complex	841	427	363
Rubber Repair Building	359	165	125
Mine Service Building	805	383	278
Administration Building	644	349	178
	58,909	27,387	21,618
In-Pit Loads			
9 Shovel Sub-stations (1 spare)	13,500	8,142	2,714
Pumping and Miscellaneous	455	180	126
	13,955	8,322	2,840
Total	72,864	35,709	24,458

TABLE 10-4

Estimated Annual Load and Energy Demands

Hat Creek Project Mining Feasibility Report 1978

Year	Average Annual Load (kW)	Typical Load (kW)	Annual Energy (MW hours)
-2	2,625	3,833	22,988
-1	13,799	20,117	120,702
1	18,836	27,501	165,006
2	19,971	29,158	174,948
3	20,999	30,659	183,954
4	22,448	32,774	196,644
5	22,089	32,250	193,500
6	23,853	34,825	208,950
7	23,853	34,825	208,950
8	23,853	34,825	208,950
9	23,853	34,825	208,950
10	23,853	34,825	208,950
11	21,313	31,117	186,702
12	22,940	33,492	200,952
13	22,940	33,492	200,952
14	23,933	34,942	209,652
15	23,933	34,942	209,652
16	24,458	35,709	208,254
17	24,458	35,709	208,254
18	24,458	35,709	208,254
19	24,458	35,709	208,254
20	24,458	35,709	208,254
21	23,647	34,525	634,525
22	23,647	34,525	634,525
23	23,647	34,525	634,525
24	23,647	34,525	634,525
25	23,647	34,525	634,525
26	21,626	31,574	189,446
27	20,240	29,550	177,300
28	19,829	28,950	173,700
29	19,829	28,950	173,700
30	19,829	28,950	173,700
31	19,829	28,950	173,700
32	19,829	28,950	173,700
33	19,829	28,950	173,700
34	19,829	28,950	173,700
35	19,829	28,950	173,700



LEGEND

1. MINE DRY
2. LABORATORY
3. ADMINISTRATION BUILDING
4. GATEHOUSE
5. FIRE TRUCK/AMBULANCE GARAGE
6. MAINTENANCE COMPLEX & WAREHOUSE
7. LUBE STORAGE BUILDING
8. FENCED STORAGE AREA
9. FUEL STORAGE & DISPENSING AREA
10. MINE SERVICES BUILDING
11. FIELD MAINTENANCE CENTRE
12. RUBBER REPAIR SHOP
13. TRUCK READY LINE

(P) PARKING

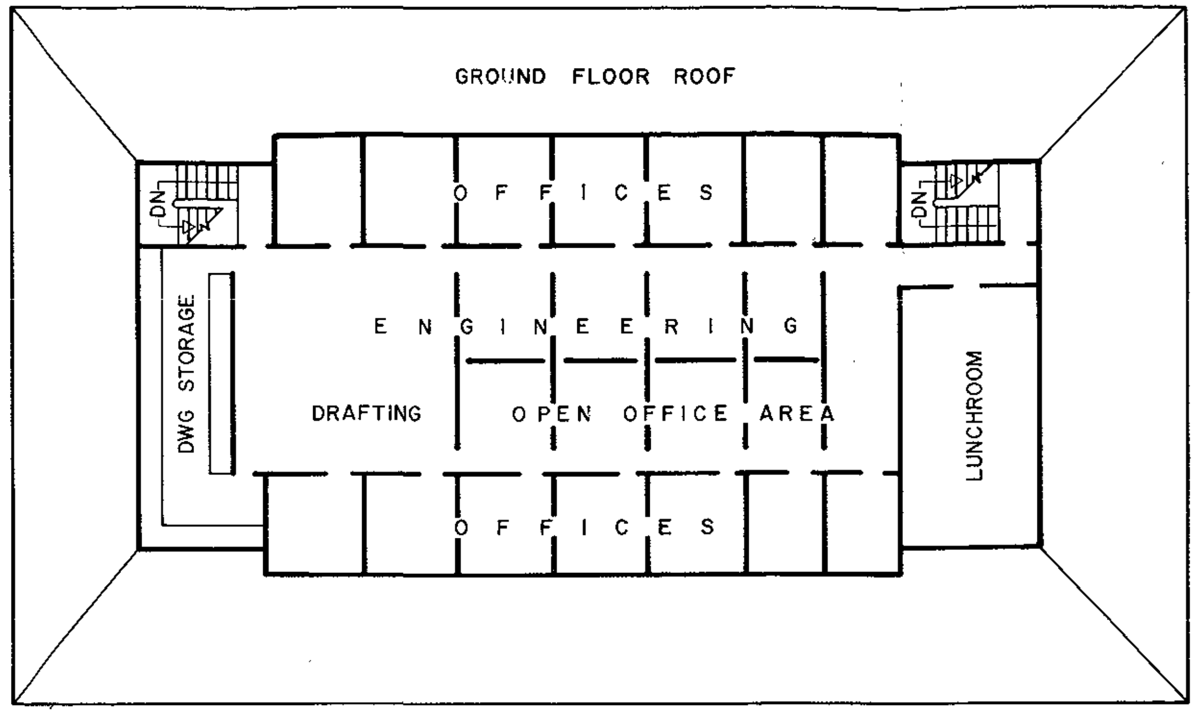
50m 0 100m 200m

HAT CREEK PROJECT

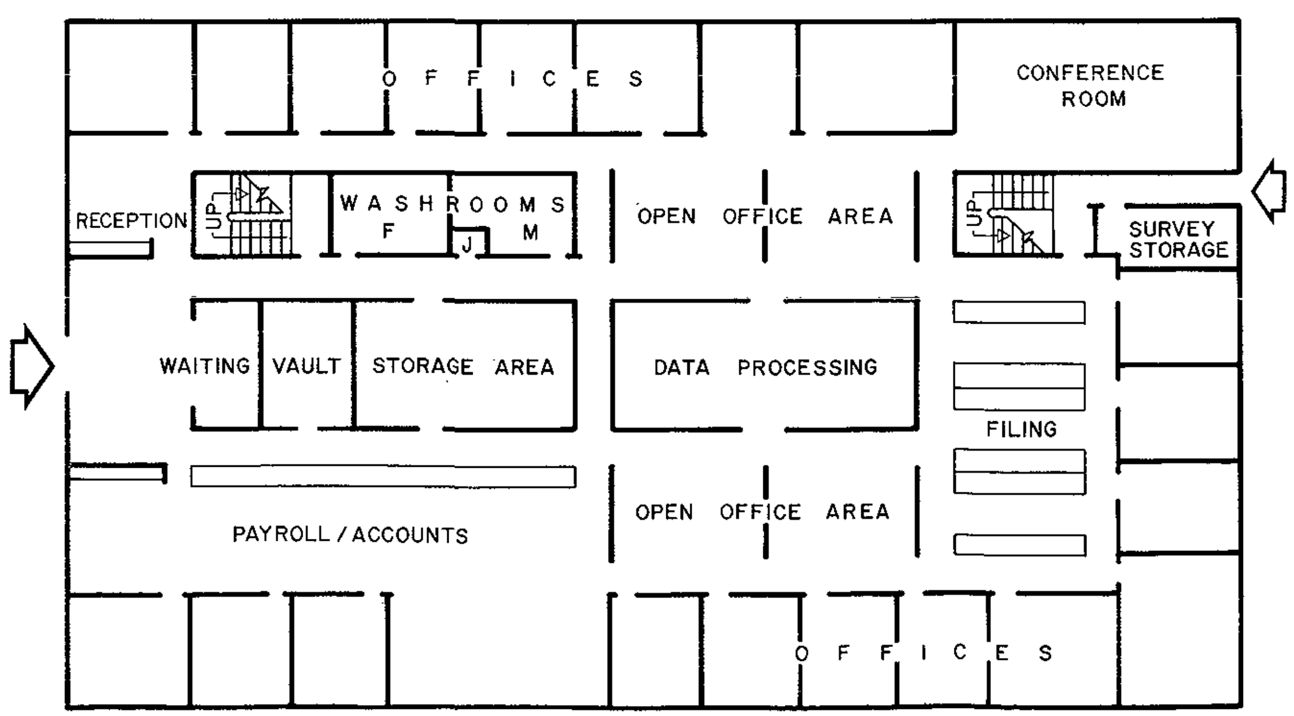
FIGURE 10-1

GENERAL ARRANGEMENT MINE SERVICES AREA

SOURCE: British Columbia Hydro and Power Authority



SECOND FLOOR PLAN



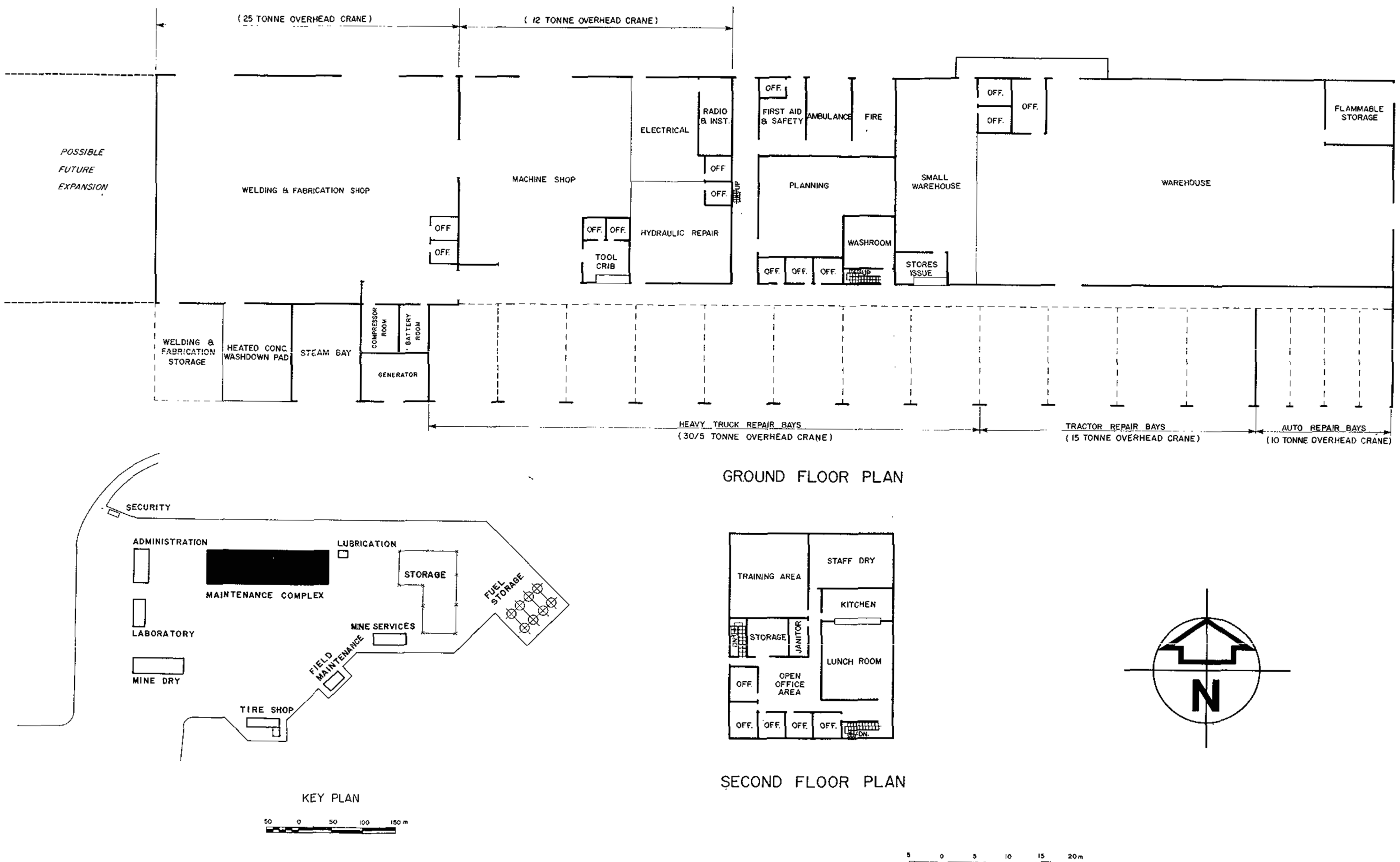
GROUND FLOOR PLAN

HAT CREEK PROJECT

FIGURE 10-2

ADMINISTRATION BUILDING

SOURCE: Cominco-Monenco Joint Venture

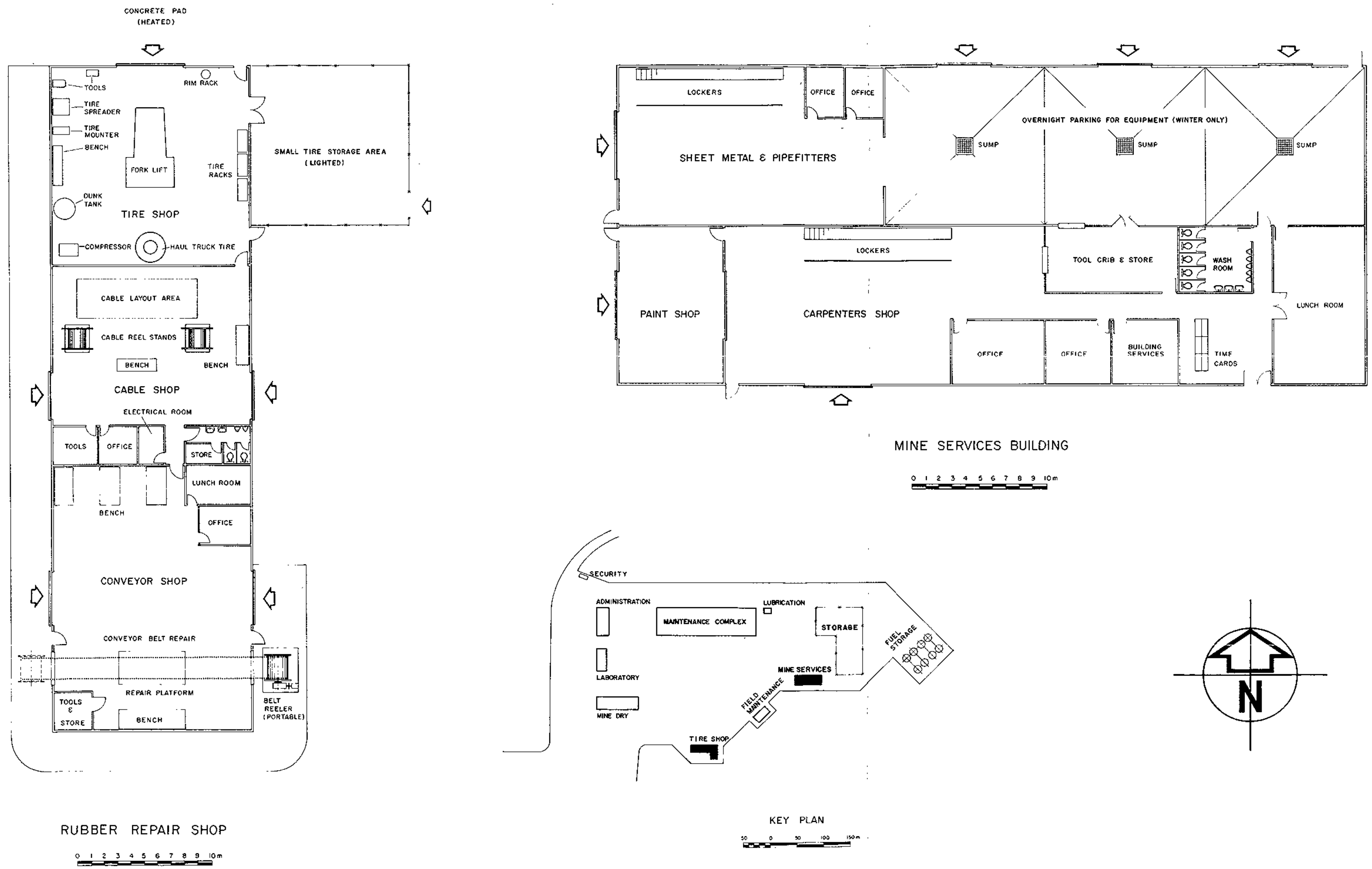


HAT CREEK PROJECT

FIGURE 10-3

MAINTENANCE COMPLEX

SOURCE: Cominco-Monenco Joint Venture

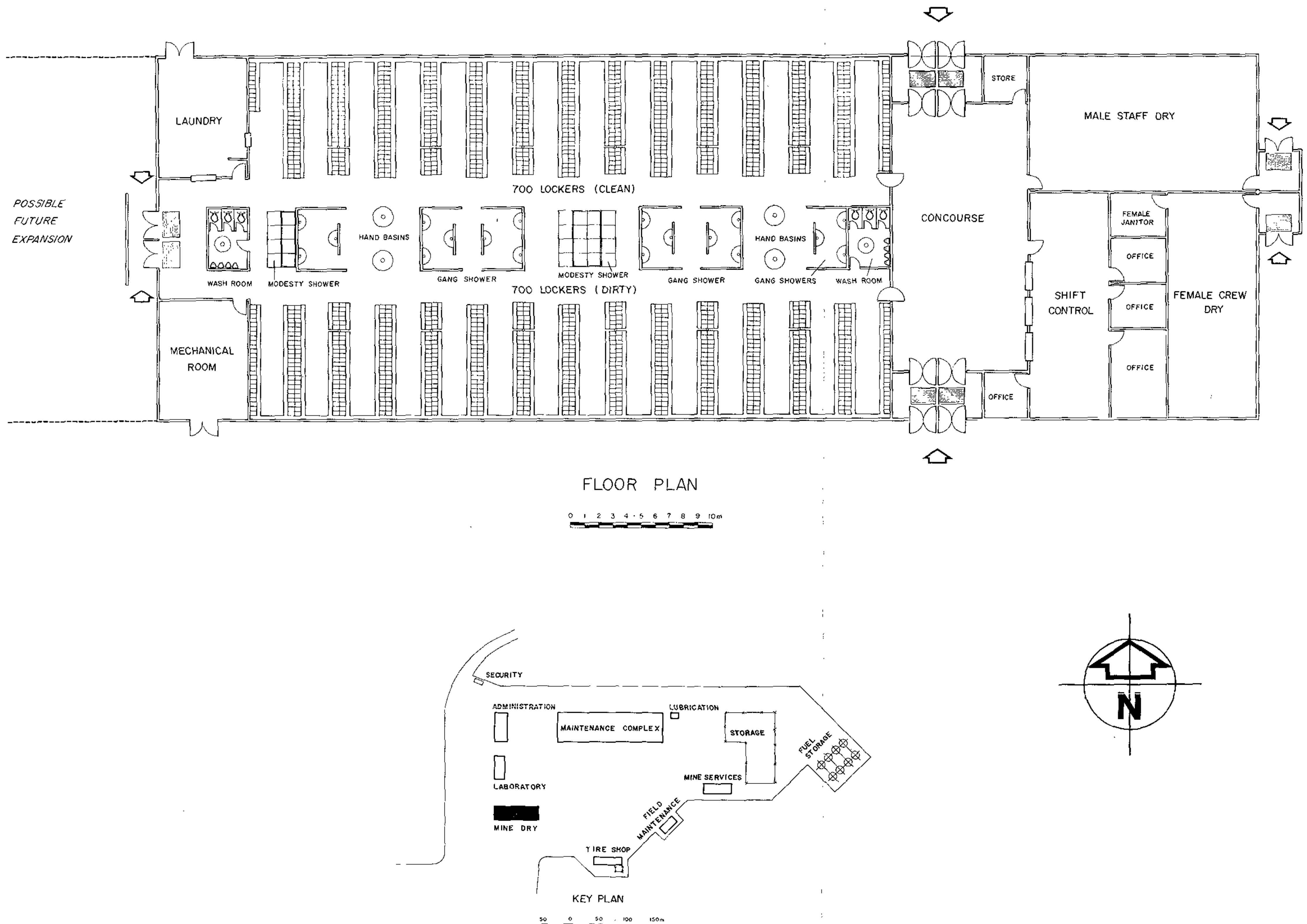


HAT CREEK PROJECT

FIGURE 10-4

Mine Services Building and Rubber Repair Shop

SOURCE: Cominco-Monenco Joint Venture

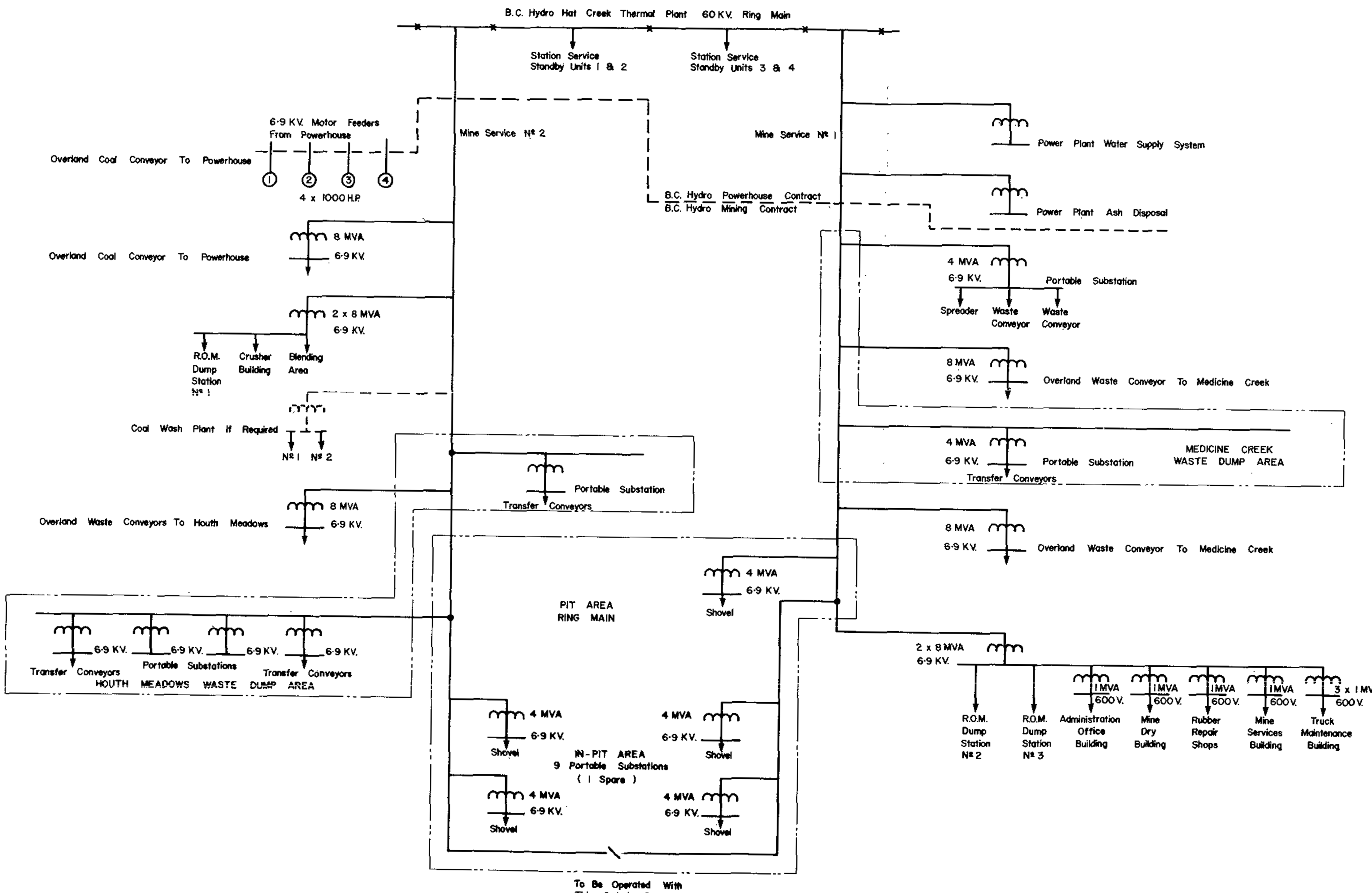


HAT CREEK PROJECT

FIGURE 10-5

MINE DRY

SOURCE: Cominco-Monenco Joint Venture

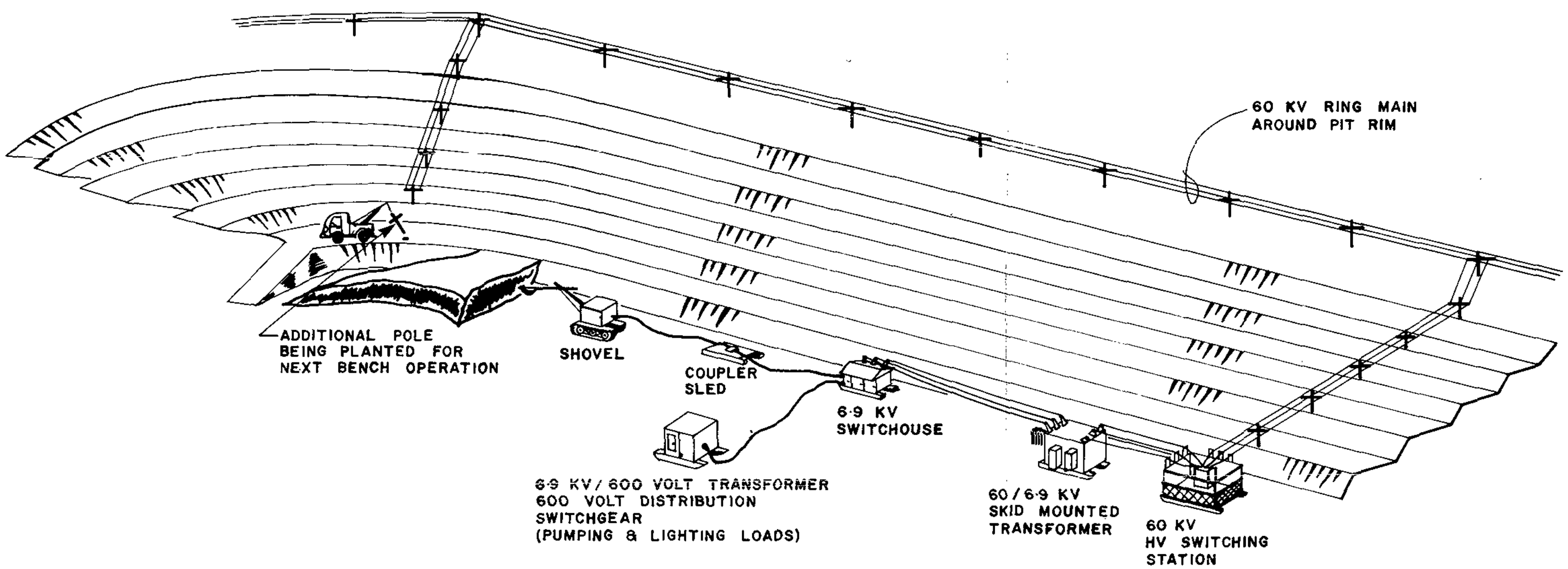


HAT CREEK PROJECT

FIGURE 10-6

MINE POWER DISTRIBUTION NETWORK

SOURCE: Cominco-Monenco Joint Venture

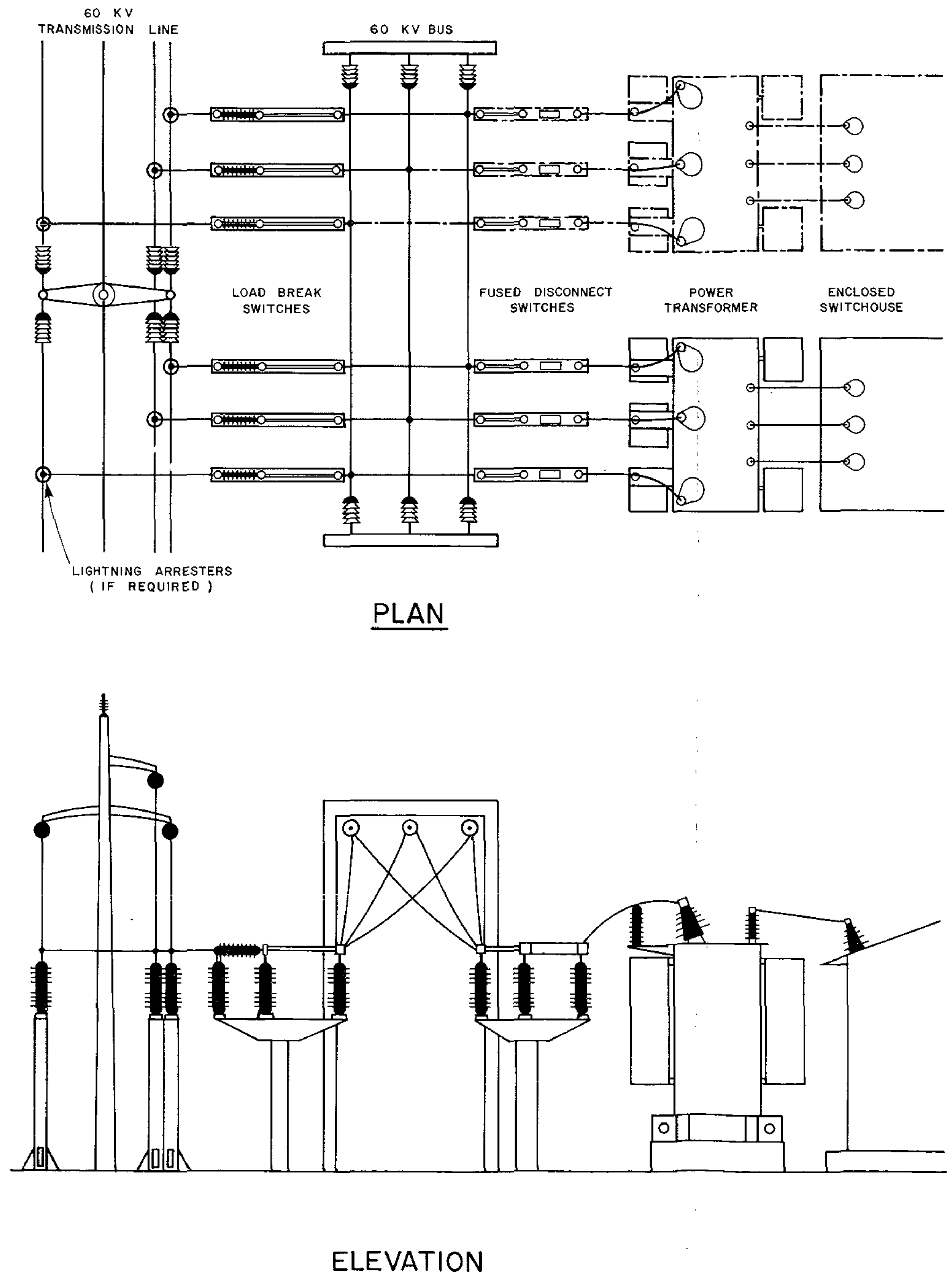


HAT CREEK PROJECT

FIGURE 10-7

TYPICAL
PORTABLE SUBSTATION LAYOUT

SOURCE: Cominco-Monenco Joint Venture



HAT CREEK PROJECT

FIGURE 10-8

TYPICAL PERMANENT SUBSTATION LAYOUT

SOURCE: Cominco-Monenco Joint Venture

11 ENVIRONMENTAL PROTECTION

Page

LIST OF TABLES

LIST OF FIGURES

SECTION 11

ENVIRONMENTAL PROTECTION

11.1 INTRODUCTION

The project area is situated within the Hat Creek drainage basin. Several small creeks, Medicine, Finney, Ambusten, and Houth, 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 and Finney 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 mm/a, 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 South-West 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 condition 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.

11.2 DUST

Initial studies of the air quality impacts of the mine indicated a potentially serious problem with dust. As a result, B.C. Hydro instructed the Mining Consultants (CMJV) to examine the problem and to devise suitable measures to ensure that the B.C. Pollution Control Branch guidelines 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. Results of this work, endorsed by the original air quality consultants, indicated that dust was indeed controllable, provided certain actions were undertaken. These proposed dust control measures, reviewed and accepted by B.C. Hydro, include both design changes and operating factors, for example:

- Blending piles: The present blending area was moved from its original position where the present mine services facilities are located. In addition, the area would be constructed "into" the adjacent hill to an elevation of 930 m, a protective dike to 950 m would be constructed along the SW edge of the area, and the coal piles would be suitably contoured to reduce erosion. Stacking out would be carried out with a telescopic chute on the boom conveyor. An effective water spray system would be installed;

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

- Binding agents would be used to control erosion where appropriate;

- Areas that would remain stripped for extended periods of time would be revegetated.

11.3 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 dBA in the areas away from Highway 12. Adjacent to the highway, noise levels range from 44 to 51 dBA. By comparison, a soft whisper would produce a sound level of about 30 dBA, and a quiet wind through the trees would be around 50 dBA.

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 dBA which is close to the 45 dBA typically set as a nighttime level by many communities.

The South-Western 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 dBA; the next two, levels of between 45 and 49 dBA; and the two furthest away, levels of 41 to 42 dBA. As the natural background level is 35 to 40 dBA, the occasional level of noise from the mining operation is not expected to cause annoyance to anyone reasonably disposed.

11.4 MINE DRAINAGE AND WATER QUALITY

Drainage measures in so far 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. Details are provided in Section 6. 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 water course;

(2) Run-off contaminated with suspended solid material would undergo sedimentation to reduce the concentration of suspended solids to less than 50 mg/L;

(3) All water of unsuitable quality for discharge would be collected in leachate pond and disposed of on site by re-use in dust control or by spray evaporation on waste dumps.

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 re-establish a suitable wetland habitat in the early stages of the project. All drainage ditches would be revegetated to reduce suspended solids contamination.

11.5 LAND RECLAMATION

11.5.1 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 B.C. Hydro 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.

Slope Plots

Sloped revegetation test areas were constructed at Houth Meadows and Medicine Creek to examine the revegetation potential on typical embankment material, gravel at Houth Meadows and till at Medicine Creek, at slopes of 22^{o}, 26^{o}, and 30^{o}. Half of each plot at Houth Meadows was covered with a thin layer of top soil. Aspect and altitude were selected to simulate as closely as possible climatic conditions to be encountered at the Medicine Creek and Houth Meadows waste disposal embankments. Both areas were hydro-seeded with a single seed mix in the Fall of 1977 and subsequently fertilised in the Spring of 1979.

Growth assessments were made during the latter part of the 1978 and 1979 growing seasons and the plots examined for signs of water-borne erosion. Results of these examinations have shown that there is essentially no difference in the success of vegetation establishment on the materials without topsoil at the three slope angles; in all cases growth was satisfactory with a good mix of grass and legume. Soil erosion due to runoff was not apparent even though several thunderstorms were experienced during this period. On the topsoil-treated plots at Houth Meadows, growth of seeded species was severely inhibited by the abundance and vigorous growth of weeds, the seeds of which were transported to the site in the topsoil.

From the results of these studies it is concluded that embankment slopes at Houth Meadows and at Medicine Creek could be constructed to stable and reclaimable slopes at least up to 30°.

Waste Material Test Plots

Seven waste materials were identified during the excavations of the 1977 Bulk Sample Program. Samples of these materials and of fly-ash from the Battle River Combustion Tests were set out in 15 m x 15 m x 1 m plots near Aleece Lake. Half of each plot was covered with a thin cover of topsoil and seeded in the Fall of 1977 with three different seed mixes of four species each. The soil characteristics of the mine waste materials suggest that they fall into essentially three categories, namely surficial materials such as colluvium (till), gravel, and baked clay; non-seam waste, gritstone (sandstone/claystone), and bentonitic clay; seam waste such as carbonaceous shale and waste coal. Each plot was fertilized during the Spring of both 1978 and 1979, based on recommendations from the B.C. Ministry of Agriculture following soils testing.

Detailed vegetation monitoring was carried out after one growing season to determine the success of revegetation based on seedling emergence and biomass production. A less comprehensive evaluation was conducted during 1979 to further monitor the progress of these test plots. The results of these studies may be summarized as follows:

(1) 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. These results indicate that the materials selected for stripping, stockpiling and/or use as surface growth media, may comprise any of these surface materials, gravel, colluvium (till), baked clay, and topsoil, either separately or in combination. 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;

(2) Revegetation of non-seam mine waste, gritstone (sandstone/claystone), and bentonitic clay proved to be more difficult to achieve in the short term. In addition to low emergence success, the biomass production of vegetation was poor; most plants exhibited leaf chlorosis. The physical and chemical properties of these soils contribute to poor soil structure under extreme conditions of moisture and nutrient imbalances.

    The addition of topsoil proved beneficial although plants remained somewhat stunted. Subsequent 1979 observation showed some species of vegetation progressing well on the untreated gritstone. Nevertheless, it is considered that a surface capping of surficial material would be required to satisfactorily revegetate these waste materials;

(3) Seam waste was the most difficult of all materials tested to vegetate. However, the chemical characteristics of these materials appear to be less of a deterrent than do their physical properties, particularly their dark colour resulting in excessive surface temperatures and the hydrophobic nature of the carbonaceous shale. A capping of surficial material would be required for satisfactory reclamation.

Waste Dumps

Waste material stockpiles from trenches A and B were seeded in late 1977. Piles at Trench C and unsatisfactory portions of the piles at Trench A were seeded in the Fall of 1978. Topsoil (15 cm) was applied to half of the uppermost dump at Trench A and to half of the dumps at Trench C. In addition, water retention furrows were constructed across the dump fall line in an attempt to improve moisture retention on the dump surface.

The results on these dumps confirm the results at Aleece Lake and the slope plots. Gravels and baked clay are readily revegetated, while bentonitic clay and gritstone show less success. Germination in topsoil was substantially less successful than germination in baked clay. Further, 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 mm.

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 mixes of four and five species were devised and, in some instances, species were used individually.

Results of these field tests have identified several species which could be used for reclamation purposes at Hat Creek. Among the grass species the following perennial grasses show excellent potential: Crested Wheatgrass (Nordan), Streambank Wheatgrass (Sodar), Slender Wheatgrass, Tall Wheatgrass (Altar), and Smooth Bromegrass (Manchar). 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, it is suspected of inhibiting the growth of other perennial species with which it was seeded. As a result, its use would be restricted to those occasions where short-term revegetation - for example, for dust control - is required.

Several legumes have been tested. Of these Alfalfa (Drylander) and Sainfoin (Melrose) have proved most successful. Double-cut red clover and white clover, a biannual, showed lesser success, but may be useful as minor species in seed mixes. All legumes performed better when competition from other plants was absent.

The selection of species for revegetation of waste dumps and related areas at Hat Creek would be largely based on those identified above. 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.

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

11.5.2 *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.

Waste dumps will be concurrently revegetated to an end use comparable with adjacent lands at similar elevation. Topography and diversity of native species similar to pre-mining conditions cannot be duplicated, but reclamation of waste dumps will be designed to provide a revegetated, self-sustaining, stable surface composed of materials similar to, or better, than those of adjacent lands. Presently, land in the area is used for mixed wildlife and agricultural (mostly ranching) purposes. It is proposed to revegetate waste dump surfaces to a similar land use as now exists.

As a result of on-site testing as described in 11.5.1, it is proposed to strip and to stockpile surface soils only from those areas where the depth of the soil allows for economic extraction methods. Allowance has been made to cover waste dumps comprised of seam or non-seam material with approximately 1 m of surficial material, though the precise depth required would be established through further on-site testing. Those areas exposed by stripping, and stockpiles of surficial material, would be temporarily revegetated to prevent erosion and dusting.

11.5.3 Material Storage Areas After Abandonment

The coal stockpile and blending area will be levelled and sloped to harmonize with the surrounding topography. The contoured surfaces will then be covered with a buffering medium of non-sodic overburden, and seeded.

The surficial soil stockpiles will decrease progressively as the soil is spread over disturbed lands throughout the mine site. The remains will be levelled, sloped to blend in with the surrounding topography, and seeded.

11.5.4 Transportation Corridors

These take up about 4% of the disturbed land within the mine area. Before construction, suitable surface soils will be removed and stockpiled. Inactive sections will be seeded as soon as possible after construction, to minimize dusting and erosion. Cut and fill slopes will be graded to 26° and seeded. Trees and brush will be removed from rights-of-way to reduce any fire hazard.

During the years immediately following shutdown of the mine, conveyors, transmission lines, and culverts will be dismantled and removed. Wherever possible, corridors will be re-sloped to blend in with the surrounding topography. All roads (except main access roads) will be ripped to relieve compaction, and seeded. Water bars will be constructed on slopes with a potential for rill erosion.

11.5.5 Support Facilities

The present site has a poor quality rating in terms of land use, and the reclamation measures will be designed to enhance the value of the disturbed land.

During construction, the buildings will be screened from the main access roads by a belt of trees, not merely to improve the appearance, but to prevent dusting. No significant reclamation will take place until after the mine closes.

During the years following closure, buildings not retained for alternative uses will be dismantled, sold, and levelled to their foundations. Any areas littered will be cleared during the mine clean-up operation. Most of the Mine Services Area will then be ripped to relieve compaction, covered with 15 to 30 cm of soil, and seeded. Where practical, slopes will be regraded to blend in with the surrounding topography. Where surface materials are unsuitable for plant growth, a suitable depth of overburden will be placed before soil coverage and seeding.

11.5.6 The Open Pit After Abandonment

Considerable planning has gone into measures designed to minimize the potential hazard to human life, livestock, and wildlife, of a large void constructed of weak material. A proposal to flood the pit and convert it into a lake was explored but rejected on the grounds of poor stability of the surrounding ground, the anticipated poor quality of the pit water, and the costly and possibly irrevocable nature of a decision that would make it virtually impossible to re-open the pit at some future time in order to extract the substantial coal reserves which will remain.

The plan adopted provides for re-sloping the top three benches (about 115 ha) from 45° to 26° to provide a safer perimeter and lessen the visual impact. No re-sloping will be done below this level. After re-sloping, fertilizer and seed will be aerially broadcast on all pit benches. Germination is expected to take place readily on those portions which consist of non-sodic glacio-fluvial and glacial till overburden, less readily on those composed of saline slide deposits, sodic siltstones, claystones, and coal. In time, revegetated overburden and slide areas may be expected to creep and to slump into the pit.

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

11.5.7 Disturbances and Possible Resource Losses

A maximum of 1,931 ha will, at one time or another, have been disturbed by Year 35. By Year 45, however, all but 571 ha will have been restored. This represents less than 1% of the Hat Creek Watershed. Of this, 80 ha include transportation corridors, lagoons, reservoirs, and remaining facilities for long-term environmental monitoring. The remaining 491 ha represents the lower portions of the pit itself which would remain as is so as not to preclude the further economic extraction of the coal resource. Table 11-1 shows the details of areas disturbed and those reclaimed by Years 35 and 45. The areas to be reclaimed by Year 45 are shown in Figure 11-1.

The only resources likely to be buried or otherwise alienated by the mine are aggregate (sand and gravel), and some limestone deposits. The latter are insignificant in comparison with the large reserves of limestone in areas immediately adjacent. And much of the aggregate excavated during construction will be stockpiled for future use.

11.6 OTHER MEASURES

11.6.1 Spontaneous Combustion

Tests were carried out in 1977 to determine the spontaneous combustion of loose and compacted coal piles. These showed that with a temperature rise of 60°-$70^{\circ}C$, fires began in piles of loose, uncompacted coal in between four to eight weeks. Precautionary measures will be required to crush and compact the coal in such stockpiles. The problem can be minimized by the restacking time coal spends in uncompacted piles. Monitoring of coal-pile temperature will be required.

11.6.2 Environmental Services Complex

All aspects of the Reclamation and Environmental Protection Plan will be under the control of staff housed in the Environmental Services Complex, which is equipped with modern laboratory facilities. The plant propagation nursery forms part of the complex.

11.6.3 Monitoring

Geotechnical monitoring of pit slopes, waste dumps, and slide areas will continue during mining to ensure operational safety as well as to develop reliable abandonment procedures. The temperature of carbonaceous material will be monitored to prevent spontaneous combustion. The quality of soil and buffer material will be monitored to ensure adequate depth of uncontaminated growth medium. In addition, the quantity, quality, regeneration potential, nutrient and metal content of vegetation grown on disturbed land will be monitored to determine if the vegetation is self-sustaining and satisfactory for livestock and wildlife consumption.

Surface and groundwater will be monitored to ensure compliance with PCB objectives. Seepage and leachate flows will be monitored in groundwater wells; also the discharge from all treatment lagoons. Broadly speaking, the primary objective of water monitoring will be to segregate sediment-laden water, clear water, waste water, and other surface flow, treat those parts which may require it, and discharge the rest.

Air quality would be monitored for suspended particulate and dustfall levels.

Monitoring will continue for at least 10 years after the mine closes.

11.6.4 Archaeology

A conservation strategy will be adopted to preserve significant heritage resources discovered during the process of excavation. As scraper work will be carried on progressively during mining, there will be sufficient time to complete an inventory of such heritage resources as may be uncovered.

11.7 COSTS

The estimated capital and operating costs of the plan amount to $40 million. This includes the capital cost of all buildings, field equipment, seed and plant stock. Operating costs include staff and all materials required to carry out the plan.

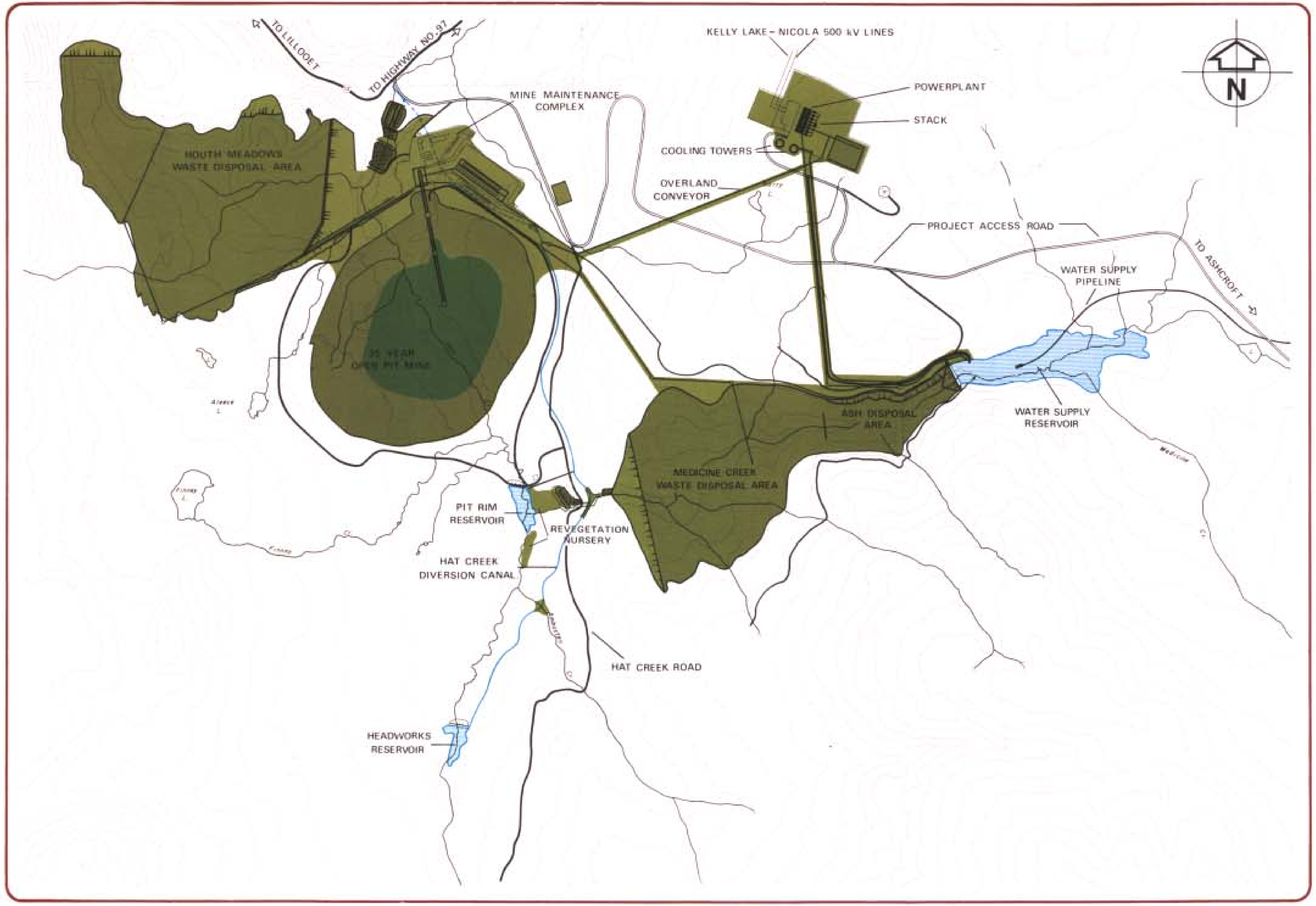
The reclamation costs span a 6-year pre-production period, 35 years of production, and a 10-year post-production period devoted to reclamation, which will cost approximately $7 million.

> The Environmental Protection Section of this report is based upon Volume V of CMJV's July 1978 report entitled "Mine Reclamation and Environmental Protection". It has not been adjusted to reflect changes in the 1979 Mining Plan, which are considered to be insignificant.

TABLE 11-1

ESTIMATE OF AREAS DISTURBED AND RECLAIMED BY YEAR 35 AND YEAR 45

Location	Disturbed by Year 35	Reclaimed by Year 35	Reclaimed by Year 45
Open Pit			
Upper 3 Berms	115	0.0	115
Balance	491		
Sub-total	606		115
Waste Dumps			
Houth Meadows	610	380.0	610
Medicine Creek	385	212.0	385
Sub-total	995	592.0	995
Stockpiles			
Low-Grade Coal	17.2	17.2	17.2
Coal	26.4	0.0	26.4
Topsoil	13.6	13.6	13.6
Sub-total	57.2	30.8	57.2
Service Yards	107	6.0	106.8
Roads			
Pit Perimeter	47.3	15.0	15.0
Main Access	3.0	1.0	1.0
Sub-total	50.3	16.0	16.0
Conveyor Corridors			
Thermal Plant	14.0	7.0	14.0
Medicine Creek	6.0	3.0	6.0
Sub-total	20.0	10.0	20.0
Water Treatment Lagoons			
Main	9.0	2.0	2.0
Medicine Creek	2.0	0.5	0.5
Sub-total	11.0	2.5	2.5
Clearwater Reservoirs			
Headworks (upper)	6.1	2.0	2.0
Pit Rim (lower)	8.8	4.0	4.0
Sub-total	14.9	6.0	6.0
Ditches	27.0	6.5	6.5
Stream Diversions			
Hat Creek	33.6	27.0	27.0
Finney Creek	8.9	8.0	8.0
Sub-total	42.5	35.0	35.0
GRAND TOTAL	1,931.0	704.8	1,360.0



LEGEND

RECLAIMED AREA (MAJOR RECLAMATION)

RECLAIMED AREA (MINOR RECLAMATION)

SCALE — 1:40,000

1 km 0 km 1 km 2 km

CONTOUR INTERVAL — 50 METRES

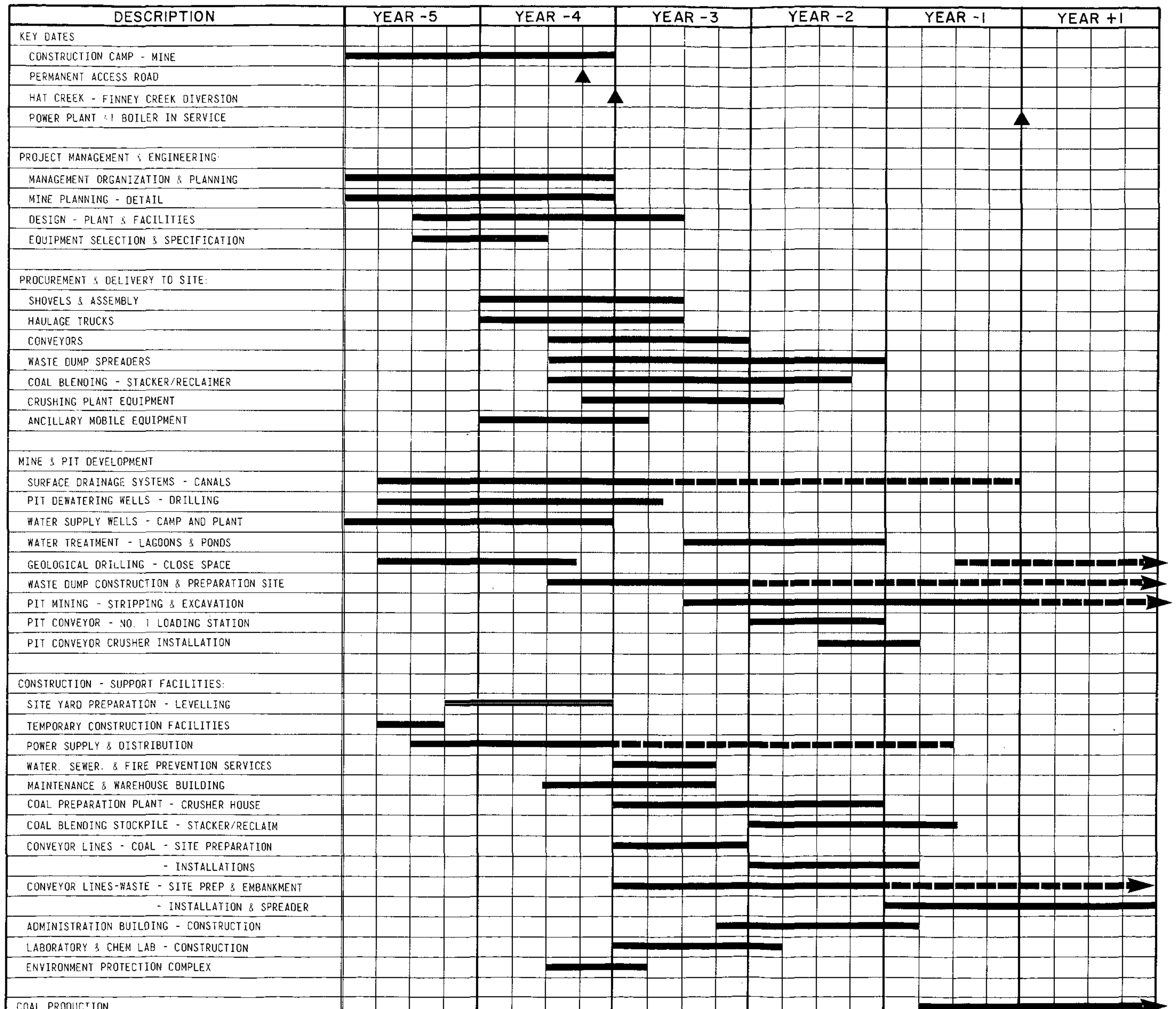
HAT CREEK PROJECT

FIGURE 11-1

RECLAMATION — YEAR 45

00142 2/2 2

SOURCE: Cominco — Monenco Joint Venture (5)



HAT CREEK PROJECT

FIGURE 12-1
CONSTRUCTION SCHEDULE

SOURCE: British Columbia Hydro and Power Authority

No.	LIST OF TABLES	Page

LIST OF FIGURES

TABLE 13-1

SCHEDULE OF PEAK MANPOWER REQUIREMENTS

	Pre-prod.	1	2	3	4	5	6-15	16-25	26-35
1. Management									
Managers	2	2	2	2	2	2	2	2	2
Industrial Engineering	4	4	4	4	4	4	4	4	4
Public Relations	2	2	2	2	2	2	2	2	2
Computer Services	4	4	4	4	4	4	4	4	4
Laboratory Services	5	5	5	5	5	5	5	5	5
Environment Protection and Reclamation	7	7	7	7	7	7	7	9	9
Secretaries, Clerical, Stenos.	8	8	8	8	8	8	8	8	8
	32	32	32	32	32	32	32	34	34
2. Administration - Supt.	1	1	1	1	1	1	1	1	1
Finance - Cost Accounting and Payroll	4	4	4	4	4	4	4	4	4
Materials Management - Purchasing, Warehouse	10	10	10	10	10	10	10	10	10
Security Services	5	5	5	5	5	5	5	5	5
Fire Department	3	3	3	3	3	3	3	3	3
Janitorial Services and Office Bldgs. Maintenance	10	10	10	10	10	10	10	10	10
Dry Facilities	4	4	4	4	4	4	4	4	4
Communications - Radio, Telephone, Telex	12	12	12	12	12	12	12	12	12
Water, Fuel, Sewerage; Wastewaters, Garbage Disposal	5	5	5	5	5	5	5	5	5
Transportation - Bus Service and Delivery Drivers	5	18	21	21	21	21	21	21	21
Surface Labour - Road Maintenance, Plant Yard Scrap	6	6	6	6	6	6	6	6	6
Secretaries, Clerical, Stenos.	29	29	29	29	29	29	29	29	29
	90	103	106	106	106	106	106	106	106

	Pre-prod.	1	2	3	4	5	6-15	16-25	26-35
3. Human Resources - Supt.	1	1	1	1	1	1	1	1	1
Labour Relations Dept.	2	2	2	2	2	2	2	2	2
Training School	8	8	8	8	8	8	8	8	8
Personnel Department	3	3	3	3	3	3	3	3	3
Safety and First Aid	5	7	7	7	7	7	7	7	7
Secretaries, Steno.	6	6	6	6	6	6	6	6	6
	25	27	27	27	27	27	27	27	27
4. Mine Engineering - Supt.	1	1	1	1	1	1	1	1	1
Pit Engineer	1	1	1	1	1	1	1	1	1
Planning Engineers	4	4	4	4	4	4	4	4	4
Draftsmen	2	2	2	2	2	2	2	2	2
Samplers and Grade Officer	3	3	4	5	5	5	5	5	5
Surveyors	4	4	4	4	4	4	4	4	4
Geologists	4	5	6	6	6	6	6	6	6
Geotechnical Engineers and Technician	2	2	2	2	2	2	2	2	2
Helpers (Survey, Rodmen)	4	4	4	4	4	4	4	4	4
Secretary, Clerical, Stenos.	2	2	2	4	4	4	4	4	4
	27	28	30	33	33	33	33	33	33
5. Mine Supervision (Operations)									
Supt. and Assistant	2	2	2	2	2	2	2	2	2
Shift Supervisors	2	2	4	4	4	4	4	4	4
Production - Shift Foremen	3	4	4	4	4	4	4	4	4
Production - Dump Pocket Foremen, Crusher	4	4	4	4	4	4	4	4	4
Processing - Coal Plant Foremen	2	2	3	4	4	4	4	4	4
Processing - Waste Dump Foremen	4	4	4	4	4	4	4	4	4
Pit Maintenance - Roads Foremen	4	4	4	4	4	4	4	4	4
Pit Maintenance - Drainage Foremen	4	4	4	4	4	4	4	4	4
Clerical and Steno.	3	3	4	5	5	5	5	5	5
	28	29	33	35	35	35	35	35	35

	Pre-prod.	1	2	3	4	5	6-15	16-25	26-35
6. Maintenance Engineering (Supervision)									
Supt. and Assistant	2	2	2	2	2	2	2	2	2
Engineering Design - Engineers, Draftsmen	3	3	3	3	3	3	3	3	3
Maintenance Planning	2	2	3	5	5	5	5	5	5
Operations Maintenance - Mechanical	3	5	6	8	9	9	9	9	9
Operations Maintenance - Electrical	4	5	6	6	6	6	6	6	6
Pit Equipment Maintenance - Mechanical Supervisor	2	3	4	5	5	5	5	5	5
Surface Yard and Carpenter Foremen	2	2	2	2	2	2	2	2	2
Clerical and Steno.	3	4	4	4	4	4	4	4	4
	21	26	30	35	36	36	36	36	36
7. Mine Operations									
Shovel - Operators	4	6	10	15	21	23	24	20	15
Shovel - Oilers	2	3	5	6	9	10	11	8	6
Haulage Truck Drivers	9	14	24	36	52	60	62	59	42
Conveying Coal	10	10	21	26	29	29	30	30	30
Conveying Waste		23	24	28	28	28	36	36	36
Drilling and Blasting	2	3	3	3	3	3	4	3	3
Heavy Equipment Operators	50	57	68	68	73	81	81	54	42
Service Truck Drivers	8	9	11	12	16	18	18	18	15
Mobile Crusher	2	2	2	2	2	2	2	2	2
Pit Dewatering and Drainage	5	6	6	6	7	7	7	7	7
Total Mine Operations	92	139	174	192	240	261	275	237	198

	Pre-Prod.	1	2	3	4	5	6-15	16-25	26-35
8. Maintenance Labour									
Central Shops and Services:									
H.D. Mechanic	24	30	36	44	49	54	54	44	34
Auto Mechanic	6	8	8	12	18	19	19	16	12
Tiremen	6	8	8	8	8	10	10	8	6
Welder	7	7	10	10	14	16	18	14	8
Machinist	4	4	6	6	6	8	10	8	6
Carpenter	3	3	4	6	6	8	8	8	6
Pipefitter	3	3	3	3	3	3	3	3	3
Painter	2	2	2	2	2	2	2	2	2
Radio Technician	2	3	3	3	3	3	3	3	3
Electrician	12	14	18	23	23	23	23	23	23
Crane Operator	3	3	3	3	3	3	3	3	3
Labourer	10	10	12	12	14	16	16	14	12
Field Services:									
H.D. Mechanic	8	8	18	22	28	28	28	22	15
Conveyor Mechanic	3	12	17	27	34	34	40	45	38
Belt Vulcanizer	2	4	4	4	4	4	4	4	4
Lube Servicemen	5	6	9	9	10	10	10	10	10
Total Labour	100	125	161	194	225	241	251	229	185

<u>TABLE 13-2</u>

MANPOWER SCHEDULE - SUMMARY

	Pre-prod.	1	2	3	4	5	6-15	16-25	26-35
Management and Reclamation P.C.	32	32	32	32	32	32	32	34	34
Administration and Site Services	90	103	106	106	106	106	106	106	106
Human Resources	25	27	27	27	27	27	27	27	27
Mine Supervision - Engineering	27	28	30	33	33	33	33	33	33
Mine Supervision - Operations	28	29	33	35	35	35	35	35	35
Maintenance Supervision	21	26	30	35	36	36	36	36	36
Mine Operations - Labour	92	139	174	192	240	261	275	237	198
Maintenance - Labour	100	125	161	194	225	241	251	229	185
Subtotals	415	509	593	654	734	771	795	737	654
Contingency - 10%	42	51	59	65	73	77	79	74	65
Totals	457	560	652	719	807	848	874	811	719

HAT CREEK PROJECT

FIGURE 13-1

Organization Chart

00142 2/2 3

SOURCE: British Columbia Hydro and Power Authority

14 ECONOMICS AND COSTS ESTIMATES

Page

LIST OF TABLES

LIST OF FIGURES

SECTION 14

ECONOMICS AND COSTS ESTIMATES

14.1 SUMMARY

Having selected the shovel/truck/conveyor mining system, a production schedule was developed which satisfied the annual fuel requirements of the proposed generating station over the 35-year project life. The capital and operating costs associated with all mining operations necessary to deliver coal to the powerplant in accordance with this schedule are presented in this section.

The following criteria were used in preparing the cost estimates:

(1) The cost estimates are presented in 1979 Canadian dollars;

(2) Operating costs incurred in the pre-production years have not been transferred to capital costs;

(3) The following costs were excluded from the estimates: B.C. Hydro corporate overhead, land purchase or lease costs, mineral rights purchase or lease costs, and the costs of housing mine personnel.

On the basis of the recommended mining system, the total estimated capital costs of this project are \$538,261,000, and the total estimated operating costs are \$1,836,060,000. The estimated capital requirements to full production, i.e. to the end of the third production year, are \$248,463,000. Table 14-1 provides an overall project cost summary, including the annual operating and capital costs, the cumulative cash flow, and the annual unit costs. The total estimated capital and operating expenditures over the life of the project are summarized according to major cost centres in Table 14-2.

TABLE 14-1

SUMMARY OF ANNUAL COSTS, CANADIAN $ OCTOBER 1979

HAT CREEK PROJECT MINING REPORT 1979

Year	Annual Coal Production tonnes x 10^6	Annual Coal Production MJ x 10^9	$000's Annual Operating Cost	$ Annual Operating Cost/tonne	$ Annual Operating Cost/GJ	$000's Annual Capital Costs	$000's Total Annual Capital + Operating Cost	$000's Total Cumulative Operating + Capital Cost
-6						69	69	69
-5			72			6,144	6,216	6,285
-4			2,367			25,864	28,231	34,516
-3			5,531			40,985	46,516	81,032
-2			18,638			76,219	94,857	175,889
-1	1.14	15.35	28,548	25.04	1.86	38,395	66,943	242,832
1	2.95	43.56	34,091	11.56	0.78	19,205	53,296	296,128
2	4.76	66.27	39,523	8.30	0.60	34,504	74,027	370,155
3	7.35	101.77	46,480	6.32	0.46	7,078	53,558	423,713
4	9.23	130.63	53,434	5.79	0.41	11,242	64,676	488,389
5	10.46	146.43	57,221	5.47	0.39	22,176	79,397	567,786
6	10.60	150.02	57,615	5.44	0.38	9,563	67,178	634,964
7	10.52	149.69	57,091	5.43	0.38	16,643	73,734	708,698
8	11,49	150.31	57,438	5.00	0.38	17,126	74,564	783,262
9	10.69	149.65	57,226	5.35	0.38	10,168	67,394	850,656
10	11.37	150.48	59,441	5.23	0.39	9,152	68,593	919,249
11	11.17	150.39	56,669	5.07	0.38	17,878	74,547	993,796
12	10.86	150.37	59,898	5.52	0.40	9,747	69,645	1,063,441
13	11.64	149.60	60,782	5.22	0.41	12,970	73,752	1,137,193
14	11.40	150.00	60,356	5.29	0.40	15,260	75,616	1,212,809
15	11.12	149.72	56,165	5.05	0.38	2,846	59,011	1,271,820
16	10.06	138.53	54,271	5.39	0.39	4,852	59,123	1,330,943
17	10.06	139.30	53,624	5.33	0.38	10,085	63,709	1,394,652
18	10.67	139.58	52,352	4.91	0.38	4,919	57,271	1,451,923
19	10.15	138.99	50,808	5.01	0.37	18,582	69,390	1,521,313
20	9.90	139.35	50,420	5.09	0.36	14,869	65,289	1,586,602
21	9.66	139.67	48,503	5.02	0.35	4,396	52,899	1,639,501
22	9.57	139.10	48,964	5.12	0.35	4,212	53,176	1,692,677
23	10.40	139.23	48,972	4.71	0.35	12,966	61,938	1,754,615
24	10.03	139.65	48,707	4.86	0.35	3,905	52,612	1,807,227
25	9.83	139.12	48,569	4.94	0.35	3,150	51,719	1,858,946
26	8.01	117.65	46,557	5.81	0.40	10,817	57,374	1,916,320
27	9.19	118.11	46,552	5.07	0.39	2,214	48,766	1,965,086
28	8.39	118.10	46,548	5.55	0.39	6,016	52,564	2,017,650
29	8.55	117.73	46,696	5.46	0.40	13,131	59,827	2,077,477
30	8.58	117.49	46,211	5.39	0.39	3,175	49,386	2,126,863
31	8.64	117.65	45,913	5.31	0.39	3,251	49,164	2,176,027
32	8.61	117.90	45,874	5.33	0.39	9,475	55,349	2,231,376
33	8.25	118.02	45,706	5.54	0.39	1,866	47,572	2,278,948
34	8.05	117.62	39,984	4.97	0.34	2,378	42,362	2,321,310
35	7.62	117.75	39,426	5.17	0.33	508	39,934	2,361,244
36			2,416			94	2,510	2,363,754
37			2,416			17	2,433	2,366,187
38			2,416			22	2,438	2,368,625
39			2,095			35	2,130	2,370,755
40			2,095			45	2,140	2,372,895
41			483			10	493	2,373,388
42			483			23	506	2,373,894
43			289			14	303	2,274,197
44			62				62	2,374,259
45			62				62	2,374,321
Total	330.95	4,574.78	1,836,060	5.55	0.40	538,261	2,374,321	2,374,321

TABLE 14-2

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

($000's October 1979)

Hat Creek Project Mining Report 1979

Cost Centre	Amount ($000's)	($) Unit Cost/tonne of Coal Delivered	($) Unit Cost/GJ
Engineering and Construction Costs	29,168	0.09	
Mine Property Development	44,566	0.13	
Buildings and Structures	18,077	0.05	
Mining Equipmen	257,757	0.78	
Coal Conveying, Crushing, and Blending Equipment	51,341	0.16	
Low-grade Coal Beneficiation Equipment	9,549	0.03	
Waste Disposal Equipment	73,101	0.22	
Reclamation and Environmental Protection	1,535	0.01	
Contingency	53,167	0.16	
TOTAL CAPITAL COSTS	538,261	1.63	0.12
Drilling	2,008	0.01	
Blasting	5,764	0.02	
Loading	97,739	0.30	
Hauling	262,072	0.79	
Coal-handling System	70,835	0.21	
Waste-handling System	85,585	0.26	
Auxiliary Equipment	111,740	0.34	
Power	140,293	0.42	
General Mine Expense (less Reclamation and Environmental Protection)	385,068	1.16	
Reclamation and Environmental Protection	38,638	0.12	
Overhead	251,416	0.76	
Royalties	115,837	0.35	
Contingency	145,110	0.44	
Contractor's Allowance	123,955	0.37	
TOTAL OPERATING COSTS	1,836,060	5.55	0.40

14.2 ESTIMATING CRITERIA

14.2.1 Introduction

This section of the report discusses the approach used in developing the cash flow of capital and operating costs for the recommended mining scheme.

The capital cost estimating criteria, primarily concerning the cost and service life of the major equipment, are provided in Section 14.2.3.

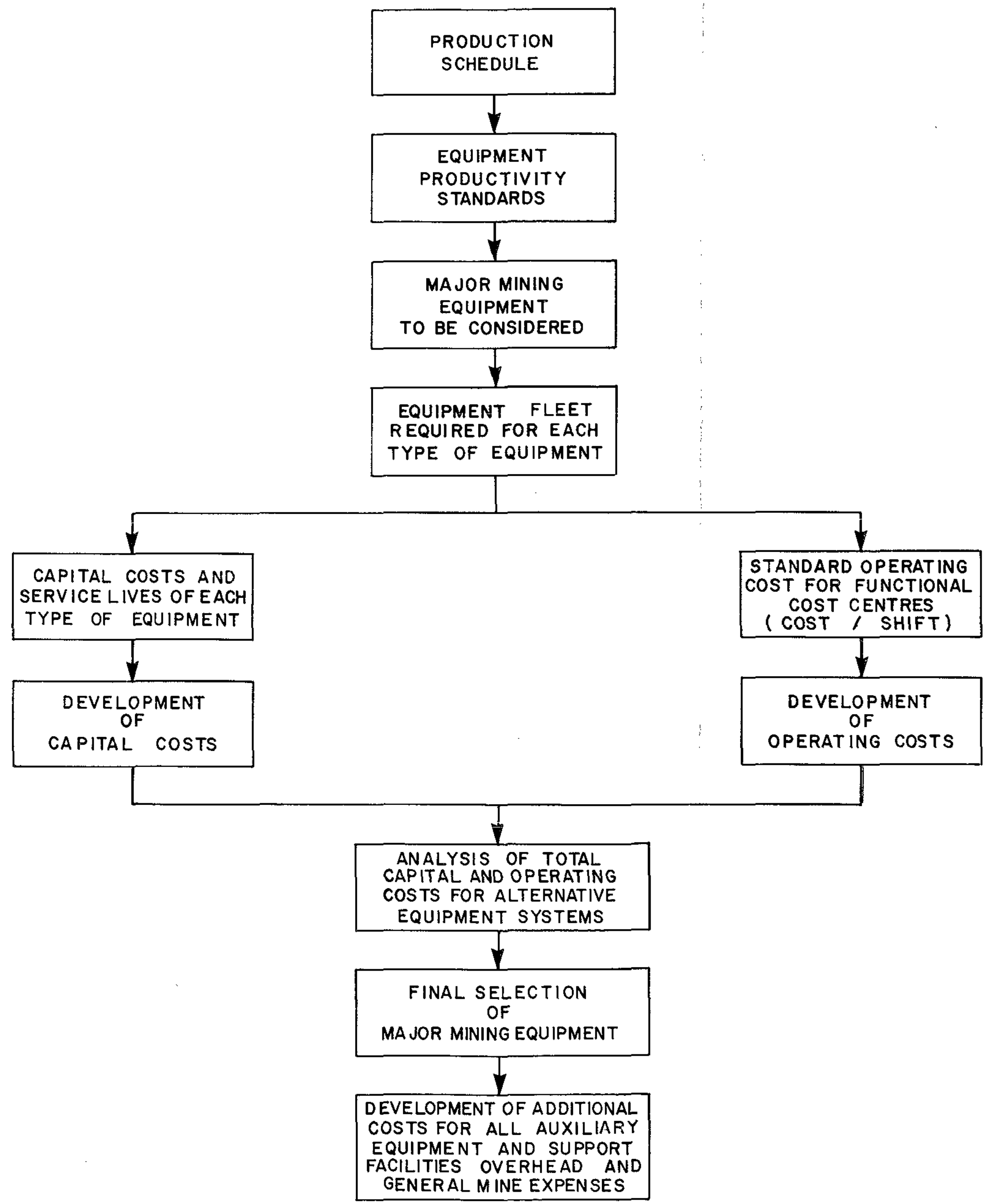
The estimating criteria used in developing the operating costs are presented in Section 14.2.4 and consist mainly of labour rates and major equipment productivity and cost standards.

14.2.2 Overall Approach to the Development of the Project Cash Flow

The system used to develop annual capital and operating costs is shown in Figures 14-1, 14-2, and 14-3.

Having determined a practical and economic production schedule for the recommended shovel/truck/conveyor mining system, the next phase of the study was to carry out an analysis to determine the size and type of shovels and trucks to be employed in the mining operation. This analysis is discussed in Section 9. Basically, owning and operating costs were developed for various combinations of shovel/truck systems at critical periods during the life of the project. An analysis of this information then resulted in the selection of the shovels and trucks for the mining system. Capital and operating costs for the major mining equipment were then produced for the project life.

All additional costs to support the mining system were then developed. These include the costs for all auxiliary equipment, support facilities, material-handling systems, reclamation and environmental protection, overhead and general mine expenses.

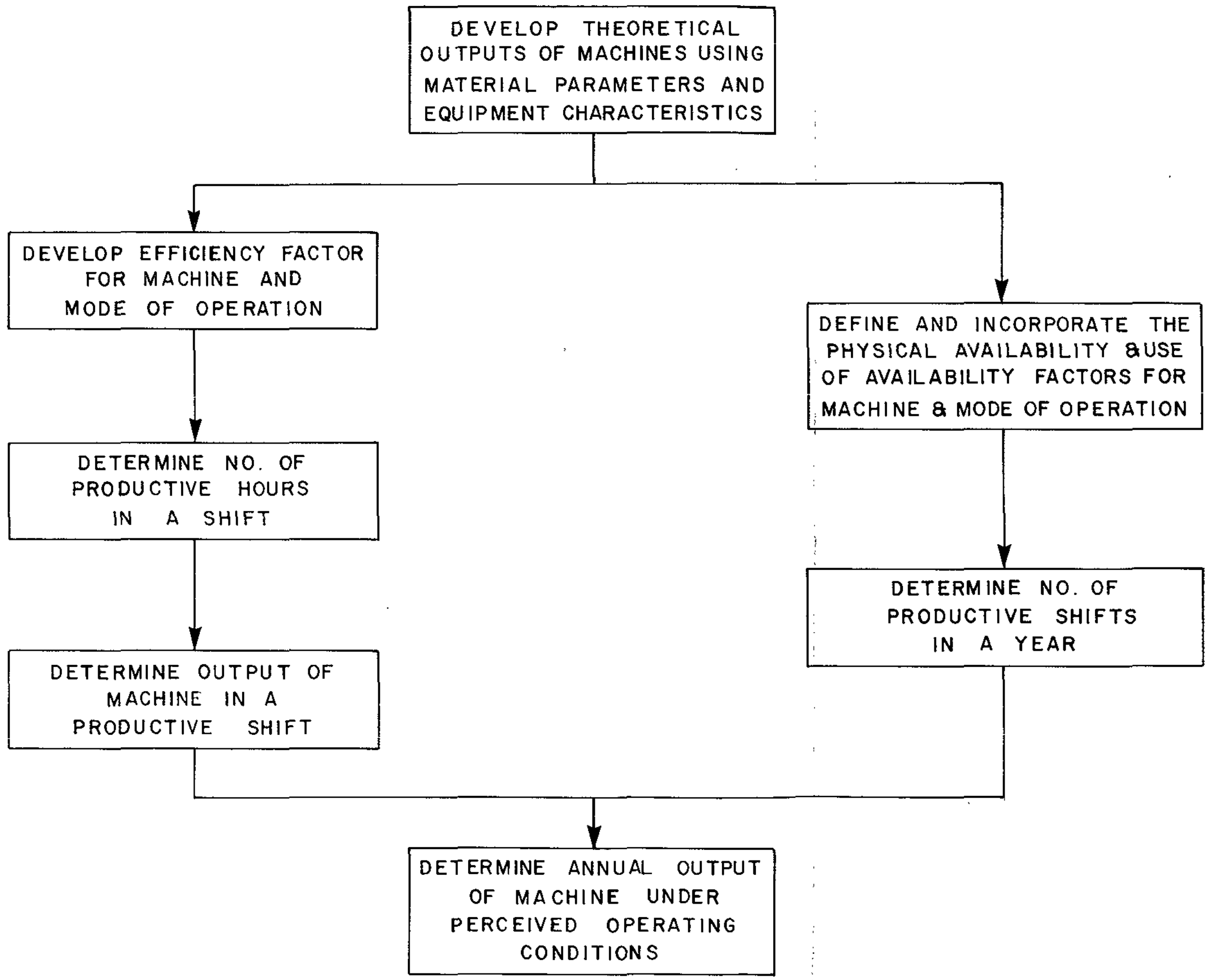


HAT CREEK PROJECT

FIGURE 14-1

Overall Approach to Development of Cash Flow

SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 14-2

Development of Equipment Productivity Standards

SOURCE: British Columbia Hydro and Power Authority

DEVELOP FORMAT FOR DETERMINING
COST / SHIFT OF A FUNCTIONAL COST CENTRE

↓

INCORPORATE COSTS FOR OPERATING LABOUR AND SUPPLIES AND MAINTENANCE LABOUR, SUPPLIES & OVERHEAD. ALL BASIC UNITS ARE IN SHIFT HOURS (8 HOURS / SHIFT)

↓

IDENTIFY THE VARIABLE TO WHICH EACH OF THE FIVE COST ITEMS ARE DIRECTLY RELATED

FOR EXAMPLE

OPERATING LABOUR (SEMI – FIXED) —— SCHEDULE WORKING HOURS OF MACHINE IN A YEAR OR NUMBER OF PRODUCTIVE SHIFTS X 8

MAINTENANCE LABOUR AND SUPPLIES (TOTALLY VARIABLE) —— PRODUCTIVE HOURS IN A SHIFT

↓

DETERMINE THE FACTOR WHICH MAKES ALL FIVE COST ITEMS DIRECTLY VARIABLE WITH THE NO. OF PRODUCTIVE HOURS

FOR EXAMPLE

A. IF OPERATING LABOUR IS RELATED TO SCHEDULED HOURS IN A YEAR

$$\text{CONVERSION FACTOR} = \frac{1}{\text{EFFICIENCY FACTOR}} \times \frac{1}{\text{P.A.}} \times \frac{1}{\text{U.O.A.}}$$

B. IF OPERATING LABOUR IS RELATED TO PRODUCTIVE SHIFTS

$$\text{CONVERSION FACTOR} = \frac{1}{\text{EFFICIENCY FACTOR}}$$

C. IF MAINTENANCE LABOUR IS RELATED TO PRODUCTIVE HOURS

CONVERSION FACTOR = 1

↓

DETERMINE STANDARD COST / SHIFT AND OPERATING AND MAINTENANCE MANPOWER REQUIRED FOR FUNCTIONAL COST CENTRE

HAT CREEK PROJECT

FIGURE 14-3

Development of Standard Costs per Shift for Machine Operations

SOURCE: *British Columbia Hydro and Power Authority*

Equipment requirements and costs for the project were developed with the aid of equipment productivity and cost standards.

14.2.2.1 Equipment Productivity Standards

Theoretical hourly outputs of machines were produced using the material parameters and machine characteristics. The number of productive hours in a shift were then determined for the mode of operation envisaged. From these factors the actual output of a machine in a shift was calculated. The experience of operating mines was used to develop the physical availability of machines working under a particular regime, as well as the use of such availability. These parameters enabled the number of productive shifts in a year to be calculated. Knowing the machines' output in a shift and the number of productive shifts in a year, the annual productivity of the machine under certain expected operating conditions was determined.

14.2.2.2 Equipment Cost Standards

The cost standards for different pieces of equipment were developed from the following five cost items: operating labour, operating supplies, maintenance labour, maintenance parts and supplies, and maintenance overhead. These cost standards were developed on a shift basis similar to those of the productivity standards.

Depending on the mode of operation, equipment, fleet size, labour agreements, etc., the cost item can be established as fixed, variable, or semi-fixed. For each cost item the price per basic unit was identified (e.g. labour rate for a shovel operator, power cost per hour of shovel operation), and the variable to which this basic unit was related (e.g. scheduled working hours, productive hours). From these data the standard cost per shift for each major piece of equipment was established. The cost data used in these calculations were based on the experience of operating mines in the Province. From the systematic development of a cost standard for each piece of equipment in a functional cost centre, it was then possible to build up the operating and maintenance manpower requirements.

14.2.3 Capital Cost Estimating Criteria

14.2.3.1 Buildings and Civil Works

The costs of civil works were developed taking into account prevailing labour agreements and productivity in the B.C. construction industry. Unit costs of major building components were reviewed with trade contractors.

14.2.3.2 Major Equipment

The capital costs for the major mobile mining equipment and the coal conveying, crushing, and blending equipment were developed based on manufacturers' listed prices and quotations in October 1979 dollars. The capital unit cost for each item of equipment includes:

- purchase cost of equipment FOB factory;
- allowance for optional extras;
- freight and insurance to site;
- Provincial sales tax at 4% of FOB site cost;
- erection costs at site.

Where manufacturers' quotations were in U.S. dollars, an exchange rate of $1.15 Canadian to $1.00 U.S. was used. A summary of the capital costs and service lives of the major equipment is presented in Table 14-3. These figures are based partly on suppliers' recommendations and partly on actual figures obtained from a survey of similar operations.

TABLE 14-3

EQUIPMENT CAPITAL COSTS AND SERVICE LIVES

Item	Capital Cost 1979 $ FOB Hat Creek	Service Life Op. Hours
Drills		
Auger: truck-mounted	198,000	15,000
Air-Trac c/w Compressor	130,000	15,000
Shovels (rope)		
16.8 m^3	3,360,000	90,000
Shovels (hydraulic)		
Poclain 1000 CK	1,365,000	36,000
Demag H241	2,220,000	45,000
Front-end Loader		
5.4 m^3	342,000	15,000
9.6 m^3	671,000	15,000
Haulage Truck		
32 t	295,000	25,000
77 t (coal box)	524,000	33,000
154 t (rock box)	776,000	33,000
Scraper		
Cat 631	360,000	15,000
Cat 637	415,000	15,000
Dozer (track)		
Cat 955	81,000	15,000

...continued...

Item	Capital Cost 1979 $ FOB Hat Creek	Service Life Op. Hours
Cat D7	175,000	15,000
Cat D8 with ripper	246,000	15,000
Cat D9 with ripper	355,000	15,000
Dozer (wheel)		
Cat 824B	219,000	25,000
Compactor		
Cat 825B	240,000	20,000
Vibratory (towed)	40,000	20,000
Grader		
Cat 14G	195,000	25,000
Cat 16G	261,000	25,000
Crane		
15 t	135,000	20,000
45 t	255,000	20,000
70 t	382,000	35,000
Trucks (miscellaneous)		
5 t service	19,000	20,000
3 t flatdeck (with 2 t crane)	26,000	20,000
Tire truck	38,000	20,000
Line truck	70,000	32,000
Fuel truck	60,000	20,000
Lube truck	91,000	20,000
Water wagon 45.5 kL	300,000	25,000
Pick-up 1 t	11,000	8,000
Pick-up 3/4 t	9,000	8,000
Fire truck	65,000	50,000
Ambulance	18,000	20,000
Personnel bus (24 passengers)	20,000	20,000
Personnel bus (12 passengers)	12,000	20,000
Sanding truck 10 t	35,000	25,000

...continued...

Item	Capital Cost 1979 $ FOB Hat Creek	Service Life Op. Hours
Pumps		
10 cm diesel	4,000	13,000
15 cm diesel	7,000	13,000
Welders (portable)		
600 A diesel	6,000	13,000
600 A electric	3,000	20,000
Miscellaneous		
Backhoe (1 m^3)	160,000	30,000
Compressor (17 m^3/min.)	63,000	25,000
Steam cleaner (mobile)	64,000	20,000
Lighting plant (3 kw)	11,000	10,000
Gradall	134,000	32,000
50 kw generator	21,000	32,000
Lo-boy tractor	86,000	32,000
Hi-boy trailer	43,000	32,000
Crushing plant	322,000	25,000
CaCl spreader box only	8,000	20,000
Lube island	86,000	PIT LIFE
Forklift 3 t (warehouse)	27,000	12,000
Forklift 5 t (shops)	54,000	12,000

14.2.4 Operating Cost Estimating Criteria

14.2.4.1 Staff Salaries and Benefits

Staff salaries were developed based on a salary survey conducted in 1979 for the Mining Association of British Columbia, as well as in-house experience of current salary levels in operating mines. A summary of the salaries used is presented in Table 14-4.

14.2.4.2 Hourly Labour Rates

Hourly wage rates and benefits were developed from a review of current labour agreements in eight B.C. mines and are based on mine operating schedules of 3 shifts/day and 1 shift/day. The rates used in the study are presented in Table 14-5. The payroll burden was estimated at 28% of the basic rate and included the following benefits:

	% of Base Rate
Company Pension Plan	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
Unemployment Insurance Commission	1.3
Canada Pension Plan	1.0
Medical Services Plan	0.8
Extended Health Plan	0.1
Dental Plan	0.8
Miscellaneous	1.0
Total	28.0%

14.2.4.3 Mine Operating Schedules

The schedule of operating and maintenance shifts at the mine is summarized below:

Operating and maintenance days per year (statutory holidays will not be worked)	354 days/year
Mine production shifts (1 crew on swing shift)	3 shifts/day
Mine maintenance shifts (1 crew on swing shift)	3 shifts/day
Shop maintenance shifts (1 crew on swing shift)	3 shifts/day
General service shifts (5 days/week, 52 weeks/year, no swing shift)	1 shift/day
Operating and maintenance hours/shift	8 hours/shift

14.2.4.4 Materials Parameters

The materials parameters, upon which calculations of productivities for the loading equipment are based, are given below:

Swell Factors	
Coal (fuel)	35%
Unconsolidated waste (above bedrock)	20%
Consolidated waste	30%
Specific Gravity	
Coal (fuel)	1.49
Unconsolidated waste (above bedrock)	2.00
Consolidated waste	2.00

14.2.4.5 Equipment Parameters

14.2.4.5.1 Drilling and Blasting Equipment

Drilling and blasting operations would be a minor part of the mining system employed at the Hat Creek Mine. Two drills (one crawler and one truck-mounted drill) and one 5-t blasters' truck will be employed in these operations. Drilling and blasting would be confined to approximately 10% of the total waste materials mined.

Tables 14-6 and 14-7 provide the productivity and cost standard for the drills.

14.2.4.5.2 Loading Equipment

Hydraulic shovels will be used for loading coal and waste materials, while front-end loaders will be employed as supplementary loading equipment. Coal shovels will be equipped with 10.7 m^3 buckets, while the waste shovels will have 14.5 m^3 buckets. The smaller hydraulic shovels will be required to load waste partings and waste zones within the major coal zones of the Hat Creek Deposit. Tables 14-6 and 14-7 provide the productivity and cost standards for the shovels performing specific tasks.

It should be noted that the coal shovels will have a low use of availability. This will occur because of the widespread work areas required to blend the various coals, and it is more economical and practical to have extra pieces of equipment to cover the pit than to have long and frequent equipment moves.

14.2.4.5.3 Haulage Trucks

77-t rear dump trucks will be used for hauling coal and some waste partings, while 154-t rear dump trucks will be the primary waste haulers. These trucks were selected after an economic analysis. (See Section 9) The productivities of these trucks in Year 6 and their cost standards are provided in Tables 14-6 and 14-7. Productivities were determined for the trucks throughout the 35-year project life to reflect the changing haulage cycles.

14.2.4.5.4 Coal and Waste-handling Systems

Equipment employed in the coal and waste-handling systems include truck dump stations, control stations, distribution points, permanent and transfer conveyors, spreaders, reclaimers, stackers, crushing plant, and screens. Conveyors will be 1,400 mm wide and vary in speed from 2.5 m/s to 4.5 m/s.

The operating costs for coal and waste-handling systems are computed on an annual basis. The operating labour is fixed for a given material-handling system design operating on a given shift schedule. The maintenance labour, parts, supplies, and overhead, as well as operating supplies, are all determined as a percentage of the capital cost of the equipment.

Using this method of cost estimation, the operating cost is "step-fixed", varying only as the material-handling system design changes - such as increasing conveyor lengths and adding equipment.

Table 14-10 shows the annual operating cost development for the waste-handling system in Year 8 when the Houth Meadows Dump is being used and the second truck dump station is in operation. Annual operating costs were developed for various system designs incorporated throughout the project life, and these are reflected in the schedule of operating costs on Table 14-11.

The productivity standards shown in Table 14-6 incorporate the factors used for swell, specific gravity, fill factor, and the speed for a 90^{o} swing cycle of a shovel. An efficiency factor of 70% was used in all productivity calculations to reflect the actual productive hours in a shift. 145 minutes of unproductive time comprised:

(1) shift changes - 30 minutes;

(2) lunch break - 30 minutes;

(3) 7 x 50-minute hours, i.e. 10 minutes unproductive time per hour - 70 minutes;

(4) coffee break - 15 minutes.

The machine physical availability is defined as:

$$\frac{\text{Number of shifts the machine is mechanically available (in working condition) per year}}{\text{Scheduled working shifts/year}} \times 100\%$$

The use of availability is defined as:

$$\frac{\text{Working shifts per year}}{\text{Number of shifts the machine is mechanically available per year}} \times 100\%$$

The use of availability of a machine would be dependent on the mine operating schedule, the fleet size of the equipment, and the manpower coverage for the equipment. The effective utilization which gives an overall operating efficiency of a particular piece or type of equipment will be the multiple of the three factors above.

The effective utilization is defined as:

$$\frac{\text{Working hours}}{\text{Scheduled hours}} \times 100\%$$

TABLE 14-4

ANNUAL SALARIES OF MINE STAFF
HAT CREEK PROJECT MINING REPORT: 1979

Position	Base Rate per annum	Payroll Burden 28%	Rate per annum
Mine Manager	56,000	16,000	72,000
Assistant Mine Manager	51,000	14,000	65,000
Superintendent, Mine	42,000	11,800	53,800
Superintendent, Plant and Maintenance	38,500	10,800	49,300
Superintendent, Engineering	38,500	10,800	49,300
Superintendent, Administration	36,000	10,100	46,100
Superintendent, Human Resources	36,000	10,100	46,100
Assistant Mine Superintendent	36,000	10,000	46,000
Assistant Maintenance Superintendent	36,000	10,000	46,000
Chief Accountant	33,000	9,200	42,200
Chief Geologist	33,000	9,200	42,200
Pit Engineer	33,000	9,200	42,200
Superintendent, Environmental and Reclamation	33,000	9,200	42,200
Senior Accountant	31,000	8,700	39,700
Purchasing Agent	31,000	8,700	39,700
Warehouse and Stores Superintendent	31,000	8,700	39,700
Personnel and Labour Relations	31,000	8,700	39,700
Senior Mine Engineer/Geologist	31,000	8,700	39,700
Chief Electrician	31,000	8,700	39,700
Master Mechanic	31,000	8,700	39,700
Environmental Pollution Engineer	31,000	8,700	39,700
Mine Shift Supervisor	31,000	8,700	39,700
Electrical/Mechanical Engineer	31,000	8,700	39,700
Mine Pit Foreman (Shift Boss)	28,500	8,000	36,500
Assay Laboratory Superintendent	28,500	8,000	36,500
Chief Surveyor	28,500	8,000	36,500
Chief Sampler	28,000	8,000	36,500
Safety Supervisor, Training	28,500	8,000	36,500
Public Relations Information	26,000	7,300	33,300
Roads and Pioneer Foreman	26,000	7,300	33,300
Crushing Plant Foreman	26,000	7,300	33,300
Payroll Accountant Cost Supervisor	26,000	7,300	33,300
Secretaries Senior, Confidential	24,000	6,700	30,700
Junior Engineer/Geologist	23,000	6,400	29,400
Chief Security Superintendent	22,000	6,200	28,200
First Aid Officer and Chief Fireman	21,000	5,900	26,900
Senior Technician/Surveying/Geology	27,000	7,600	34,600
Senior Technician Draftsman Design	25,000	7,000	32,000
Surveyor-transitman	23,000	6,400	29,400
Payroll Clerks/Warehouse Clerks	22,000	6,200	28,200
Accounting Clerks	22,000	6,200	28,200
Technicians-draftsmen	21,000	5,900	26,900
First Aid Attendant/Fireman	21,000	5,900	26,900
Training Officers-Assistant	21,000	5,900	26,900
Warehouse Clerks (Shipper)	21,000	5,900	26,900
Security Guards	20,000	5,600	25,600
Samplers	19,400	5,400	24,800
Rodmen-Surveyor helper	19,400	5,400	24,800
Secretaries	18,200	5,100	23,300
Mail Truck Driver	16,000	4,500	20,500
Typists/Stenographers	15,650	4,350	20,000
Timekeeper/Mail Clerk	15,300	4,300	19,600
Keypunch Operator Payroll Clerk	15,600	4,400	20,000

TABLE 14-5

LABOUR RATES FOR MINE OPERATIONS STAFF
(October 1979 Dollars)

Hat Creek Project Mining Report 1979

	Continuous Shift Hourly Rate[1]	Day Shift Hourly Rate[2]
Journeyman Tradesman	14.67	14.16
Shovel Operator Waste Stacker Operator	14.32	13.80
Crane Operator Rotary Driller Coal Stacker and Reclaimer Operator Conveyor Controller	13.98	13.45
Blaster Secondary Crushing Operator Tireman	13.64	13.09
Production Truck Driver Dozer Operator Grader Operator Front-end Loader Operator	13.30	12.75
Shovel Helper Drill Helper Service Truck Driver Airtrac Driller Primary Crushing Operator Lube Serviceman	12.95	12.39
Utility Truck Driver Blaster's Helper Warehouseman Conveyor Patrolman	12.61	12.04
Counterman	12.27	11.69
Pumpman	11.93	11.28
Labourer	11.58	10.98

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

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

TABLE 14-6

PRODUCTIVITY STANDARDS FOR FUNCTIONAL COST CENTRES

100 - DRILLING

Account item: 120 and 130 Drilling Waste Material

Part 1

(1)	Material to be drilled	- waste
(2)	Drill type	- truck-mtd. auger
(3)	Weighted average rate of penetration	- 25 m/h
(4)	Drill pattern	- 9 m x 9 m
(5)	Bench height	- 15 m
(6)	Sub-grade	- 2.4 m
(7)	Effective m^3 material/drilled $\frac{(5) \times (4)}{(6) + (5)}$	- 70 m^3
(8)	Efficiency factor for drilling operation (losses due to shift change, lunch break, 50-min. hour, etc.)	- 70%
	Productive hours per shift (apply to standard cost)	
(9)	No. of productive hrs. per shift (8) x 8	- 5.6
	Productivity in a normal working shift (apply to costing sheet - operating cost)	
(10)	Effective m^3 of material drilled per shift (7) x (3) x (9)	- 9,800 m^3

Part 2

Based on mode of operation, the following factors are developed:

(11)	Scheduled working shifts/machine/year, 1 shift/day, 5 days/week	- 249
(12)	Machine physical availability - $\frac{\text{No. of hrs. machine available for operation}}{\text{Scheduled working hrs. in pit}}$	- 80%
(13)	Use of availability - $\frac{\text{No. of hrs. machine manned \& available for production}}{\text{No. of hrs. machine available for operation}}$	- 90%
	Production from one machine (apply to costing sheet - capital cost)	
(14)	No. of productive shifts per year (11) x (12) x (13)	- 179
(15)	Effective m^3 of material drilled off per year (10) x (14)	- 1,754,000 m^3

Account item: 220 and 230 Blasting Waste Material

Blasting productivity in a normal working shift is equal to drilling productivity.

Productivity in a normal working shift (apply to costing sheet - operating cost)

(1)	Effective m^3 of material blasted per shift (see drilling productivity #10)	- 9,800 m^3
(2)	No. of holes drilled per shift - $\frac{(1)}{\text{Drill pattern x bench height}}$	- 8
(3)	Powder factor	- 0.19 kg/m^3
(4)	Explosives required per hole - Drill pattern x bench height x (3)	- 231 kg
(5)	Explosives required per shift (4) x (2) (apply to standard cost)	- 1,848 kg

Account item: 310 Loading Coal (10.7 m^3 Hydraulic Shovel)

Part 1

(1)	Maximum-rated suspended load - heavy lift circuit	- 37,921 kg
(2)	Bucket weight - bottom dump	- 22,226 kg
(3)	Maximum payload	- 15,695 kg
(4)	Bucket capacity	- 10.7 m^3
(5)	Average in-situ density of material	- 1,490 kg/bank m^3
(6)	Swell % (>100%)	- 135%
(7)	Swell factor $\frac{1}{(6)}$	- 0.74
(8)	Fill factor	- 0.90
(9)	Bucket factor (7) x (8)	- 0.67
(10)	Bank m^3 per cycle (4) x (9)	- 7.2 bank m^3
(11)	Average cycle time	- 32 s
(12)	No. of cycles/hour	- 113
(13)	Theoretical output (10) x (12)	- 814 bank m^3
(14)	Efficiency factor for loading operations (losses due to shift changes, lunch break, 50-min. hour, etc.)	- 70%
	Productive hours per shift (apply to standard cost)	
(15)	No. of productive hours per shift (14) x 8	- 5.6
	Productivity in a normal working shift (apply to costing sheet - operating cost)	
(16)	Bank m^3 of material loaded per shift (13) x (15)	- 4,558 bank m^3 (6,791 t)

Part 2

Based on mode of operation, the following factors are developed:

(17)	Scheduled working shifts/machine/year	- 1,062
(18)	Machine physical availability	- 80%
(19)	Use of availability	- 75%
	Production from one machine (apply to costing sheet - capital cost)	
(20)	No. of productive shifts per year (17) x (18) x (19)	- 637
(21)	Bank m^3 of material loaded per year (16) x (20)	- 2.9 million bank m^3 (4.3 million t)

Note: Check that (10) x (5) does not exceed (3)
7.2 x 1,490 = 10,728 <15,695

Account item: 320 Loading Consolidated Waste Partings ($10.7\ m^3$ Hydraulic Shovel)

Part 1

(1)	Maximum-rated suspended load	- 37,921 kg
(2)	Bucket weight	- 22,226 kg
(3)	Maximum payload	- 15,695 kg
(4)	Bucket capacity	- 8.0 m^3
(5)	Average in-situ density of material	- 2,000 kg/bank m^3
(6)	Swell % (>100%)	- 130%
(7)	Swell factor $\frac{1}{(6)}$	- 0.77
(8)	Fill factor	- 0.90
(9)	Bucket factor (7) x (8)	- 0.69
(10)	Bank m^3 per cycle (4) x (9)	- 5.5 bank m^3
(11)	Average cycle time	- 32 s
(12)	No. of cycles/hour	- 113
(13)	Theoretical output (10) x (12)	- 622 bank m^3/h
(14)	Efficiency factor for loading operations (losses due to shift changes, lunch break, 50-min. hour, etc.)	- 70%
	Productive hours per shift (apply to standard cost)	
(15)	No. of productive hours per shift (14) x 8	- 5.6
	Productivity in a normal working shift (apply to costing sheet - operating cost)	
(16)	Bank m^3 of material loaded per shift (13) x (15)	- 3,483 bank m^3

Part 2

Based on mode of operation, the following factors are developed:

(17)	Scheduled working shifts/machine/year	- 1,062
(18)	Machine physical availability	- 80%
(19)	Use of availability	- 75%
	Production from one machine (apply to costing sheet - capital cost)	
(20)	No. of productive shifts per year (17) x (18) x (19)	- 637
(21)	Bank m^3 of material loaded per year (16) x (20)	- 2.22 million bank m^3

Note: Check that (10) x (5) does not exceed (3)
5.5 x 2,000 = 11,000 <15,695

Account item: 320 and 330 Loading All Waste Materials
(14.5 m^3 Hydraulic Shovel) (except waste partings)

Part 1

(1)	Maximum-rated suspended load	-
(2)	Bucket weight	-
(3)	Maximum payload	-
(4)	Bucket capacity	- 14.5 m^3
(5)	Average in-situ density of material	- 2,000 kg/bank m^3
(6)	Swell % (>100%)	- 125%
(7)	Swell factor $\frac{1}{(6)}$	- 0.8
(8)	Fill factor	- 0.9
(9)	Bucket factor (7) x (8)	- 0.72
(10)	Bank m^3 per cycle (4) x (9)	- 10.4 bank m^3
(11)	Average cycle time	- 35 s
(12)	No. of cycles/hour	- 103
(13)	Theoretical output (10) x (12)	- 1,071 bank m^3/h
(14)	Efficiency factor for loading operations (losses due to shift changes, lunch break, 50-min. hour, etc.)	- 0.70

Productive hours per shift (apply to standard cost)

(15)	No. of productive hours per shift (14) x 8	- 5.6

Productivity in a normal working shift (apply to costing sheet - operating cost)

(16)	Bank m^3 of material loaded per shift (13) x (15)	- 5,998 bank m^3

Part 2

Based on mode of operation, the following factors are developed:

(17)	Scheduled working shifts/machine/year	- 1,062
(18)	Machine physical availability	- 80%
(19)	Use of availability	- 95%

Production from one machine (apply to costing sheet - capital cost)

(20)	No. of productive shifts per year (17) x (18) x (19)	- 807
(21)	Bank m^3 of material loaded per year (16) x (20)	- 4.84 million bank m^3

Note: The 14.5 m^3 standard bucket is suitable for bulk material weights less than 1.8 t/m^3. Bulk material weight of waste is (5) x (7) - 1.6 t/m^3.

Account item: 410 Hauling Coal (Year 6)
Loading Shovel: 10.7 m^3 Hydraulic Shovel

Part 1

(1)	Capacity of trucks (77.3 m^3 struck capacity coal box)	- 77 t
(2)	Material to be handled:	- coal
	(a) average in-situ density of material	- 1,490 kg/bank m^3
	(b) bank m^3 per shovel load (see loading equipment analysis)	- 7.2 bank m^3
(3)	No. of shovel loads/truck load $\frac{(1)}{(2a)} \times \frac{1{,}000}{1} \times \frac{1}{(2b)}$ (nearest lower whole number)	- 7
(4)	Actual capacity per truck load - (3) x (2b)	- 50.4 bank m^3
(5)	Determination of fixed time per cycle:	
	(a) waiting and spotting time	- 30 s
	(b) loading time (30) x cycle time per shovel load (see loading equipment analysis) 7 x 32	- 224 s
	(c) turning and dumping time	- 78 s
	Total fixed time $\frac{a + b + c}{60}$ mins.	- 5.53 mins.
(6)	Average hauling time (Year 6)	- 8.95 mins.
(7)	Average cycle time (5) + (6)	- 14.48 mins.
(8)	Theoretical output (4) x $\frac{60}{(7)}$	- 208.8 bank m^3/h
(9)	Efficiency factor for hauling operations (losses due to shift changes, lunch break, 50-min. hour, etc.)	- 70%
	Productive hours per shift (apply to standard costs)	
(10)	No. of productive hours per shift (9) x 8	- 5.6
	Productivity in a normal working shift (apply to costing sheet - operating cost)	
(11)	Bank m^3 of material hauled per shift (8) x (10)	- 1,169 bank m^3 (1,742 t)

Part 2

Based on mode of operation, the following factors are developed:

(12)	Scheduled working shifts/truck/year	- 1,062
(13)	Machine physical availability	- 73%
(14)	Use of availability	- 95%
	Production from one machine (apply to costing sheet - capital cost)	
(15)	No. of productive shifts per year (12) x (13) x (14)	- 737
(16)	Bank m^3 of material hauled in Year 6 (11) x (15)	- 861,553 bank m^3 (1,283,714 t)

Note: Productive hours per annum - 737 x 5.6 = 4,127

Account item: 420 Hauling Consolidated Waste Partings (Year 6)
Loading Shovel: 10.7 m^3 Hydraulic Shovel

Part 1

(1)	Capacity of trucks (77.3 m^3 struck capacity coal box)	- 77 t
(2)	Material to be handled:	- consolidated waste
	(a) average in-situ density of material	- 2,000 kg/bank m^3
	(b) bank m^3 per shovel load (see loading equipment analysis)	- 5.5 bank m^3
(3)	No. of shovel loads/truck load $\frac{(1)}{(2a)} \times \frac{1{,}000}{1} \times \frac{1}{(2b)}$ (nearest lower whole number)	- 7
(4)	Actual capacity per truck load - (3) x (2b)	- 38.5 bank m^3
(5)	Determination of fixed time per cycle:	
	(a) waiting and spotting time	- 30 s
	(b) loading time (3) x cycle time per shovel load (see loading equipment analysis) 7 x 32	- 224 s
	(c) turning and dumping time	- 78 s
	Total fixed time $\frac{a + b + c}{60}$ mins.	- 5.53 mins.
(6)	Average hauling time (Year 6)	- 8.95 mins.
(7)	Average cycle (5) + (6)	- 14.48 mins.
(8)	Theoretical output (4) x $\frac{60}{(7)}$	- 159.5 bank m^3/h
(9)	Efficiency factor for hauling operations (losses due to shift changes, lunch break, 50-min. hour, etc.)	- 70%
	Productive hours per shift (apply to standard costs)	
(10)	No. of productive hours per shift (9) x 8	- 5.6
	Productivity in a normal working shift (apply to costing sheet - operating cost)	
(11)	Bank m^3 of material hauled per shift (8) x (10)	- 893 bank m^3

Part 2

Based on mode of operation, the following factors are developed:

(12)	Scheduled working shifts/truck/year	- 1,062
(13)	Machine physical availability	- 73%
(14)	Use of availability	- 95%
	Production from one machine (apply to costing sheet - capital cost)	
(15)	No. of productive shifts per year (12) x (13) x (14)	- 737
(16)	Bank m^3 of material hauled in Year 6 (11) x (15)	- 658,141 bank m^3

Account item: 420 and 430 Hauling All Waste Materials (Year 6)
Loading Shovel: 14.5 m^3 Hydraulic Shovel (except waste partings)

Part 1

(1)	Capacity of trucks (90.2 m^3 struck capacity rock box)	- 154 t
(2)	Material to be handled:	- all waste materials (except waste partings)
	(a) average in-situ density of material	- 2,000 kg/bank m^3
	(b) bank m^3 per shovel load (see loading equipment analysis)	- 10.4 bank m^3
(3)	No. of shovel loads/truck load $\frac{(1)}{(2a)} \times \frac{1,000}{1} \times \frac{1}{(2b)}$ (nearest lower whole number)	- 7
(4)	Actual capacity per truck load - (3) x (2b)	- 72.8 bank m^3
(5)	Determination of fixed time per cycle:	
	(a) waiting and spotting time	- 30 s
	(b) loading time (3) x cycle time per shovel load (see loading equipment analysis) 7 x 35	- 245 s
	(c) turning and dumping time	- 83 s
	Total fixed time $\frac{a + b + c}{60}$ mins.	- 5.97
(6)	Average hauling time (Year 6)	- 8.58 mins.
(7)	Average cycle time (5) + (6)	- 14.55 mins.
(8)	Theoretical output (4) x $\frac{60}{(7)}$	- 300.2 bank m^3/h
(9)	Efficiency factor for hauling operations (losses due to shift changes, lunch break, 50-min. hours, etc.)	- 70%
	Productive hours per shift (apply to standard cost)	
(10)	No. of productive hours per shift (9) x 8	- 5.6
	Productivity in a normal working shift (apply to costing sheet - operating cost)	
(11)	Bank m^3 of material hauled per shift (8) x (10)	- 1,681 bank m^3

Part 2

Based on mode of operation, the following factors are developed:

(12)	Scheduled working shifts/truck/year	- 1,062
(13)	Machine physical availability	- 73%
(14)	Use of availability	- 95%
	Production from one machine (apply to costing sheet - capital cost)	
(15)	No. of productive shifts per year (12) x (13) x (14)	- 737
(16)	Bank m^3 of material hauled in Year 6 (11) x (15)	- 1,238,897 bank m^3

TABLE 14-7

STANDARD COSTS FOR FUNCTIONAL COST CENTRES

100 - DRILLING

Account item: 120 and 130 Drilling Waste Material

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	13.45	8.00	107.60	1.32
(b) Helper/Oiler	MH	12.39	8.00	99.12	1.32
Subtotal				206.72	
Operating Supplies					
(a) Fuel	litre	0.17	344.40	58.55	
(b) Wear Parts				17.01	
Subtotal				75.56	
Total Operating				282.28	
Maintenance Labour					
(a) Repair	MH	14.67	4.03	59.12	0.50
(b) Service	MH	12.95	0.50	6.48	0.06
Subtotal				65.60	
Maintenance Parts and Supplies					
(a) Repair	$			104.85	
(b) Tires	$			13.10	
Subtotal				117.95	
Maintenance Overhead					
(a) Staff	$/repair labour hour	2.52	4.03	10.16	
(b) Repair	MH	14.67	0.48	7.04	0.06
(c) Supplies	$			16.19	
Subtotal				33.39	
Total Maintenance				216.94	

Total Operating and Maintenance Cost/Shift: $499.22

Account item: 220 and 230 Blasting Waste Material

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	13.09	8.00	104.72	1.32
(b) Helper/Oiler	MH	12.04	8.00	96.32	
Subtotal				201.04	
Operating Supplies					
(a) Fuel	litre	0.17	65.06	11.06	
(b) Explosives	kg	0.62	1,848	1,145.76	
Subtotal				1,156.82	
Total Operating				1,357.86	
Maintenance Labour					
(a) Repair	MH	14.67	1.74	25.52	0.22
(b) Service	MH	12.95	0.25	3.24	0.03
Subtotal				28.76	
Maintenance Parts and Supplies					
(a) Repair	$			22.12	
(b) Tires	$			6.89	
Subtotal				29.01	
Maintenance Overhead					
(a) Staff	$/repair labour hour	2.52	1.74	4.38	
(b) Repair	MH	14.67	0.21	3.08	0.03
(c) Supplies	$			6.91	
Subtotal				14.37	
Total Maintenance				72.14	

Total Operating and Maintenance Cost/Shift: $1,430.00

Account item: 310 and 320 Loading Coal and Consolidated Waste Partings with 10.7 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	14.32	10.56	151.22	1.32
(b) Helper/Oiler					
Subtotal				151.22	
Operating Supplies					
(a) Wear Parts (teeth, etc.)	$			50.10	
(b) Power	kW	0.02	1656.5	33.13	
Subtotal				83.23	
Total Operating				234.45	(201.32) excl.power
Maintenance Labour					
(a) Repair	MH	14.67	10.08	147.87	1.26
(b) Service	MH	12.95	0.50	6.48	0.06
Subtotal				154.35	
Maintenance Parts and Supplies					
(a) Repairs	$			200.86	
Subtotal				200.86	
Maintenance Overhead					
(a) Staff	$/repair labour hour	2.52	10.08	25.40	
(b) Repair	MH	14.67	1.21	17.75	0.15
(c) Supplies	$			40.31	
Subtotal				83.46	
Total Maintenance				438.67	

Total Operating and Maintenance Cost/Shift: $673.12/$639.99 (excl. power)

Account item: 320 and 330 Loading All Waste Material (except waste partings) 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	14.32	10.56	151.22	1.32
(b) Helper/Oiler	MH	12.95	10.56	136.75	1.32
Subtotal				287.97	
Operating Supplies					
(a) Wear Parts	$			66.31	
(b) Power	kW	0.02	2486.5	49.73	
Subtotal				116.04	
Total Operating				404.01	(354.28) excl. power
Maintenance Labour					
(a) Repair	MH	14.67	11.20	164.30	1.40
(b) Service	MH	12.95	0.50	6.48	0.06
Subtotal				170.78	
Maintenance Parts and Supplies					
(a) Repair	$			295.33	
Subtotal				295.33	
Maintenance Overhead					
(a) Staff	$/repair labour hour	2.52	11.20	28.22	
(b) Repair	MH	14.67	1.34	19.66	0.17
(c) Supplies	$			44.86	
Subtotal				92.74	
Total Maintenance				558.85	

Total Operating and Maintenance Cost/Shift: $962.86/$913.13 (excl. power)

Account item: 410 and 420 Hauling Coal and Consolidated Waste Partings with 77-t Rear Dump Truck

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	13.30	8.00	106.40	1.00
(b) Helper/Oiler					
Subtotal				106.40	
Operating Supplies					
(a) Fuel	litre	0.17	437.60	74.39	
Subtotal				74.39	
Total Operating				180.79	
Maintenance Labour					
(a) Repair	MH	14.67	5.60	82.15	0.70
(b) Service	MH	12.95	0.50	6.48	0.06
Subtotal				88.63	
Maintenance Parts and Supplies					
(a) Repair	$			123.97	
(b) Tires	$			51.52	
Subtotal				175.49	
Maintenance Overhead					
(a) Staff	$/repair labour hour	2.52	5.60	14.11	
(b) Repair	MH	14.67	0.67	9.83	0.08
(c) Supplies	$			22.43	
Subtotal				46.37	
Total Maintenance				310.49	

Total Operating and Maintenance Cost/Shift: $491.28

Account item: 420 and 420 Hauling All Waste Materials (except waste partings) with 154-t Rear Dump Truck

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	13.30	8.00	106.40	1.00
(b) Helper/Oiler					
Subtotal				106.40	
Operating Supplies					
(a) Fuel	litre	0.17	585.00	99.45	
Subtotal				99.45	
Total Operating				205.85	
Maintenance Labour					
(a) Repair	MH	14.67	7.56	110.91	0.95
(b) Service	MH	12.95	0.50	6.48	0.06
Subtotal				117.39	
Maintenance Parts and Supplies					
(a) Repair	$			191.10	
(b) Tires	$			119.62	
Subtotal				310.72	
Maintenance Overhead					
(a) Staff	$/repair labour hour	2.52	7.56	19.05	
(b) Repair	MH	14.67	0.91	13.31	0.11
(c) Supplies	$	4.00		30.24	
Subtotal				62.60	
Total Maintenance				490.71	

Total Operating and Maintenance Cost/Shift: $696.56

14.3 CAPITAL COSTS

The capital cost estimate for the recommended mining scheme was prepared in accordance with the following major cost centres:

Engineering and Construction Costs;

Mine Property Development;

Buildings and Structures;

Mining Equipment;

Coal Conveying, Crushing and Blending Equipment;

Low-grade Coal Beneficiation Equipment;

Waste Disposal Equipment;

Reclamation and Environmental Protection;

Contingency.

Table 14-9 summarizes the project cash flow of capital costs for these nine major cost centres over the five-year pre-production period, 35-year production period, and 10-year post-production reclamation period. The cash flow of mine equipment capital costs was developed according to the system shown in Figure 14-1. The capital costs and service lives of equipment presented in Table 14-3, and the equipment productivity standards presented in Table 14-6, were the main sources of input in developing these cash flows.

Table 14-8 presents sample calculations for the development of capital costs for loading and haulage equipment.

14.3.1 Description of Costs Included in the Major Cost Centres

14.3.1.1 Engineering and Construction (Account Code 90000)

The engineering and construction costs include the capital costs of project management and design, pre-production survey and drilling, and construction costs. Included in the construction costs is the operating cost of a construction ramp at $18.00 per man-day. The construction schedule is shown in Section 12.

14.3.1.2 Mine Property Development (Account Code 91000)

The mine property development costs are the construction costs of the permanent roads in the service area, the mine water supply, sewer and drainage systems, in-pit electrical distribution, fuel distribution station, pit communications (radio and telephone), and the estimated costs of site improvements. Not included in the development estimates are the costs of land and mineral rights purchase.

14.3.1.3 Buildings and Structures (Account Code 92000)

The buildings and structures costs include the capital costs of the following:

(1) Administration and Office Building;

(2) Maintenance shops and Warehouse;

(3) Mine Dry;

(4) Mine service buildings;

(5) Bulk Fuel and Lube Storage;

(6) Equipment and furnishings for the buildings during the project life.

The cost of these buildings are estimated on a unit cost basis per square metre of floor area.

No allowances have been made for a townsite, the construction of off-site housing, or on-site accommodation of mine personnel at any stage of construction or operation.

14.3.1.4 Mining Equipment (Account Code 93000)

The estimated mining equipment costs are the initial and capital replacement costs of all mobile mining equipment, auxiliary and support equipment, and initial spare parts.

14.3.1.5 Coal Conveying, Crushing, and Blending Equipment (Account Code 94000)

The costs in this account include the capital costs of:

- Coal conveyors within the pit;
- Truck dump station equipment;
- Overland coal conveyor;
- Crushing Plant;
- Transfer conveyors, conveyors in the blending area, stackers, reclaimers, transfer car, sampling system, and weigh scale;
- Initial spare parts;
- Construction costs of truck dump stations, overpasses, crusher and transfer houses, conveyor corridors, access roads, and conveyor supports.

14.3.1.6 Low-grade Coal Beneficiation Equipment (Account Code 95000)

The costs in this account include the capital costs of:

- The low-grade coal conveyor within the pit:
- Truck dump station equipment;
- Low-grade coal plant equipment including screens, crushers, bulk density meters, surge bins and reject conveyors;
- Initial spare parts;
- Construction costs of the truck dump station, conveyor corridors and supports.

14.3.1.7 Waste Disposal Equipment (Account Code 96000)

The estimated waste disposal costs are the capital costs of:

- Waste conveyors within the pit;
- Truck dump station equipment;
- Overland waste conveyors;
- Waste dump transfer conveyors, shiftable conveyors, portable belt conveyors, trippers, and spreaders;
- Initial spare parts;
- Construction costs of conveyor foundations and supports, conveyor corridors, service roads, and truck dump stations.

14.3.1.8 Reclamation and Environmental Protection (Account Code 97000)

The costs in this account include the initial capital costs of the reclamation complex, including greenhouses, machinery shed, and equipment for these buildings, and the estimated initial capital and replacement costs of light vehicles, agricultural equipment, laboratory and testing equipment, office equipment, and seed and plant stock.

14.3.1.9 Contingency (Account Code 98000)

The contingency allowance was developed following assessment of the variable risks involved in the major cost centres, as well as consideration of the degree of completeness of cost information, and the labour portion of the cost.

A low risk factor was applied to mobile mining equipment, conveying equipment, and vehicles, since it was considered possible that the preliminary manufacturers' budget quotations for these items could be improved upon at the time of purchase. Higher risk factors were applied to cost centres involving high labour content such as construction work.

The total estimated contingency was based on the following factors:

Category	Contingency Factor
All equipment employed in the mining operations;	10%
Buildings and structures, insurance, and construction costs, and project management and engineering costs.	15%

The contingency allowance provides only for those risks described above, and is not intended to be a provision against unforeseeable risks such as foreign exchange fluctuations on foreign purchases, lengthy industrial disruptions, or events of force majeure of any type.

ACCOUNT CODE	DESCRIPTION	LIFE IN YEARS	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
	PURCHASE AND REPLACEMENT SCHEDULE																								
93100	10.7 m^3 HYDRAULIC COAL SHOVEL	10						1 0 / 1	1 0 / 2		1 0 / 3	1 0 / 4		1 0 / 5				0 1 / 5	0 1 / 5		0 1 / 5	0 1 / 5		0 1 / 5	
93100	14.5 m^3 HYDRAULIC WASTE SHOVEL	10					1 0 / 1		1 0 / 2	1 0 / 3			1 0 / 4				0 1 / 4		0 1 / 4	0 1 / 4					
93200	77 TONNE REAR DUMP COAL TRUCK	8						3 0 / 3	3 0 / 6		2 0 / 8	1 0 / 9	1 0 / 10			0 2 / 9	0 3 / 9		1 2 / 10	0 1 / 10	1 1 / 11			0 2 / 11	0 3 / 11
93200	154 TONNE REAR DUMP WASTE TRUCK	8					3 0 / 3		3 0 / 6	3 0 / 9		3 0 / 12		1 0 / 13	0 3 / 13		0 3 / 13	1 3 / 14		0 3 / 14		0 1 / 14	0 1 / 12		0 3 / 12
	SCHEDULE OF CAPITAL COSTS																								
	CANADIAN $000'S OCTOBER 1979																								
93100	10.7 m^3 HYDRAULIC COAL SHOVEL							1365	1365		1365	1365		1365				1365	1365		1365	1365		1365	
93100	14.5 m^3 HYDRAULIC WASTE SHOVEL						2220		2220	2220			2220				2220		2220	2220					
93200	77 TONNE REAR DUMP COAL TRUCK							1572	1572		1048	524	524			1048	1572		1572	524	1048			1048	1572
93200	154 TONNE REAR DUMP WASTE TRUCK						2328		2328	2328		2328		776	2328		2328	3104		2328		776	776		2328

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD																		TOTAL
		18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	
	PURCHASE AND REPLACEMENT SCHEDULE																			
93100	10.7 m^3 HYDRAULIC COAL SHOVEL			0 1 / 5	0 1 / 5		0 1 / 5			0 1 / 4				0 1 / 4	0 1 / 4					5 11 / 16
93100	14.5 m^3 HYDRAULIC WASTE SHOVEL		0 1 / 3			0 1 / 2							0 1 / 2			0 1 / 2				4 7 / 11
93200	77 TONNE REAR DUMP COAL TRUCK		0 2 / 10		0 1 / 8			0 2 / 8	0 3 / 8			1 0 / 7	0 1 / 7			0 1 / 6				13 24 / 37
93200	154 TONNE REAR DUMP WASTE TRUCK	0 2 / 10		0 3 / 10		0 1 / 10	0 1 / 10		0 1 / 8	0 2 / 8		0 3 / 8		0 1 / 8	0 1 / 8					14 32 / 46
	SCHEDULE OF CAPITAL COSTS																			
	CANADIAN $000'S OCTOBER 1979																			
93100	10.7 m^3 HYDRAULIC COAL SHOVEL			1365	1365		1365			1365				1365	1365					21840
93100	14.5 m^3 HYDRAULIC WASTE SHOVEL		2220			2220							2220			2220				24420
93200	77 TONNE REAR DUMP COAL TRUCK		1048		524			1048	1572			524	524			524				19388
93200	154 TONNE REAR DUMP WASTE TRUCK	1552		2328		776	776		776	1552		2328		776	776					35696

CANADIAN $ 000 OCTOBER, 1979

LEGEND

2 0 / 3

- 2: Purchase of additional piece of equipment
- 0: Replacement of a piece of equipment
- 3: Fleet size

HAT CREEK PROJECT

Table No. 14-8

Development of Capital Costs for Major Mine Equipment

SOURCE: British Columbia Hydro and Power Authority

HAT CREEK PROJECT

Table No. 14-9

Summary of Capital 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
90000	ENGINEERING AND CONSTRUCTION		4892	6391	7959	5538	3975	413											
91000	MINE PROPERTY DEVELOPMENT		450	12592	8171	12652	1267	281	197	316	863	394	435	23	0	935	1001	58	209
92000	BUILDINGS AND STRUCTURES			3480	4507	4162	2669	90	70		128	70		132	123	287	90	70	
93000	MINING EQUIPMENT				200	17985	3889	6255	16587	5205	9179	16328	3771	3930	11401	7924	7174	16115	7911
94000	COAL CONVEYING CRUSHING AND BLENDING				15402	18147	3152	4785	830					3672					
95000	LOW GRADE COAL BENEFICIATION PLANT					1754	3515	1826						2454					
96000	WASTE DISPOSAL EQUIPMENT					7970	16043	3760	13670	837		3323	4435	4909	4015				705
97000	RECLAMATION AND ENVIRONMENTAL PROTECTION			31	82	66	36	14		63	5	24	34	4	25	43	5	4	27
98000	CONTINGENCY	69	802	3370	4664	7945	3849	1781	3150	657	1067	2037	888	1519	1562	979	882	1631	895
	TOTAL CAPITAL COSTS	69	6144	25864	40985	76219	38395	19205	34504	7078	11242	22176	9563	16643	17126	10168	9152	17878	9747
	TOTAL CUMULATIVE CAPITAL COSTS TO FULL PRODUCTION	69	6213	32077	73062	149281	187676	206881	241385	248463	259705	281881	291444	308087	325213	335381	344533	362411	372158

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD																						
		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																							
91000	MINE PROPERTY DEVELOPMENT	2695	517	224	0	70	0	0	631	0	0	0	315	149					121					
92000	BUILDINGS AND STRUCTURES	90	135		132	70	53	412	70		90	70	37	132	70		151	377		90	70		128	22
93000	MINING EQUIPMENT	5282	10585	2265	3796	8986	4325	8065	11837	3927	3721	8835	3133	2548	9761	1959	5265	11524	2688	2839	8535	1595	2023	409
94000	COAL CONVEYING CRUSHING AND BLENDING							5353																
95000	LOW GRADE COAL BENEFICIATION PLANT																							
96000	WASTE DISPOSAL EQUIPMENT	3548	2592		471			3039	930			2854												
97000	RECLAMATION AND ENVIRONMENTAL PROTECTION	49	14	88	5	35	91	5	17	69	14	25	49	22		54	45	19	72	22	5	101	5	30
98000	CONTINGENCY	1306	1417	269	448	924	450	1708	1384	400	387	1182	371	299	986	201	555	1211	294	300	865	170	222	47
	TOTAL CAPITAL COSTS	12970	15260	2846	4852	10085	4919	18582	14869	4396	4212	12966	3905	3150	10817	2214	6016	13131	3175	3251	9475	1866	2378	508
	TOTAL CUMULATIVE CAPITAL COSTS TO FULL PRODUCTION	385128	400388	403234	408086	418171	423090	441672	456541	460937	465149	478115	482020	485170	495987	498201	504217	517348	520523	523774	533249	535115	537493	538001

ACCOUNT CODE	DESCRIPTION	POST PRODUCTION PERIOD										TOTAL
		36	37	38	39	40	41	42	43	44	45	
90000	ENGINEERING AND CONSTRUCTION											29168
91000	MINE PROPERTY DEVELOPMENT											44566
92000	BUILDINGS AND STRUCTURES											18077
93000	MINING EQUIPMENT											257757
94000	COAL CONVEYING CRUSHING AND BLENDING											51341
95000	LOW GRADE COAL BENEFICIATION PLANT											9549
96000	WASTE DISPOSAL EQUIPMENT											73101
97000	RECLAMATION AND ENVIRONMENTAL PROTECTION	85	15	20	32	41	9	21	13			1535
98000	CONTINGENCY	9	2	2	3	4	1	2	1			53167
	TOTAL CAPITAL COSTS	94	17	22	35	45	10	23	14			538261
	TOTAL CUMULATIVE CAPITAL COSTS TO FULL PRODUCTION	538095	538112	538134	538169	538214	538224	538247	538261			538261

CANADIAN $ 000 OCTOBER, 1979

SOURCE: British Columbia Hydro and Power Authority

14.4 OPERATING COSTS

The operating cost estimate for the recommended mining schemes was prepared in accordance with the following major cost centres:

Drilling;

Blasting;

Loading;

Hauling;

Coal-handling System (truck dump stations to powerplant);

Waste-handling System (truck dump stations to waste dumps);

Auxiliary Equipment;

Power;

General Mine Expense;

Overhead;

Royalties;

Contingency;

Contractor's Allowance.

Table 14-11 summarizes the project cash flow of operating costs for these 13 major cost centres over the five-year pre-production period, 35-year production period, and 10-year post-production reclamation period. Pre-production operating costs were not transferred to capital costs. The cash flow of operating costs was developed according to the system shown in Figures 14-1, 14-2, and 14-3. Operating cost estimating criteria shown in Tables 14-4 to 14-7 were the main sources of input in developing these cash flows.

Table 14-10 shows the development of operating costs for the functional cost centres, drilling, blasting, loading, hauling, and waste-handling.

14.4.1 Description of Costs Included in the Major Cost Centres

14.4.1.1 Drilling and Blasting (Account Codes 100 and 200)

The costs in these accounts include the operating costs for drilling and blasting approximately 10% of the waste materials mined from the open pit. Tests on the strength of the waste and coal materials in the Hat Creek Deposit indicate that the hydraulic shovels employed for loading operations will be capable of digging almost all materials with no prior blasting. An allowance has therefore been made only for isolated areas of consolidated waste materials that would require blasting.

14.4.1.2 Loading and Hauling (Account Codes 300 and 400)

The costs in these accounts are the costs of operating the hydraulic shovels and the waste and coal rear dump trucks for the loading and hauling of coal and waste materials to the truck unloading stations. Provision has been made for the continual removal of topsoil by scrapers as the pit expands. In addition, scrapers will remove 2.2 million bank m^3 of overburden in the pre-production period to establish suitable working benches for the hydraulic shovels and trucks.

14.4.1.3 Coal and Waste-handling Systems (Account Codes 500 and 600)

Material-handling costs are the operating costs incurred in conveying waste to the waste dumps and coal material to the powerplant. These costs can be broken into the following components:

1. Conveying Waste

This covers the operating cost of the waste-conveying system and the dump-handling system. The costs of relocating the dump-conveying systems as the dumps are developed are also included.

2. Conveying Coal

This includes the operating costs of the inclined coal conveyor, truck dump stations, the crushing plant, the overland conveyor to the generating station, and the low-grade coal-handling equipment.

3. Coal Stockpiling and Blending

This includes the operating costs of the stackers, reclaimers, conveyors, and clean-up equipment within the stockpiling and blending area.

14.4.1.4 Auxiliary Equipment (Account Code 700)

This includes the operating costs of front-end loaders, dozers, graders, small trucks, compactors, and traxcavators employed in a wide variety of operations, which include embankment construction for the waste dumps, causeway construction, levelling on dumps, pushing and compacting stockpiles, bench pioneering, clean-up work, and assisting loading and hauling equipment. These costs also allow for the initial construction of the Medicine Creek Dump in Years 12, 13, and 14.

14.4.1.5 Power (Account Code 800)

The annual kwh consumption was determined for the shovels and material-handling equipment. The power costs were then developed based on a rate of 20 mills/kW.h.

14.4.1.6 General Mine Expense (Account Code 900)

The general mine expense includes the operating costs of 14 account items which are described below:

1. Pit Dewatering and Drainage

This provides for the installation, operation, and maintenance of the pit dewatering well system and in-pit sumps. Two well systems are required, the first to operate from the pre-production period to Year 15, and the second system to be phased in between Year 10 and Year 15 as the first well system is mined out. Costs allow for repair and replacement of pumps, piping, tankage, well monitoring, and relocation costs of headers and piping as required to support pit development.

2. Electrical Maintenance

This includes the cost of repairs, routine maintenance, and periodic moves of the in-pit overhead power distribution system, power distribution system of the dumps, and dewatering. This account also includes the maintenance of all site electrical services and the handling, repair, and replacement of trailing cables for shovels, conveyors, stackers, and reclaimers.

3. Road Construction and Maintenance

This includes the costs of digging, loading, hauling, placing, and compacting suitable road bases and surface materials, and the additional costs of crushing road surface materials as required for the construction of roads in and around the pit and dumps. These roads are necessary for truck and conveyor access. These costs also allow for snow removal and spreading calcium chloride on all roads.

4. Mine Service Vehicles and Equipment

This includes the costs of operating mine service vehicles such as cranes, service trucks, tire truck, mobile cable reelers, vulcanizing equipment, steam cleaners, lo-boy and hi-boy, Hiab cranes, and forklifts.

5. Field Lubrication/Fuelling

This includes the cost of operating labour and supplies, and repair labour and parts for the operation of the main fuel dump and in-pit fuelling station. Also included are the operating costs of fuel and lube trucks.

6. Reclamation and Environmental Protection

This includes the costs of staff, maintenance of greenhouses and storage buildings, stripping and stockpiling of surface soils, surface regrading, placement of buffer materials and growth media, revegetation and subsequent maintenance.

7. Mine Supervision

This provides for the salaries and expenses of the mine superintendent, shift supervisors, shift foremen, together with stenographic and clerical personnel.

8. Mine Engineering and Geology

This includes the salaries and expenses of the superintendent of mine engineering, pit engineer, planning engineers, senior mine geologist, geologists and technicians, survey supervisor and crews, samplers, coal-quality control technicians, together with stenographic and clerical personnel.

9. Maintenance Engineering

This includes the salaries and expenses of the maintenance superintendent; the design group, consisting of a mechanical engineer, an electrical engineer, and draftsmen; the electrical maintenance supervisors, consisting of the chief electrician, shop foremen, and shift foremen; and the clerical and stenographic staff required in support of these functions. All other costs for maintenance personnel have been allocated to the maintenance costs of the mine equipment or are included in the account item, "Electrical Maintenance".

10. Mine Communications

This includes the salaries of dispatchers and repair technicians, the costs of radio repair parts, and the annual replacement cost of truck and portable radios.

11. Mine Transportation I

This includes the operating and maintenance costs of personnel buses providing daily transport from the dry to the mine.

12. Mine Transportation II

This includes the operating costs of light vehicles assigned to the mine supervisory and technical personnel.

13. Mine Training

This provides for the salaries and expenses of mine training officers, the wages of hourly paid personnel while in training, the costs of training supplies, staff supervisory training and courses.

14. Close-Spaced In-Pit Drilling

This includes the cost of operating drilling equipment for quality-control drilling.

14.4.1.7 Overhead (Account Code 1000)

Overhead costs comprise the operating costs of the following groups:

(1) Management;

(2) Administration;

(3) Administration Services;

(4) Administration Site Services;

(5) Human Resources;

(6) Local Taxes and Insurance.

These six groups are described below.

1. Management Group

This provides for the salaries and expenses of the mine manager, assistant mine manager, supervisor of industrial engineering, industrial and contract engineers, public relations officer, chief chemist, assayers, together with stenographic and secretarial personnel.

2. Administration

This provides for the salaries and expenses of the administration superintendent, accountants, purchasing agents, expeditors, warehouse supervisor, stores foremen, and the clerical and stenographic staff required in support of these functions.

3. Administration Services

This provides for the salaries and expenses of systems analysts, programmers, security officers and supervisors, fire chief and deputy fire chief, janitors, carpenters, painters, and exchange operators. Also included are the operation and maintenance of the fire trucks and fire extinguishing equipment.

4. Administration Site Services

This provides for the costs of operating and maintaining the following site services:

- Mine Dry, including cost of utilities and supplies, maintenance labour and supplies;
- Plant yard, roads, and parking areas;
- Delivery truck operating between the mine site and local communities;
- Garbage truck for garbage removal from mine site and service area;
- Water treatment plant, water distribution system, sewage disposal system, and treatment lagoons.

An allowance for overtime meals is also included in this account.

5. Human Resources

This provides for the salaries and expenses of the superintendent of human resources, the personnel supervisor and officers, labour relations supervisor, safety supervisor, first-aid supervisor and attendants, and the stenographic and clerical personnel associated with these functions. Also included is the cost of operating an ambulance.

6. Local Taxes and Insurance

This provides for the payments of local taxes or grants in lieu of taxes to the municipality and premiums for all-risk insurance. The allowance for local taxes was calculated at 0.5% of fixed assets, and insurance was assessed at an average annual rate of 0.25% of total capital asset value.

14.4.1.8 Royalties (Account Code 1100)

Government regulations stipulate royalty payments as 3½% of the mine-head value. In this preliminary engineering report, royalty payments are estimated at $0.35 per tonne mined.

14.4.1.9 Contingency (Account Code 1200)

In developing the operating costs, the variable risks associated with each of the major cost centres have been accounted for. For example, a higher risk assumption was considered necessary in the hourly paid labour categories, where availability of qualified personnel can vary widely depending on market conditions. Repair labour is also subject to additional risk proportionate to the degree of care exercised in the operation of the equipment.

A contingency allowance of 10% was applied to these operating costs to cater for the potential risks involved in the major cost categories. This contingency is not intended to cover unforeseeable risks such as major labour disruptions or events of force majeure of any kind.

14.4.1.10 Contractor's Allowance (Account Code 1300)

A contractor's allowance of 10% was applied to all operating costs except for the major cost centres, power, and royalties. This allowance provides for contractor's overhead and profit should operation of the mine be contracted. No additional allowance has been made for the additional staff the owner would require for monitoring and control of the contractor.

TABLE 14-10

DEVELOPMENT OF OPERATING COSTS FOR FUNCTIONAL COST CENTRES: 100 - DRILLING

Account item: 120 and 130 Drilling Waste Material

Year	(1) Task Bank m^3 Mat. to be Drilled (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Drilled Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) $ Cost/Shift (fr. Cost Std.)	(5) $ Total Annual Cost ($x10^3$) (3) x (4)
-2					
-1	400	9,800	41	499.22	20
1	400	9,800	41	499.22	20
2	700	9,800	71	499.22	35
3	1,000	9,800	102	499.22	51
4	1,500	9,800	153	499.22	76
5	1,500	9,800	153	499.22	76
6	1,700	9,800	173	499.22	86
7	1,700	9,800	173	499.22	86
8	1,700	9,800	173	499.22	86
9	1,700	9,800	173	499.22	86
10	1,700	9,800	173	499.22	86
11	1,500	9,800	153	499.22	76
12	1,500	9,800	153	499.22	76
13	1,500	9,800	153	499.22	76
14	1,500	9,800	153	499.22	76
15	1,500	9,800	153	499.22	76
16	1,000	9,800	102	499.22	51
17	1,000	9,800	102	499.22	51
18	1,000	9,800	102	499.22	51
19	1,000	9,800	102	499.22	51
20	1,000	9,800	102	499.22	51
21	1,000	9,800	102	499.22	51
22	1,000	9,800	102	499.22	51
23	1,000	9,800	102	499.22	51
24	1,000	9,800	102	499.22	51
25	1,000	9,800	102	499.22	51
26	800	9,800	82	499.22	41
27	800	9,800	82	499.22	41
28	800	9,800	82	499.22	41
29	800	9,800	82	499.22	41
30	800	9,800	82	499.22	41
31	800	9,800	82	499.22	41
32	800	9,800	82	499.22	41
33	800	9,800	82	499.22	41
34	800	9,800	82	499.22	41
35	800	9,800	82	499.22	41
Total	39,500		4,031		2,008

Account item: 200 and 230 Blasting Waste Material

Year	(1) Task Bank m^3 Mat. to be Blasted (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Blasted Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) $ Cost/Shift (fr. Cost Std.)	(5) $ Total Annual Cost ($x10^3$) (3) x (4)
-2					
-1	400	9,800	41	1,430	59
1	400	9,800	41	1,430	59
2	700	9,800	71	1,430	102
3	1,000	9,800	102	1,430	146
4	1,500	9,800	153	1,430	219
5	1,500	9,800	153	1,430	219
6	1,700	9,800	173	1,430	247
7	1,700	9,800	173	1,430	247
8	1,700	9,800	173	1,430	247
9	1,700	9,800	173	1,430	247
10	1,700	9,800	173	1,430	247
11	1,500	9,800	153	1,430	219
12	1,500	9,800	153	1,430	219
13	1,500	9,800	153	1,430	219
14	1,500	9,800	153	1,430	219
15	1,500	9,800	153	1,430	219
16	1,000	9,800	102	1,430	146
17	1,000	9,800	102	1,430	146
18	1,000	9,800	102	1,430	146
19	1,000	9,800	102	1,430	146
20	1,000	9,800	102	1,430	146
21	1,000	9,800	102	1,430	146
22	1,000	9,800	102	1,430	146
23	1,000	9,800	102	1,430	146
24	1,000	9,800	102	1,430	146
25	1,000	9,800	102	1,430	146
26	800	9,800	82	1,430	117
27	800	9,800	82	1,430	117
28	800	9,800	82	1,430	117
29	800	9,800	82	1,430	117
30	800	9,800	82	1,430	117
31	800	9,800	82	1,430	117
32	800	9,800	82	1,430	117
33	800	9,800	82	1,430	117
34	800	9,800	82	1,430	117
35	800	9,800	82	1,430	117
Total	39,500		4,031		5,764

Account item: 310 Loading Coal

Year	(1) Task Bank m^3 Mat. to be Loaded (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Loaded Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) $ Cost/Shift (fr. Cost Std.)	(5) $ Total Annual Cost ($x10^3$) (3) x (4)
-2					
-1	764	4,558	168	639.99	107
1	1,980	4,558	434	639.99	278
2	3,194	4,558	701	639.99	448
3	4,947	4,558	1,085	639.99	694
4	6,207	4,558	1,362	639.99	872
5	7,170	4,558	1,573	639.99	1,007
6	7,015	4,558	1,539	639.99	985
7	7,019	4,558	1,540	639.99	986
8	7,755	4,558	1,701	639.99	1,089
9	7,277	4,558	1,596	639.99	1,022
10	7,498	4,558	1,645	639.99	1,053
11	7,742	4,558	1,699	639.99	1,087
12	7,115	4,558	1,561	639.99	999
13	7,730	4,558	1,696	639.99	1,085
14	7,642	4,558	1,677	639.99	1,073
15	7,437	4,558	1,632	639.99	1,044
16	6,743	4,558	1,479	639.99	947
17	6,856	4,558	1,504	639.99	962
18	7,085	4,558	1,554	639.99	995
19	6,854	4,558	1,504	639.99	962
20	6,685	4,558	1,467	639.99	939
21	6,586	4,558	1,445	639.99	925
22	6,605	4,558	1,449	639.99	927
23	6,654	4,558	1,460	639.99	934
24	6,635	4,558	1,456	639.99	932
25	6,757	4,558	1,482	639.99	949
26	5,560	4,558	1,220	639.99	781
27	5,690	4,558	1,248	639.99	799
28	5,634	4,558	1,236	639.99	791
29	5,746	4,558	1,261	639.99	807
30	5,761	4,558	1,264	639.99	809
31	5,807	4,558	1,274	639.99	815
32	5,710	4,558	1,253	639.99	802
33	5,535	4,558	1,214	639.99	777
34	5,405	4,558	1,186	639.99	759
35	5,115	4,558	1,122	639.99	718
Total	221,915	(330,950 t)	48,687		31,159

Account item: 320 and 330 Loading All Waste Materials (except waste partings)

Year	(1) Task Bank m^3 Mat. to be Loaded (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Loaded Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) $ Cost/Shift (fr. Cost Std.)	(5) $ Total Annual Cost ($x10^3$) (3) x (4)
-2	1,800	5,998	300	913.13	274
-1	2,700	5,998	450	913.13	411
1	4,080	5,998	680	913.13	621
2	6,998	5,998	1,167	913.13	1,065
3	9,983	5,998	1,664	913.13	1,520
4	14,695	5,998	2,450	913.13	2,237
5	15,505	5,998	2,585	913.13	2,360
6	15,775	5,998	2,630	913.13	2,402
7	15,895	5,998	2,650	913.13	2,420
8	15,984	5,998	2,665	913.13	2,433
9	16,914	5,998	2,820	913.13	2,575
10	16,914	5,998	2,820	913.13	2,575
11	17,094	5,998	2,850	913.13	2,602
12	16,555	5,998	2,760	913.13	2,520
13	16,255	5,998	2,710	913.13	2,475
14	14,995	5,998	2,500	913.13	2,283
15	13,795	5,998	2,300	913.13	2,100
16	13,196	5,998	2,200	913.13	2,009
17	11,996	5,998	2,000	913.13	1,826
18	10,796	5,998	1,800	913.13	1,644
19	9,777	5,998	1,630	913.13	1,488
20	9,597	5,998	1,600	913.13	1,461
21	9,277	5,998	1,547	913.13	1,412
22	9,277	5,998	1,547	913.13	1,412
23	8,997	5,998	1,500	913.13	1,370
24	8,997	5,998	1,500	913.13	1,370
25	8,997	5,998	1,500	913.13	1,370
26	8,997	5,998	1,500	913.13	1,370
27	8,997	5,998	1,500	913.13	1,370
28	8,997	5,998	1,500	913.13	1,370
29	9,853	5,998	1,643	913.13	1,500
30	9,855	5,998	1,643	913.13	1,500
31	9,863	5,998	1,644	913.13	1,502
32	9,848	5,998	1,642	913.13	1,499
33	9,822	5,998	1,637	913.13	1,495
34	2,350	5,998	392	913.13	358
35	1,960	5,998	327	913.13	299
Total	397,386		66,253		60,498

Account item: 320 Loading Consolidated Waste Partings

Year	(1) Task Bank m^3 Mat. to be Loaded (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Loaded Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) $ Cost/Shift (fr. Cost Std.)	(5) $ Total Annual Cost ($x10^3$) (3) x (4)
-2					
-1	114	3,483	33	639.99	21
1	295	3,483	85	639.99	54
2	476	3,483	137	639.99	88
3	737	3,483	212	639.99	135
4	925	3,483	266	639.99	170
5	1,068	3,483	307	639.99	196
6	1,045	3,483	300	639.99	192
7	1,046	3,483	300	639.99	192
8	1,156	3,483	332	639.99	212
9	1,084	3,483	311	639.99	199
10	1,117	3,483	321	639.99	205
11	1,154	3,483	331	639.99	212
12	1,060	3,483	304	639.99	195
13	1,152	3,483	331	639.99	212
14	1,139	3,483	327	639.99	209
15	1,108	3,483	318	639.99	204
16	1,005	3,483	288	639.99	185
17	1,022	3,483	293	639.99	188
18	1,056	3,483	303	639.99	194
19	1,021	3,483	293	639.99	188
20	996	3,483	286	639.99	183
21	982	3,483	282	639.99	181
22	984	3,483	283	639.99	181
23	992	3,483	285	639.99	182
24	1,018	3,483	292	639.99	187
25	1,007	3,483	289	639.99	185
26	829	3,483	238	639.99	152
27	848	3,483	243	639.99	156
28	840	3,483	241	639.99	154
29	856	3,483	246	639.99	157
30	858	3,483	246	639.99	158
31	866	3,483	249	639.99	159
32	851	3,483	244	639.99	156
33	825	3,483	237	639.99	152
34	806	3,483	231	639.99	148
35	762	3,483	219	639.99	140
Total	33,100		9,503		6,082

Account item: 410 Hauling Coal

Year	(1) Task Bank m^3 Mat. to be Hauled (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Hauled Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) \$ Cost/Shift (fr. Cost Std.)	(5) \$ Total Annual Cost ($x10^3$) (3) x (4)
-2					
-1	764	1,260	606	491.28	298
1	1,980	1,537	1,288	491.28	633
2	3,194	1,462	2,185	491.28	1,073
3	4,947	1,416	3,494	491.28	1,717
4	6,207	1,371	4,527	491.28	2,224
5	7,170	1,255	5,713	491.28	2,807
6	7,015	1,169	6,001	491.28	2,948
7	7,019	1,220	5,753	491.28	2,826
8	7,755	1,411	5,496	491.28	2,700
9	7,277	1,457	4,995	491.28	2,454
10	7,498	1,356	5,529	491.28	2,716
11	7,742	1,265	6,120	491.28	3,007
12	7,115	1,250	5,692	491.28	2,796
13	7,730	1,230	6,285	491.28	3,088
14	7,642	1,194	6,400	491.28	3,144
15	7,437	1,169	6,362	491.28	3,125
16	6,743	1,124	5,999	491.28	2,947
17	6,856	1,104	6,210	491.28	3,051
18	7,085	1,119	6,332	491.28	3,111
19	6,854	1,169	5,863	491.28	2,880
20	6,685	1,230	5,435	491.28	2,670
21	6,586	1,497	4,399	491.28	2,161
22	6,605	1,351	4,889	491.28	2,402
23	6,654	1,426	4,666	491.28	2,292
24	6,635	1,411	4,702	491.28	2,310
25	6,757	1,512	4,469	491.28	2,196
26	5,560	1,562	3,560	491.28	1,749
27	5,690	1,562	3,643	491.28	1,790
28	5,634	1,361	4,140	491.28	2,034
29	5,746	1,361	4,222	491.28	2,074
30	5,761	1,512	3,810	491.28	1,872
31	5,870	1,562	3,718	491.28	1,827
32	5,710	1,562	3,656	491.28	1,796
33	5,535	1,512	3,661	491.28	1,799
34	5,405	1,512	3,575	491.28	1,756
35	5,115	1,512	3,383	491.28	1,662
Total	221,915 (330,950 t)		166,778		81,935

Account item: 420 Hauling Consolidated Waste Partings

Year	(1) Task Bank m^3 Mat. to be Hauled (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Hauled Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) $ Cost/Shift (fr. Cost Std.)	(5) $ Total Annual Cost ($x10^3$) (3) x (4)
-2					
-1	400	963	118	491.28	58
1	295	1,174	251	491.28	123
2	476	1,117	426	491.28	209
3	737	1,082	681	491.28	335
4	925	1,047	883	491.28	434
5	1,068	959	1,114	491.28	547
6	1,045	893	1,170	491.28	575
7	1,046	932	1,122	491.28	551
8	1,156	1,078	1,078	491.28	527
9	1,084	1,113	974	491.28	478
10	1,117	1,036	1,078	491.28	530
11	1,154	966	1,195	491.28	587
12	1,060	955	1,110	491.28	545
13	1,152	939	1,227	491.28	603
14	1,139	912	1,249	491.28	614
15	1,108	893	1,241	491.28	610
16	1,005	859	1,170	491.28	575
17	1,022	843	1,212	491.28	595
18	1,056	855	1,235	491.28	607
19	1,021	893	1,143	491.28	562
20	996	939	1,061	491.28	521
21	982	1,143	859	491.28	422
22	984	1,032	953	491.28	468
23	992	1,090	910	491.28	447
24	1,018	1,078	944	491.28	464
25	1,007	1,155	872	491.28	428
26	829	1,194	694	491.28	341
27	848	1,194	710	491.28	397
29	856	1,040	823	491.28	404
30	858	1,155	743	491.28	365
31	866	1,194	725	491.28	356
32	851	1,194	713	491.28	350
33	825	1,155	714	491.28	351
34	806	1,155	698	491.28	343
35	762	1,155	660	491.28	324
Total	33,100		32,558		15,995

Account item: 420 and 430 Hauling All Waste Materials (except waste partings)

Year	(1) Task Bank m^3 Mat. to be Hauled (fr. Prod. Sched.) ($x10^3$)	(2) Productivity Bank m^3 Mat. Hauled Per Shift (fr. Prod. Stds.)	(3) No. of Shifts Required (1) ÷ (2)	(4) \$ Cost/Shift (fr. Cost Std.)	(5) \$ Total Annual Cost ($x10^3$) (3) x (4)
-2	1,800	1,820	989	696.56	689
-1	2,700	1,820	1,484	696.56	1,034
1	4,080	2,038	2,002	696.56	1,394
2	6,998	1,966	3,560	696.56	2,480
3	9,983	1,856	5,379	696.56	3,747
4	14,695	1,784	8,237	696.56	5,738
5	15,505	1,747	8,875	696.56	6,182
6	15,775	1,682	9,379	696.56	6,533
7	15,895	1,638	9,704	696.56	6,759
8	15,984	1,733	9,223	696.56	6,424
9	16,914	1,762	9,599	696.56	6,686
10	16,914	1,747	9,682	696.56	6,744
11	17,094	1,784	9,582	696.56	6,674
12	16,555	1,638	10,107	696.56	7,040
13	16,255	1,602	10,147	696.56	7,068
14	14,995	1,602	9,360	696.56	6,520
15	13,795	1,602	8,611	696.56	5,998
16	13,196	1,602	8,237	696.56	5,738
17	11,996	1,602	7,488	696.56	5,216
18	10,796	1,602	6,739	696.56	4,694
19	9,777	1,674	5,841	696.56	4,069
20	9,597	1,602	5,991	696.56	4,173
21	9,277	1,616	5,741	696.56	3,999
22	9,277	1,572	5,901	696.56	4,110
23	8,997	1,784	5,043	696.56	3,513
24	8,997	1,674	5,375	696.56	3,744
25	8,997	1,674	5,375	696.56	3,744
26	8,997	1,638	5,493	696.56	3,826
27	8,997	1,602	5,616	696.56	3,912
28	8,997	1,602	5,612	696.56	3,912
29	9,853	1,602	6,150	696.56	4,284
30	9,855	1,674	5,887	696.56	4,100
31	9,863	1,784	5,529	696.56	3,851
32	9,848	1,769	5,567	696.56	3,878
33	9,822	1,747	5,622	696.56	3,916
34	2,350	1,747	1,345	696.56	937
35	1,960	1,674	1,171	696.56	816
Total	397,386		235,647		164,142

Account item: 621, 622, and 623 Conveying Waste (Year 8)

Operating Labour

Operator	No. Per Shift	Shifts/Day / Days/Week	Total	Yearly Wages Per Operator	Total Annual Labour Cost
*(a) control room	1	3/7	4	$30,532	$81,420
*(b) primary crusher	2	3/7	8	$28,280	$141,400
*(c) conveyor patrol (in-pit)	1	3/7	4	$27,540	$72,454
(d) conveyor patrol (overland and dumps)	3	3/7	12	$27,540	$330,480
(e) spreader	2	3/7	8	$31,275	$250,220
(f) day crew (conv. shifting, etc.)	4	1/5	4	$25,290	$101,160
(g) labourer (clean-up)	2	1/5	2	$25,290	$50,580
			42		$1,028,000

* These operators split 2/3 to waste, 1/3 to coal.

...continued...

Maintenance Costs

Includes maintenance (labour, parts, supplies, and overhead - operating supplies also included). These costs are calculated as a percentage of capital costs.

Year 8:

Equipment	$ x 10^3 Capital Cost	% of Capital Cost	$ x 10^3 Maintenance Cost
In-pit			
Waste Crusher Station	2,019	5	101.0
Waste/Clay Crusher Station	2,842	5	142.1
Waste conveyors	1,848	5	92.5
Waste/clay conveyors	1,848	5	92.5
Surface Plant	780	5	39.0
Waste Dumps			
Overland conveyors	3,951	5	197.6
Transfer conveyors	4,195	5	209.8
Shiftable conveyors	5,086	6	304.2
Portable conveyors	160	5	8.0
Belt trippers	1,284	4	51.4
Spreaders	8,030	3	240.9
			1,480.0

Maintenance Labour

Determined as 50% of total maintenance costs.

$ x 10^3 Maintenance Costs	Maintenance Labour $ x 10^3	Average Yearly Wages Per Mechanic	No. of Mechanics
1,480	740	32,039	23

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
100	DRILLING						20	20	35	51	76	76	86	86	86	86	86	76	76	76	76	76	51	51
200	BLASTING						59	59	102	146	219	219	247	247	247	247	247	219	219	219	219	219	146	146
300	LOADING					274	539	953	1601	2349	3279	3563	3579	3598	3734	3796	3833	3901	3714	3772	3565	3348	3141	2976
400	HAULING					689	1390	2150	3762	5799	8396	9536	10056	10136	9651	9618	9990	10268	10381	10759	10278	9733	9260	8862
500	COAL HANDLING SYSTEM						383	1100	1262	1626	1971	1971	1971	1971	2015	2015	2015	2015	2015	2015	2015	2015	2015	2015
600	WASTE HANDLING SYSTEM							1054	1172	1975	2091	2091	2857	2256	2508	2508	3255	2634	2634	2650	2650	2977	2977	2977
700	AUXILIARY EQUIPMENT					2500	3011	2798	3266	3266	3297	3670	3436	3436	3436	3436	3436	2992	5992	5992	6367	2992	2910	2910
800	POWER					460	2414	3300	3499	3679	3933	3870	4179	4179	4179	4179	4179	3734	4019	4019	4193	4193	4285	4285
900	GENERAL MINE EXPENSE		60	1158	2149	7203	10103	10499	11087	11920	12788	13777	12631	12631	12631	12631	13096	11967	11534	11534	11352	11743	11236	11236
1000	OVERHEAD			815	2460	4517	5831	5991	6167	6200	6233	6307	6355	6453	6517	6548	6576	6622	6651	6714	6769	6778	6793	6818
1100	ROYALTIES						399	1033	1666	2573	3230	3661	3710	3682	4022	3741	3980	3909	3801	4074	3990	3892	3521	3521
1200	CONTINGENCY		6	197	461	1564	2375	2792	3195	3701	4228	4508	4540	4499	4500	4506	4671	4443	4724	4775	4748	4407	4281	4228
1300	CONTRACTOR ALLOWANCE		6	197	461	1431	2024	2342	2709	3195	3693	3972	3968	3917	3912	3915	4077	3889	4138	4183	4134	3792	3655	3599
	TOTAL OPERATING COSTS		72	2367	5531	18638	28548	34091	39523	46480	53434	57221	57615	57091	57438	57226	59441	56669	59898	60782	60356	56165	54271	53624

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD																	
		18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35
100	DRILLING	51	51	51	51	51	51	51	51	41	41	41	41	41	41	41	41	41	41
200	BLASTING	146	146	146	146	146	146	146	146	117	117	117	117	117	117	117	117	117	117
300	LOADING	2833	2638	2583	2518	2520	2486	2489	2504	2303	2325	2315	2464	2467	2476	2457	2424	1265	1157
400	HAULING	8412	7511	7364	6582	6980	6252	6518	6368	5916	6051	6343	6762	6337	6034	6024	6066	3036	2802
500	COAL HANDLING SYSTEM	2015	2015	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150	2150
600	WASTE HANDLING SYSTEM	2321	2525	2360	2360	2360	2843	2451	2451	2724	2436	2436	2436	2436	2436	2436	2436	2436	2436
700	AUXILIARY EQUIPMENT	2910	2584	2584	2450	2450	2450	2450	2450	2450	2450	2450	2168	2168	2155	2138	2138	2076	2076
800	POWER	4285	4285	4143	4335	4335	4335	4335	4335	3789	3546	3474	3474	3474	3474	3474	3474	3474	3474
900	GENERAL MINE EXPENSE	11236	11236	11306	10574	10574	10574	10574	10644	10393	10393	10393	10152	10152	10186	10186	10138	9673	9673
1000	OVERHEAD	6831	6922	6967	6978	6988	7036	7043	7056	7083	7089	7105	7140	7149	7157	7181	7185	7192	7194
1100	ROYALTIES	3734	3552	3465	3381	3350	3640	3510	3440	2803	3216	2936	2992	3003	3024	3014	2888	2817	2667
1200	CONTINGENCY	4104	3991	3965	3814	3855	3832	3821	3816	3697	3660	3682	3690	3649	3623	3620	3617	3146	3112
1300	CONTRACTOR ALLOWANCE	3474	3352	3336	3164	3205	3177	3164	3158	3091	3078	3106	3110	3068	3040	3036	3032	2561	2527
	TOTAL OPERATING COSTS	52352	50808	50420	48503	48964	48972	48707	48569	46557	46552	46548	46696	46211	45913	45874	45706	39984	39426

ACCOUNT CODE	DESCRIPTION	POST PRODUCTION PERIOD										TOTAL
		36	37	38	39	40	41	42	43	44	45	
100	DRILLING											2008
200	BLASTING											5764
300	LOADING											97739
400	HAULING											262072
500	COAL HANDLING SYSTEM											70835
600	WASTE HANDLING SYSTEM											85585
700	AUXILIARY EQUIPMENT											111740
800	POWER											140293
900	GENERAL MINE EXPENSE	2014	2014	2014	1745	1745	403	403	241	52	52	423706
1000	OVERHEAD											251416
1100	ROYALTIES											115837
1200	CONTINGENCY	201	201	201	175	175	40	40	24	5	5	145110
1300	CONTRACTOR ALLOWANCE	201	201	201	175	175	40	40	24	5	5	123955
	TOTAL OPERATING COSTS	2416	2416	2416	2095	2095	483	483	289	62	62	1836060

CANADIAN $ 000 OCTOBER, 1979

HAT CREEK PROJECT

Table No. 14-11

Summary of Operating Costs by Major Cost Centres

SOURCE: British Columbia Hydro and Power Authority

14.5 FINANCIAL ANALYSIS

14.5.1 Approach

Before carrying out the financial analysis all costs associated with the mining of Hat Creek coal were compiled. In addition to the costs described in the previous sections, costs were allocated to the mining project for the off-site facilities (Hat Creek Diversion, construction camp, and discretionary expenses), land, and ongoing studies associated with the project. These costs are shown in Table 14-12.

The objective of this analysis is to determine the price in 1979 dollars of Hat Creek coal delivered to the powerplant. At this price the mining venture must yield a rate of return equal to B.C. Hydro's cost of capital. The price of Hat Creek coal delivered to the powerplant could then be compared with any alternative fuel with an equivalent energy content.

14.5.2 Parameters

The financial parameters used in this analysis are as follows:

(1) The base date for economic calculations is October 1979 and the capital and operating cash flows are in 1979 dollars;

(2) Inflation rates

The following inflation rates were applied to the cash flows:

Fiscal Year	Rate %
1979-1980	8.50
1980-1981	7.75
1981-1982	7.50
1982-1983	7.25

Fiscal Year	Rate %
1983-1984	7.00
after 1984-1985	6.00

(3) Plant operating date

It is assumed that the first unit commences operation in the fiscal year 1989-1990;

(4) Debt:Equity ratio

The financial structure is 100% debt;

(5) Rate of return

A rate of return of 9% is required to cover interest only. No operating profit assumed;

(6) Income tax

No income tax is paid by B.C. Hydro;

(7) Provincial royalty

In the financial analysis, the royalties were calculated as 3.5% of the total capital and operating costs for the project;

(8) Corporate overhead

Corporate overhead has been calculated at 9% of the total capital cost incurred;

(9) Construction Insurance and Bonds

Construction insurance and bonds were calculated at 0.5% of the total capital costs plus 0.044% of the total operating costs. These costs were added to the total costs of the project.

14.5.3 Analysis

The present values of the cash outflows and inflows associated with the project were equated. The cash outflows are the annual capital and operating cash requirements associated with the mining operations and the cash inflows are determined from the schedule of coal to be supplied to the powerplant and the price of the delivered coal.

This analysis incorporated the inflation rates stipulated above, a discount rate of 9% and a time horizon of 54 years since expenditures are incurred nine years prior to the start of production, and reclamation activities continue for a 10-year period after mining ceases.

14.5.4 Conclusion

Based on the above financial parameters, the price of Hat Creek coal delivered to the powerplant is $0.60/GJ in 1979 dollars, which is equivalent to $8.27 per tonne of coal with an average heating value of 18 MJ/kg, dry basis. If the cost of power for the mining operations is excluded the price of coal is reduced to $0.57/GJ ($7.80 per tonne of coal).

TABLE 14-12

OTHER COSTS ALLOCATED TO MINING PROJECT

Fiscal Year 1989 in Service	-9 80/81	-8 81/82	-7 82/83	-6 83/84	-5 84/85	-4 85/86	-3 86/87	-2 87/88	-1 88/89	1 89/90	2 90/91	3 91/92	Totals
Ongoing Studies	555	490	445	330	330	180	205	205	205	50	2	2	2,999
Off-site Facilities:													
Creek Diversion				380	540	5,680	8,070						14,670
Construction Camp			10	72	2,056	918	375	1,652	812	35	10		5,940
Discretionary Expenses		666	1,334	1,867	5,392	4,525	1,067	333	333	333	300		16,150
Total Off-site Facilities	0	666	1,344	2,319	7,988	11,123	9,512	1,985	1,145	368	310	0	36,760
Land	910	410	70	70	70	70	70	70	70				1,810
Grand Totals	1,465	1,566	1,859	2,719	8,388	11,373	9,787	2,260	1,420	418	312	2	41,569

Canadian $000's October 1979

SECTION 15

MINING REPORTS AND STUDIES

1 Duffell, S. and K.C. McTaggart, *Ashcroft Map - Area, B.C.*, Memoir 262, p. 122, Geological Survey of Canada (including Map No. 1010A), 1951.

2 McCullough, P.T., *Hat Creek Magnetometer Survey*, B.C. Hydro File, 1975.

3 Rouse, G.E., *Palynological Zonation and Correlation of Hat Creek Core Samples*, 1976.

4 PD-NCB/Wright/Golder, *Preliminary Report on Hat Creek Openpit No. 1*, Volumes 1 and 2, March 1976 (Report No. 2).

5 PD-NCB/Wright/Golder, *Preliminary Report on Hat Creek Openpit No. 2*, Volumes 1 and 2, March 1976 (Report No. 3).

6 Birtley, *Results of Washability and Plant Washing of Samples from A, B and C - the Hat Creek Deposit*, June 1976.

7 BCH Thermal Division, *Final Report - Bulk Sample Program - Hat Creek No. 1 Deposit*, 1977.

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