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

HAT CREEK PROJECT

MINING REPORT — Summary

DECEMBER, 1979.

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604H - M084.

Excavating Trench 'A'
by Hydraulic Shovel

1977 Bulk Sample Program

HAT CREEK PROJECT

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SECTION 1

INTRODUCTION

The objective of the 1979 mining work program was to conclude the considerable study work that has been undertaken on the Hat Creek Project, and to document the results in sufficient detail to support project approval and licence applications. This Mining Summary Report, supported by the 1979 Mining Report, a comprehensive technical report, marks the achievement of this objective.

This Summary Report is primarily concerned with the mining aspects of the project; but the key interfaces with the powerplant are fully recognized and incorporated. This report presents the key results of the 1979 studies, features a description of how the mine works, and includes a more detailed section on nine major factors that influence the mine design. A practical and economic mining plan is presented for the production of fuel for the 35-year life of the base scheme 2,000 MW powerplant. This plan would extract less than half the coal in the No. 1 Deposit.

The cornerstone of the 1979 work program was the complete re-evaluation of the geological structure and the distribution of coal quality in the deposit, incorporating the results of the 1978 drilling program. The additional drilling information, and significant improvement in the interpretation of geophysical data, has resulted in a high degree of confidence in the geological interpretation of the deposit. This provided a solid foundation for the application of improved planning techniques and mining methods to develop a new mining plan and production schedule, which produce a better quality boiler fuel. The boiler fuel specification was evaluated and revised by a specialist consultant and is considered a sound basis for boiler design.

The work completed demonstrates an effective mining system for the development of the complex Hat Creek No. 1 Deposit. This represents the first phase of a process that will continue throughout the life of the mine.

The members of the Mining Department of B.C. Hydro's Thermal Division have found this assignment challenging and interesting. A wealth of experience and knowledge of the coal deposit has been accumulated, which can be applied to any development of the immense resource of Hat Creek.

Team work with other members of the project team has been rewarding and the value of their technical contribution is acknowledged. Valuable contributions have also been made by the representatives of the various consulting firms who have shared in the work. Their expertise has greatly enhanced the Hat Creek Project engineering studies.

2 SUMMARY

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SECTION 2

SUMMARY

2.1 RESULTS OF 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 1979 Mining 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 cubic metres 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 cubic metres to under 7 million cubic metres;

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

(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 equipment 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.

2.2 PREVIOUS MINING STUDIES

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 metres in length have been drilled. 206 of these holes, on a 150 metres by 150 metres grid-pattern totalling 54,000 metres, have been drilled in the No. 1 Deposit. A further 19,800 metres 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°, which can in some areas rise to 25°, 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 cubic metres of waste. The proposed open-pit would cover an area 3 kilometres by 2.5 kilometres and be 265 metres 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 kilometres away and 500 metres 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 tonnes 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-tonne sample. This indicated that coal-washing (beneficiation) was practical, though not justified at present for Hat Creek coal on technical and economic grounds.

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 subconsultants: 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 THE MINING PLAN

All the figures in this Summary Report were originally numbered for the Main Report. These numbers could not be changed in the time available. The figures have therefore been placed at the end of this section.

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SECTION 3

THE MINING PLAN

3.1 INTRODUCTION

This mining plan has been developed to provide a reliable supply of coal of consistent quality to meet the forecast requirements of a 2,000 MW mine-mouth powerplant for 35 years. The coal will be produced by open-pit mining methods from the Hat Creek No. 1 Deposit and delivered to the powerplant 4 kilometres away located on a hill 500 metres above the elevation of the pit-head facilities.

In developing the mining plan, design features have been incorporated to:

(1) Ensure the safety of the work force;

(2) Meet environmental objectives and minimize adverse impacts;

(3) Ensure maximum utilization of the resource.

In addition, security of supply and flexible, reliable operation were important considerations.

The plan, which spans a period of 50 years: five years of pre-production development, 35 years of operation followed by 10 years of reclamation, provides for mining 331 million tonnes of coal and 427 million cubic metres of waste. The product delivered to the powerplant will be a consistent, blended fuel with a heating value of 18.0 MJ/kg, 33.5% ash, and 0.51% sulphur (all on a dry-coal basis), with a 23.5% moisture content.

The mining method using hydraulic shovels, rear-dump diesel electric trucks, and conveyors was selected after a thorough review of several alternatives. The best alternative method was the use of a bucketwheel excavator-conveyor system, which was ultimately rejected because of the limited size and the shape of the pit in the early years of operation.

In the selected method, waste material is excavated using 14.5 cubic metre hydraulic shovels and loaded into 154-tonne trucks. The coal is selectively mined to reject waste partings greater than 2 metres thick, using 10.7 cubic metre hydraulic shovels loading 77-tonne trucks. Both coal and waste materials are hauled to loading stations at a central ramp for crushing and conveyor transport to surface. Three loading stations will be constructed at different elevations as the mine is progressively deepened (Figure 3-3). At the surface, the waste material is conveyed to one of two waste dumps: Houth Meadows or Medicine Creek, and placed using crawler-mounted spreaders. The coal is crushed further and conveyed to a blending stockpile, from which it is later reclaimed and moved to the powerplant by an overland conveyor.

3.2 PRE-PRODUCTION DEVELOPMENT

Pre-production development is scheduled to start in Year -5. (Note: all years referred to in this report are related to the start of commercial power production for the first 500 MW unit, being the start of Year 1. Hence, pre-production starts in the fifth year before start-up.) The initial field work will be to start the construction of the Hat Creek and Finney Creek Diversion which must be completed before any significant mining excavation can take place.

The planned diversion of Hat Creek consists of a headworks 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.

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

Site preparation and construction of the mine support and surface facilities will take place between Year -4 and Year -1. These facilities include the screening and crushing plant, the lagoons, the blending stockpile area, the overland conveyor, and the necessary maintenance shops, warehouse, and offices. The facilities will be constructed to the North of the deposit adjacent to the main access to the mine (see Figure 10-1).

During the pre-production period the mine must be prepared for production and must also produce sufficient coal for boiler testing and building the initial stockpiles.

Mine development starts in Year -2 with excavation for the first loading station and to open up the pit. The pit development begins close to the centre of the deposit in the valley bottom where the coal comes to within 10 metres to 15 metres of the surface. The waste materials excavated during this phase are used to build the roads from the initial pit to Houth Meadows Waste Dump 2.5 kilometres away and the

1.5 kilometres to the No. 1 Loading Station. When these tasks are completed, the materials will be used to build the waste conveyor ramp to Houth Meadows and the embankment base.

The development work will be started using tandem scrapers and 32-tonne trucks loaded by front-end loaders. As benches are developed, the first production shovels and trucks are phased in.

3.3 MINE OPERATION

3.3.1 Pit Planning

During the pre-production period, six mining benches will be developed. Early coal production will be concentrated along the Eastern limb of the main syncline, where the attitude of the coal seams exposes several different qualities of coal in a limited area. This permits the blending of a uniform quality of fuel for the powerplant from the start of commercial production.

As the mine moves towards full production additional benches will be developed, both by deepening the pit in coal, and widening it in waste by stepping up the hill from the valley bottom. By Year 5, ten benches will be available for operation, and in peak years around Year 15 as many as 15 benches will be in production. At this stage, the pit bottom is at 805 metres elevation, or approximately 100 metres deep. Figure 5-1 illustrates in cross-section how the pit is developed in increments through to the end of the 35-year life, when the pit will reach a depth of 235 metres.

Figure 5-1 also illustrates the flat working slopes (16^{o}) that are used in developing the mine. These slopes are significantly flatter than the final design slopes which range up to 25^{o}. This will help to ensure that the mine can develop safely until sufficient experience with the weak wall rocks can be gained to confirm the final slope angles. The pit is roughly elliptical and expands progressively to the final wall location which is not reached until the latter stages of the mine life.

The mining benches are planned for a 15-metre height. Should stability problems arise locally, the selected mining equipment is flexible enough to adjust to operating at a lower height, although some additional road building costs would be incurred. Benches in coal are laid out to permit mining of a 20-metre-wide cut at any time without the necessity of mining the bench above first. This approach improves the flexibility of short-term scheduling for quality control.

Figures 5-14 and 5-18 show the development of the pit at the end of Years 5 and 35, respectively. Plans for additional stages in the pit life have been developed and are included in the 1979 Mining Report. These figures also illustrate the development of haul roads in the pit. The road network is designed to provide access to a minimum of two locations on each bench, usually on opposite sides of the pit. This serves to reduce truck haulage distance, and also to provide better assurance of continuity of operation in the event of localized wall failures.

In developing the plan of operations, particular attention has been paid to the construction of truck haul roads. This is a crucial area in planning any trucking operation because of its impact on truck operating costs. It is even more important at Hat Creek because of the widespread occurrence of weak claystones and slide materials. During the 35-year pit life, it will be necessary to construct over 800 kilometres of haul roads. A fleet of 32-tonne trucks and scrapers have been budgeted for this work.

3.3.2 Mining Methods

Mining of both waste and coal will be performed using hydraulic shovels to load diesel-electric trucks. Experience gained during the Bulk Sample Program in 1977 indicated that this equipment could excavate most of the materials encountered in the Hat Creek Deposit without blasting. The operation has been planned on this basis, although provision has been made in the estimates for a nominal 10% of the waste material to be drilled and blasted.

The equipment system selected for waste removal is 14.5 cubic metre hydraulic shovels loading 154-tonne trucks. The selection of equipment is discussed in more detail in Section 4.5 of this report. In the initial years of operation, most of the waste to be removed is above the valley elevation, which gives the trucks a long downhill haul to Dump Station No. 1 for transport to the waste dump. The mining of the different types of waste material will be scheduled to match the requirements for dump construction. It is anticipated that excavation in the active slide areas will be more practical during the Winter, when freezing conditions will help to stabilize the material and improve its ability to support heavy equipment.

In order to produce the best quality of powerplant fuel possible from the deposit, a selective mining technique was adopted. The way the coal is deposited with bands of clay and low-grade coal interbedded, lowers the overall quality of the coal. The separate removal of these partings down to a practical minimum thickness is selective mining. For this study a minimum parting thickness of 2 metres was selected. With operating experience it will be possible in many cases to remove partings down to 1-1½ metres thick. The converse applies in predominantly waste areas where coal bands 2 metres thick can also be recovered.

In order to produce a consistent quality of fuel from a variable deposit, it is essential to be able to mine coals of differing qualities at all times, which imposes a need for flexibility in the operation. This flexibility is achieved by using more small equipment rather than fewer large machines.

These considerations led to the selection of 10.7 cubic metre hydraulic shovels for excavation, matched with 77-tonne trucks.

The coal-mining operation will produce four different products from the coal zone:

(1) Performance coal of consistent quality;

(2) High-grade, low sulphur coal;

(3) Low-grade coal;

(4) Waste parting material.

These separate operations must be carefully planned and controlled at the coal face and subsequently through the Materials-handling System, which has been designed to deal with these materials.

During the initial years of operation, coal zone materials will be hauled to Dump Station No. 1. As the pit deepens, the In-pit Conveyor System is extended and additional dumping stations are constructed when the truck haulage distances become uneconomic. Additional dumping stations are scheduled for operation in Years 8 and 20.

3.3.3 Materials-handling

The Materials-handling System includes all the equipment and facilities required to process and transport all of the project coal and waste materials from the dumping stations at the central in-pit conveyor location to their final destination. The system includes all the crushing, conveying, stacking, and reclaiming equipment required to deliver fuel to the boilers, build the waste dumps, dispose of the powerplant ash, and provide necessary surge capacity between the mine and the powerplant to permit continuous, efficient operation. Figures 8-1 and 8-2 show the details of the project Materials-handling System.

During pre-production the first flight of four parallel, in-pit conveyors, together with the first truck dump station which includes the primary crushing pockets, will be installed. As the pit deepens and expands, this in-pit system is extended in length by adding additional flights and installing two additional dump stations. Figure 8-3 shows the layout of the separate in-pit conveyors for coal, low-grade coal, construction and clay waste materials, and identifies the dump pockets at the dump stations.

All mine coal and waste materials will be delivered to the truck dump station in trucks where they will be crushed to -200 millimetres. Uncrushable materials screened off or rejected by the crushers will be disposed of by front-end loaders and trucks. The -200 millimetre material will then be transferred to the appropriate 1,400 millimetre-wide conveyor for delivery to the surface, where the materials are routed to their separate destinations.

The coal will be delivered to the crushing and screening plant from a set of four surge bins by four separate conveyor lines, each of 1,000 tonnes per hour capacity. In the plant, coal will be sized to -50 millimetres by impact crushers for delivery to the blending stockpiles on a conveying system which would be twinned to assure reliability. Automatic samplers installed on these conveyors monitor the quality of coal going to the blending system. The blending/stockpile system, as shown on Figure 8-5, will use 300,000 tonne-capacity stockpiles to produce the average quality coal - this size being about seven days' supply for the powerplant at full load conditions. Normally, one pile is being built while the other is reclaimed. Stockpiling and blending will be at a peak rate of 3,200 tonnes per hour, using conveyors and a rail-mounted slewing and luffing stacker. Blending will be achieved by building the pile in a series of windrows. After building one pile, the stacker will slew through 180° and be ready to construct the other pile.

Coal will be reclaimed at a nominal maximum rate of 2,500 tonnes per hour (peak 3,000 tonnes per hour), using a rail-mounted bucketwheel reclaimer, which operates across the face of the pile, i.e. at 90° to the axis of construction. Coal will be transferred to the reclaim conveyors which feed the overland coal conveyor. After reclaiming one pile, a transporter car will relocate the reclaimer to the other pile. The 1,400 millimetre-wide overland coal conveyor, as shown on

Figure 8-6, approximately 4 kilometres long, will deliver the coal to the powerplant some 500 metres higher in elevation than the blending system. An automatic sampler will monitor the quality of coal being delivered to the powerplant. Normal operations require all the coal from the mine to pass through the blending piles. A bypass arrangement will allow delivery of low-sulphur coal direct to the powerplant to meet MCS conditions and to replenish low-sulphur coal stockpiles at the powerplant.

Low-grade coal will also be crushed to -200 millimetres in the in-pit primary crushers and be fed to the low-grade coal conveyor for delivery to the low-grade coal dry beneficiation plant via a pair of surge bins. The material will be delivered to the plant by two conveyor lines, each of 500 tonnes per hour capacity, for screening and crushing. Different size fractions will be monitored for ash content by bulk density meters and routed either to the blending system or to the waste disposal system, depending upon the ash value.

The low-grade coal conveyor is sized to permit its use as a backup to either the main coal conveyor or the construction waste conveyor. Using the low-grade coal system, it is also possible to mine and deliver low-sulphur coal to the powerplant while the main coal system is delivering regular coal to the blending stockpile.

The essential link between the mine and the powerplant is provided by a single overland conveyor with surge capacity at each end. A study of the overall reliability of the system confirmed the selection of the single overland conveyor, using the blending stockpiles as surge at the mine end and the provision of a 14-day compacted dead storage and a 2½-day live storage at the powerplant. The 14-day storage provides sufficient coal to maintain power generation at full capacity during the longest anticipated conveyor breakdown. Space is available to extend the dead storage capacity to 30 days.

The powerplant Coal-handling System shown on Figure 8-9 will normally deliver the coal from a 600-tonne surge bin at the end of the overland conveyor direct to the boiler silos by twinned conveyor lines through 100-tonne surge bins in the Powerhouse. Coal in excess of immediate requirements will be diverted to storage. Stacking out and reclaiming of the live storage will be by conveyors and a bucketwheel stacker/reclaimer, with dead storage being built and reclaimed by mobile equipment. To prevent spontaneous combustion in coal stockpiles, it will be necessary to ensure that "live" piles are consumed within four weeks. Piles with a longer residence time, i.e. emergency piles, must be compacted to reduce the chances of combustion.

Waste materials will be delivered to the truck dump stations by 154-tonne trucks at a peak rate of 5,000 tonnes per hour.

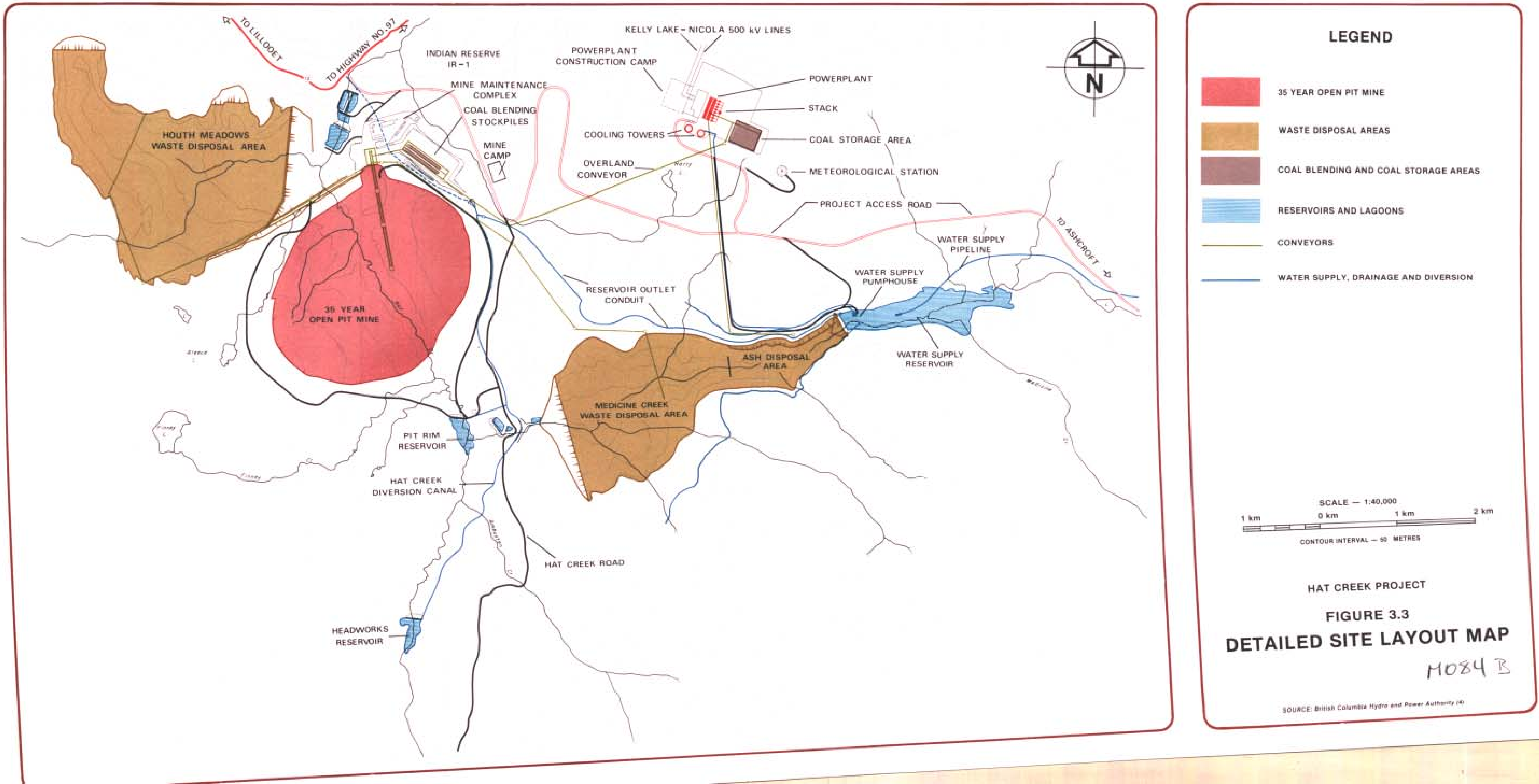
Construction-waste materials will be transferred to the in-pit waste conveyor for delivery to a pair of waste bins at the surface. Figure 8-2 shows the flow of materials. Truck-loading chutes will be included to provide materials for construction purposes.

Waste/clay materials will be routed to the waste/clay conveyor and delivered to the surface for direct transfer to the waste disposal systems. For the potentially wet clays, a separate facility will be provided to eliminate the need to pass these problem materials through a crusher. Special consideration has been given to chutes and transfer points to reduce potential problems in handling these materials.

Both types of waste materials will be routed either to Houth Meadows or to Medicine Creek (after Year 14) for disposal in the waste dumps. The waste dumps have been designed to provide long-term safety and stability. To ensure this, engineered retaining embankments will be constructed to contain the weak claystone and slide materials. The two dumps have the capacity to store all the waste from the 35-year pit and a significant additional quantity.

Two disposal systems will be installed initially in Houth Meadows. Each system will consist of overland, transfer, and shiftable conveyors, a belt tripper, and a spreader which deposits the materials in 35-metre lifts. As shown on Figure 8-7, the spreader dumps a lift of 20 metres below and ahead of, and a lift 15 metres above and behind, the shiftable conveyor. After Year 14, one of the systems will be relocated to Medicine Creek. Powerplant ash will also be disposed of in Medicine Creek. A twin conveyor system from the powerplant transports this material to a spreading system. Detailed dump development sequences have been prepared to ensure the feasibility of the scheme.

The planned materials-handling systems have been designed with sufficient spare capacity and redundant equipment to provide for flexible and reliable operation, especially in supplying coal to the powerplant. Before the final design of the equipment, further testing of the crushing and material flow characteristics will be required.



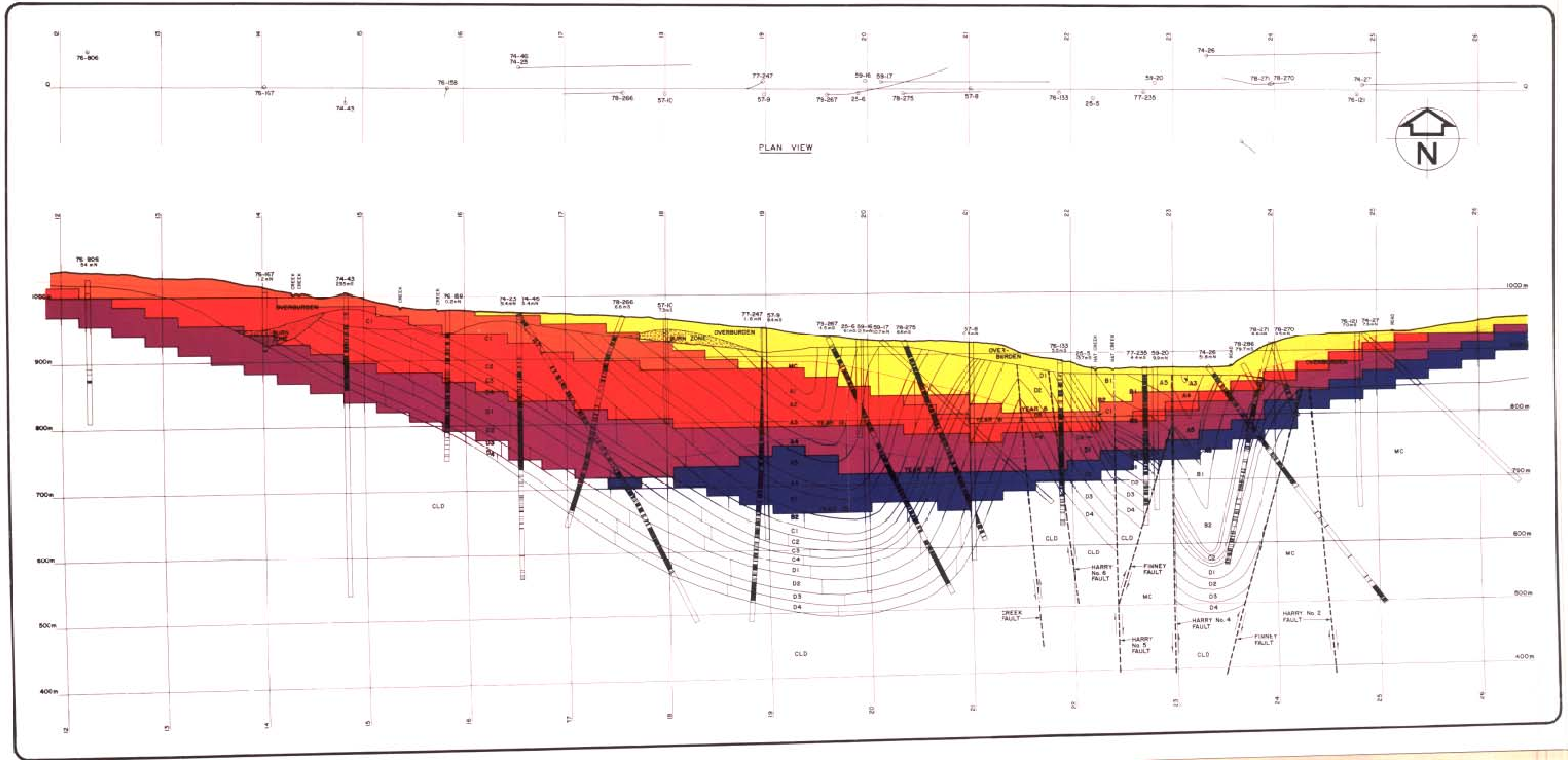
HAT CREEK PROJECT

FIGURE 3.3

DETAILED SITE LAYOUT MAP

M084 B

SOURCE: British Columbia Hydro and Power Authority (4)



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

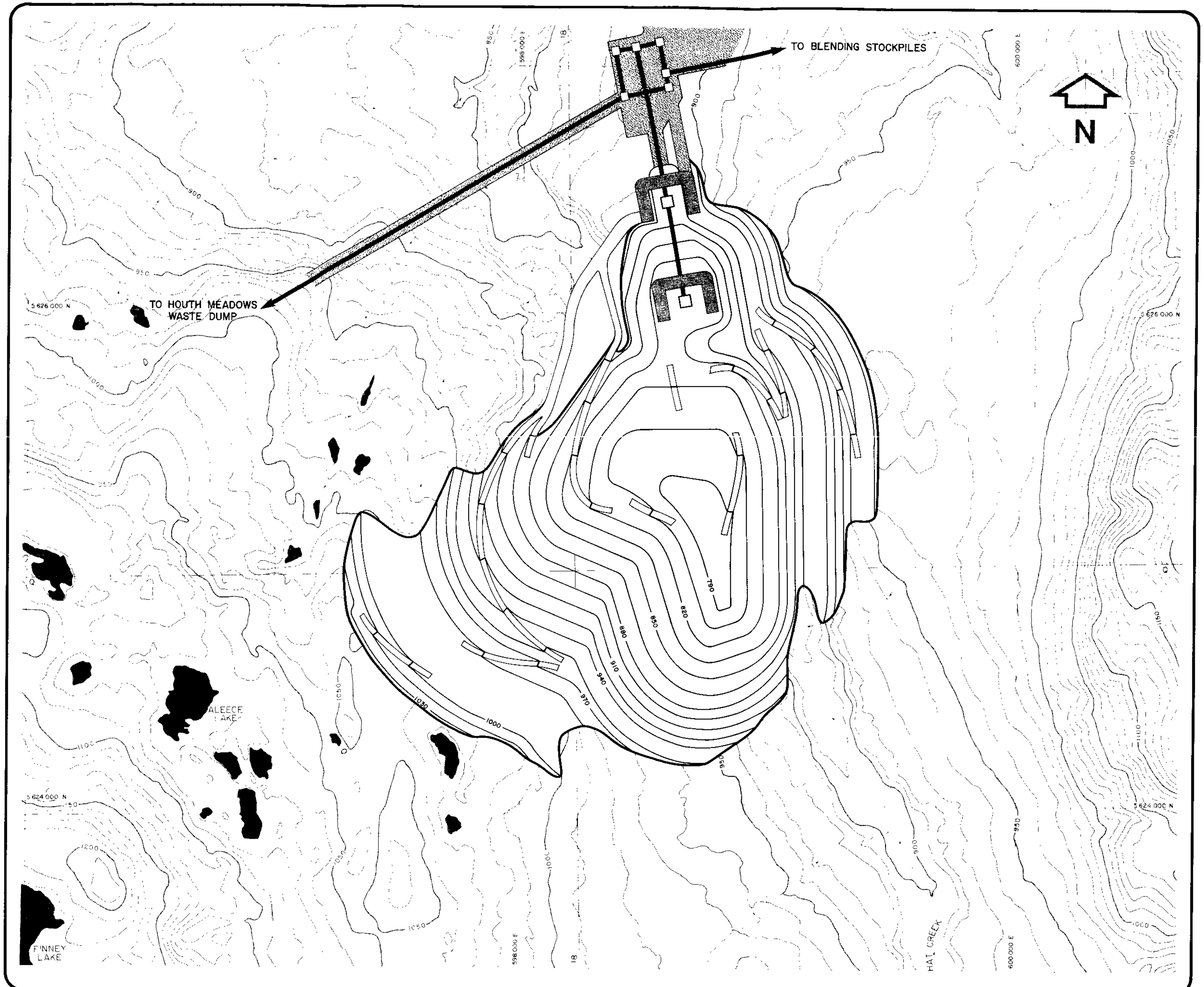
HAT CREEK PROJECT

FIGURE 5-1

Interim Pit Development: Cross Section Q

M084 C

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

SOURCE: British Columbia Hydro and Power Authority

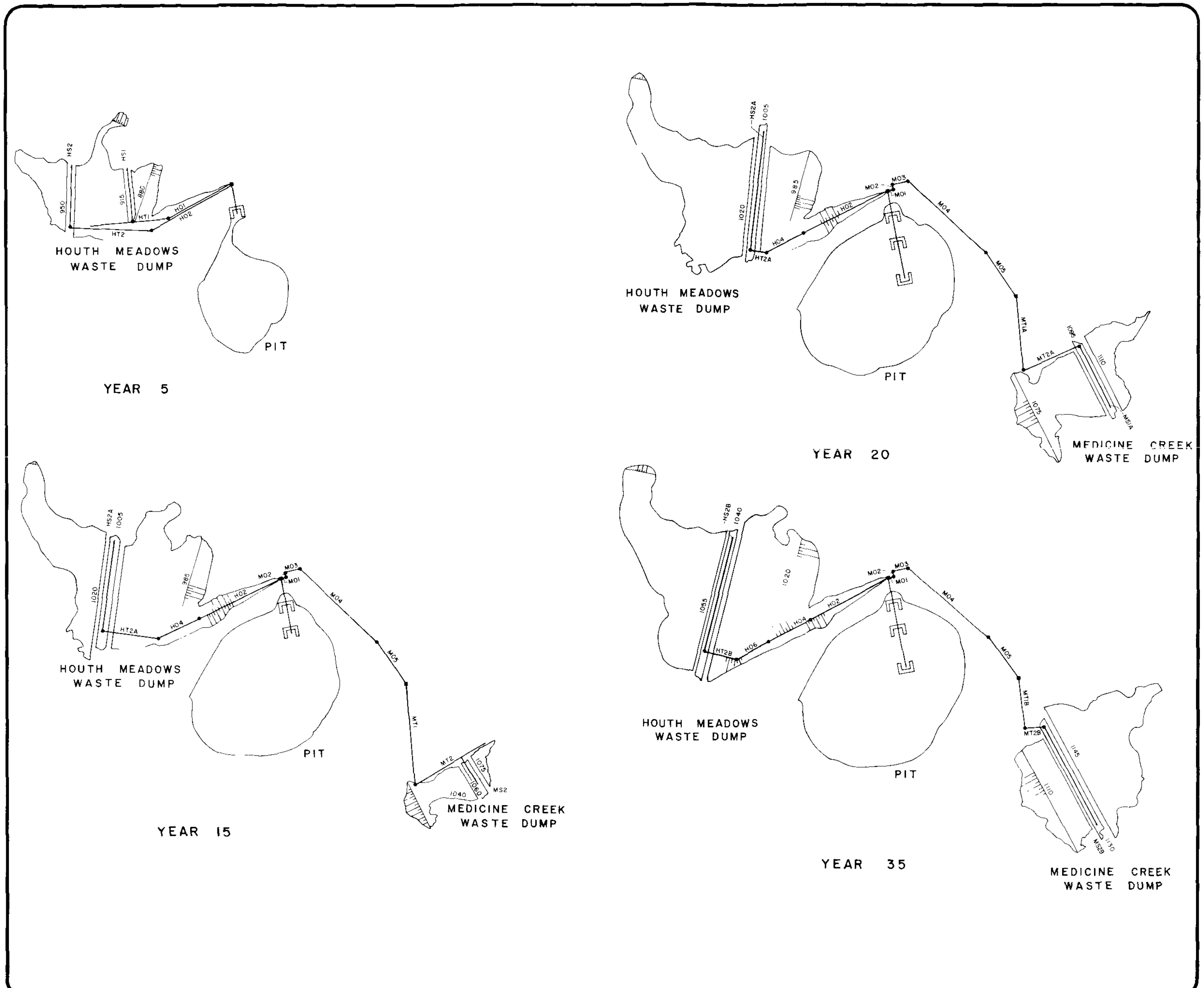


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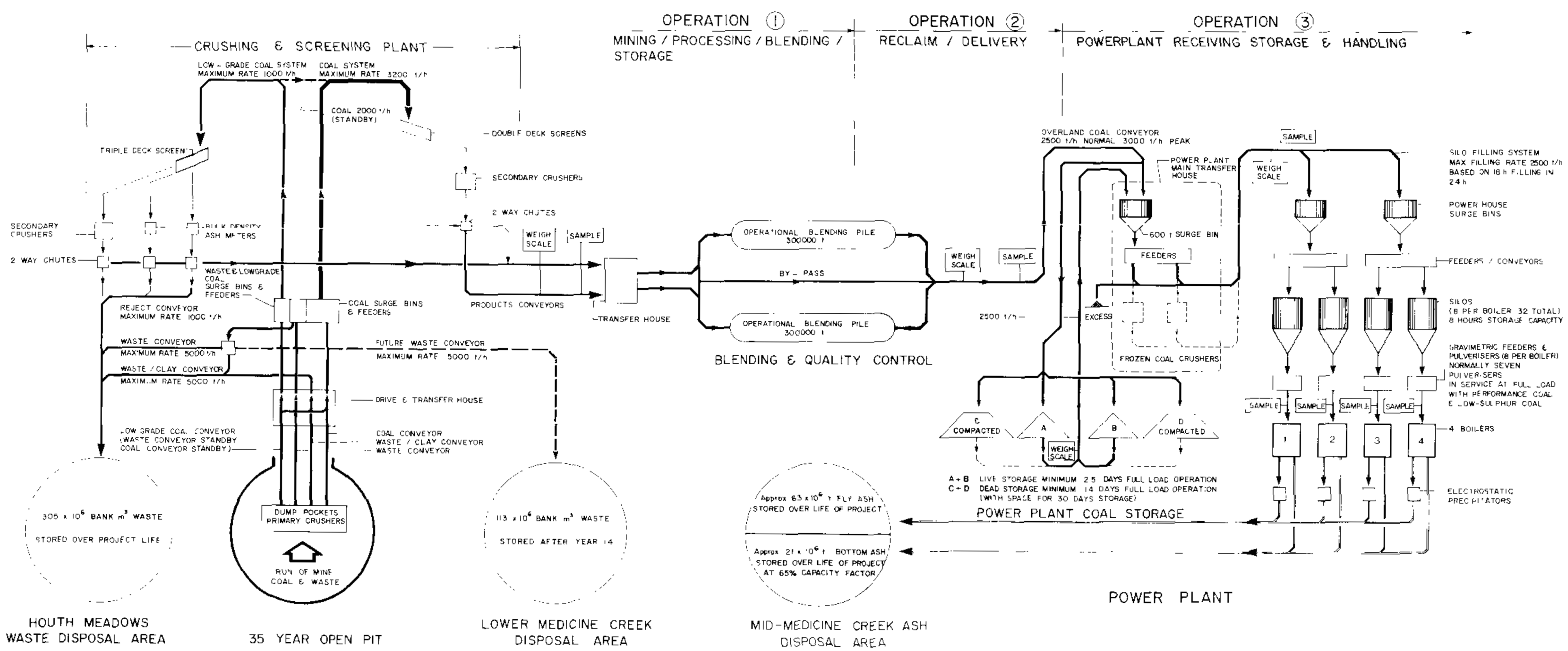
FIGURE 5-18

Waste Dump Development

SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 8-1

Overall Project Flow Diagram

SOURCE: British Columbia Hydro and Power Authority

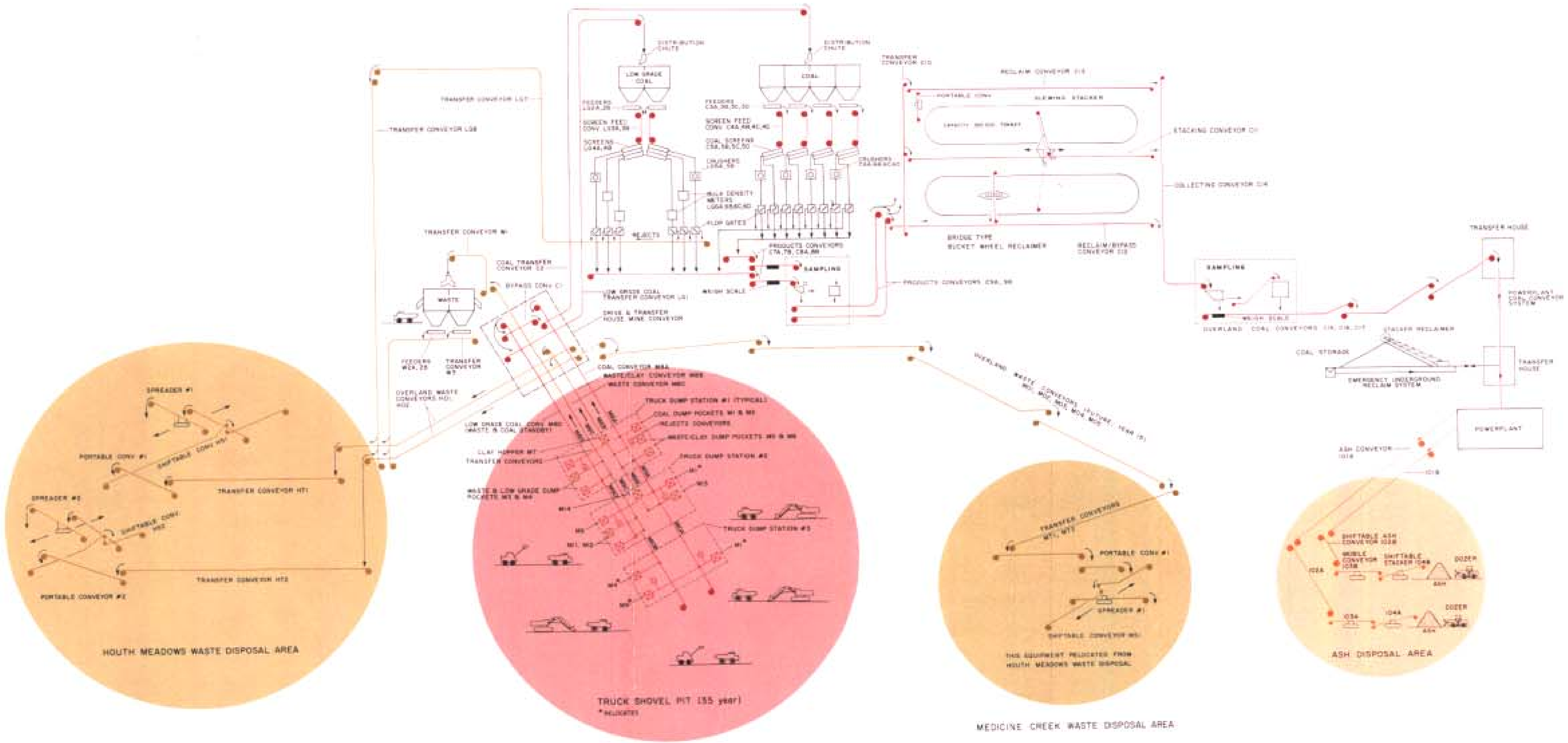


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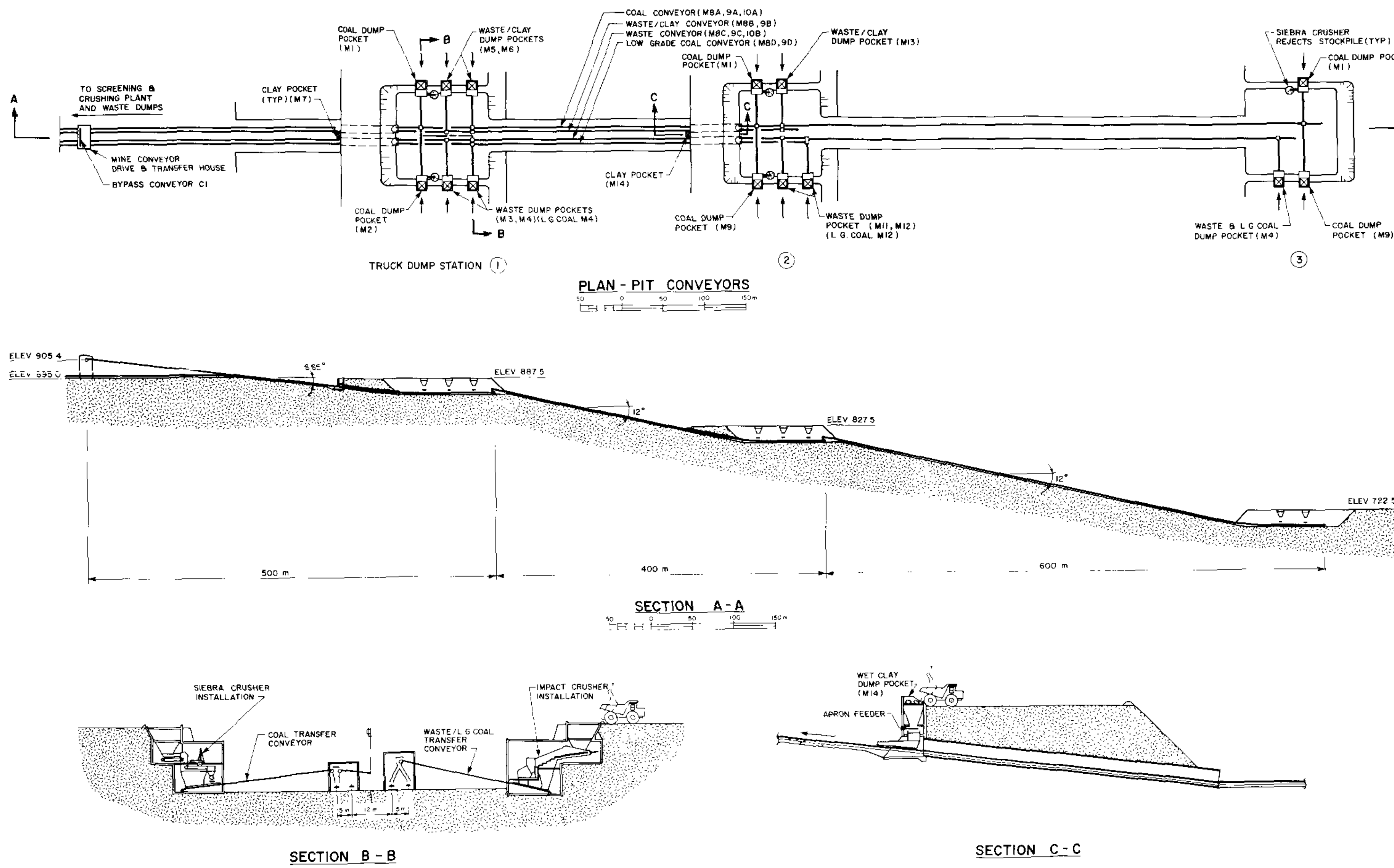
FIGURE 8-2

MATERIALS HANDLING SYSTEM FLOW DIAGRAM

M084 D

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.

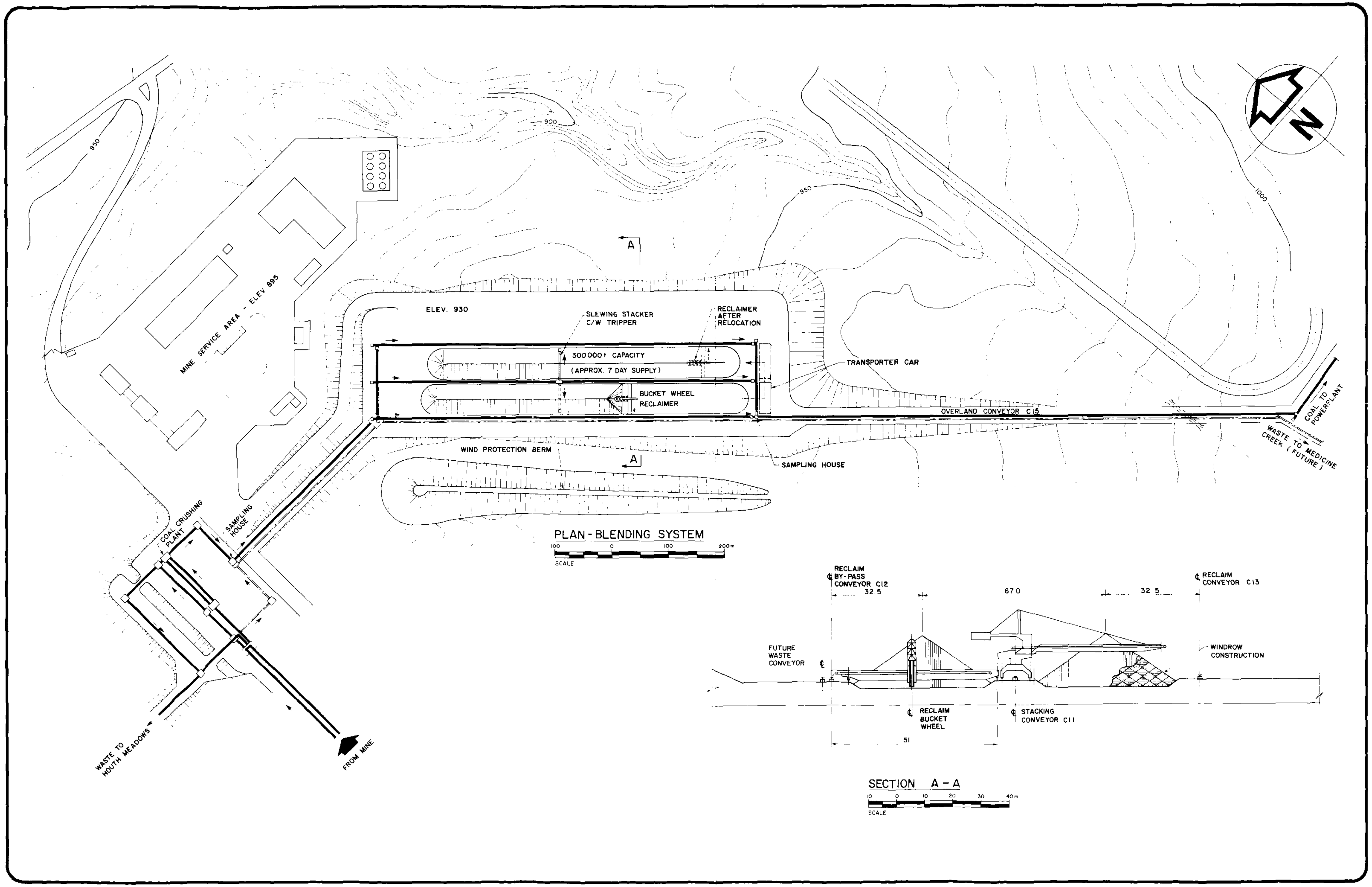


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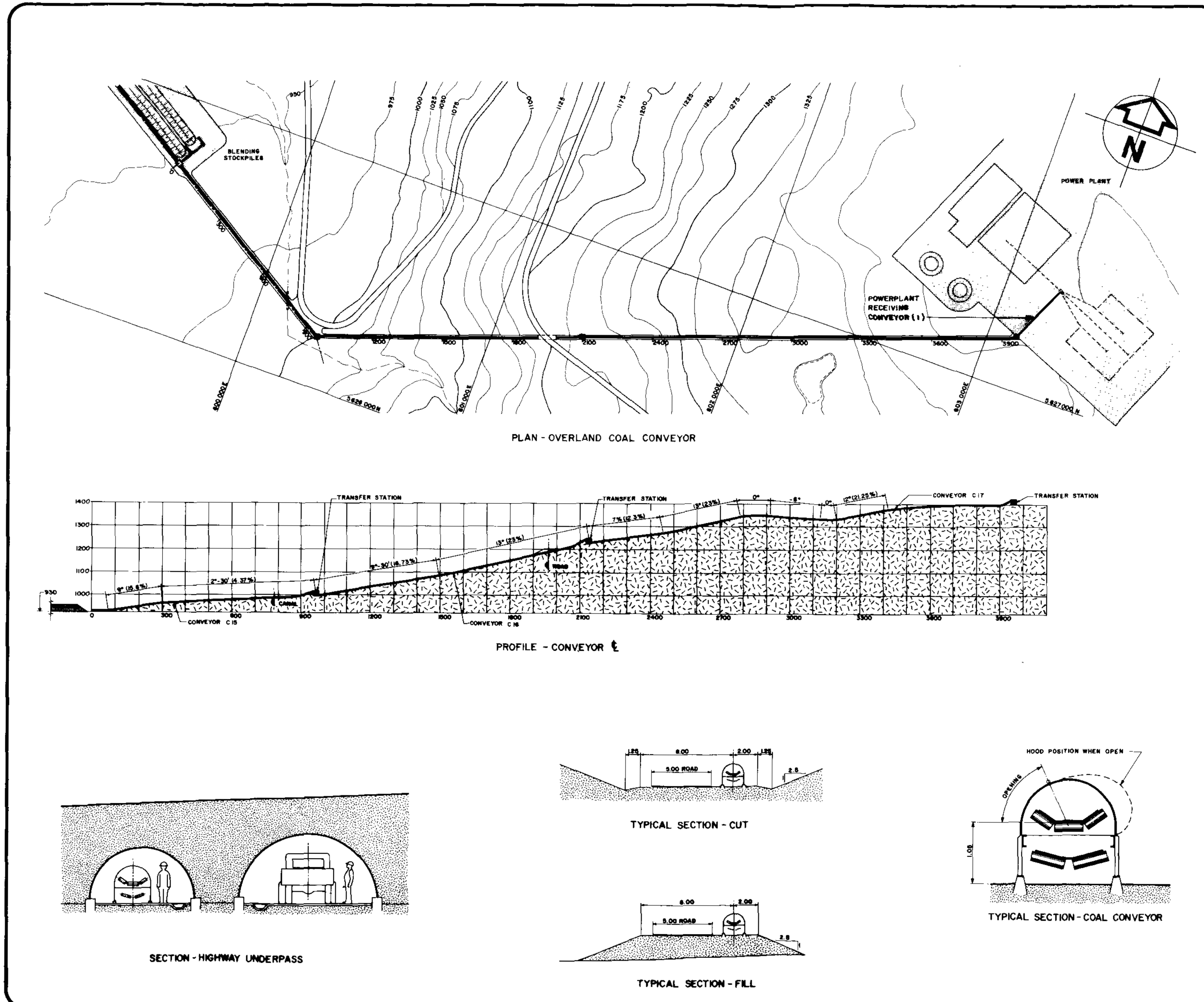
FIGURE 8-5

General Arrangement Blending / Reclaiming



0·5 km 0 0·5 1·0 km

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FIGURE 8-6

General Arrangement Overland Coal Conveyor

SOURCE: British Columbia Hydro and Power Authority

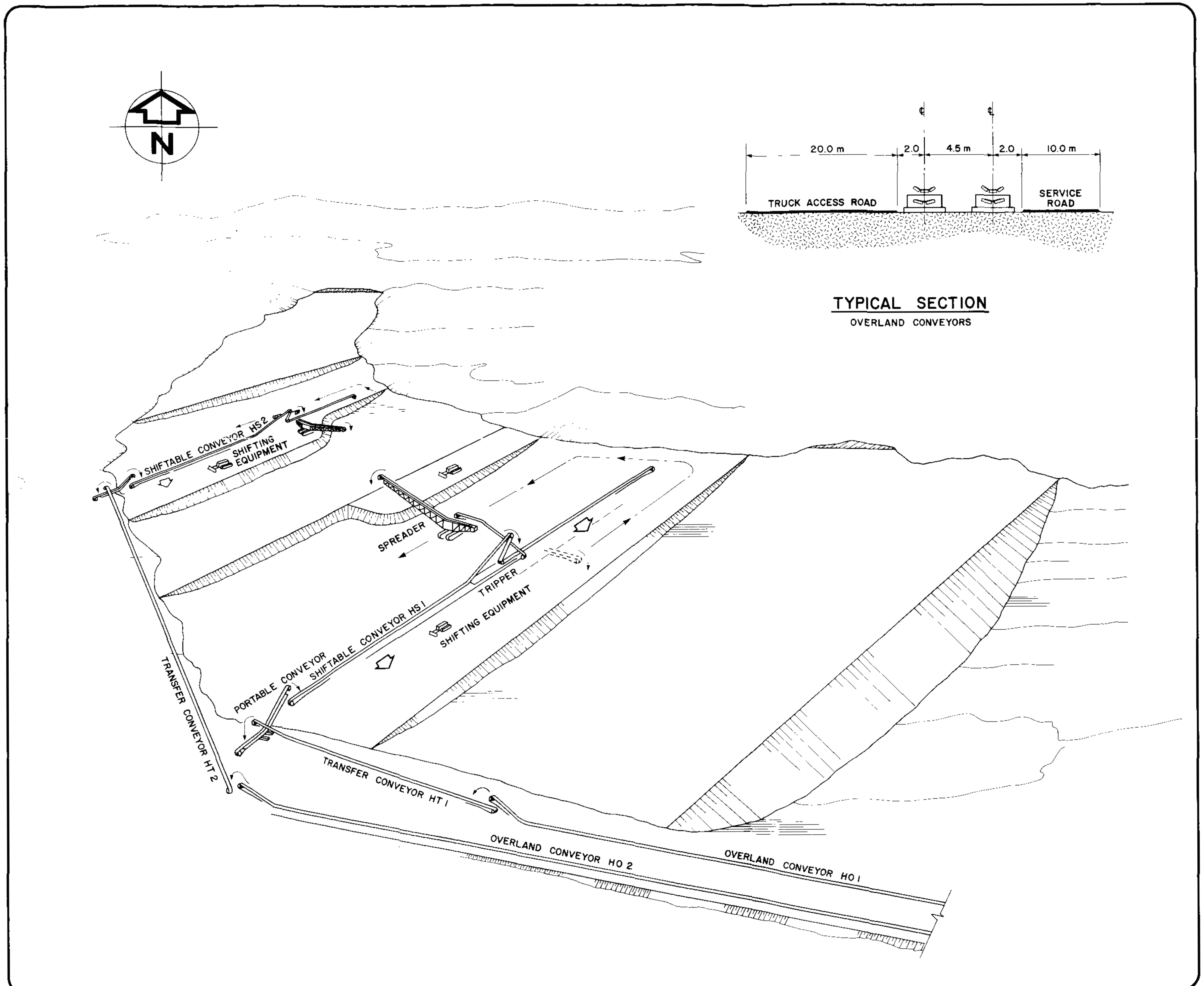

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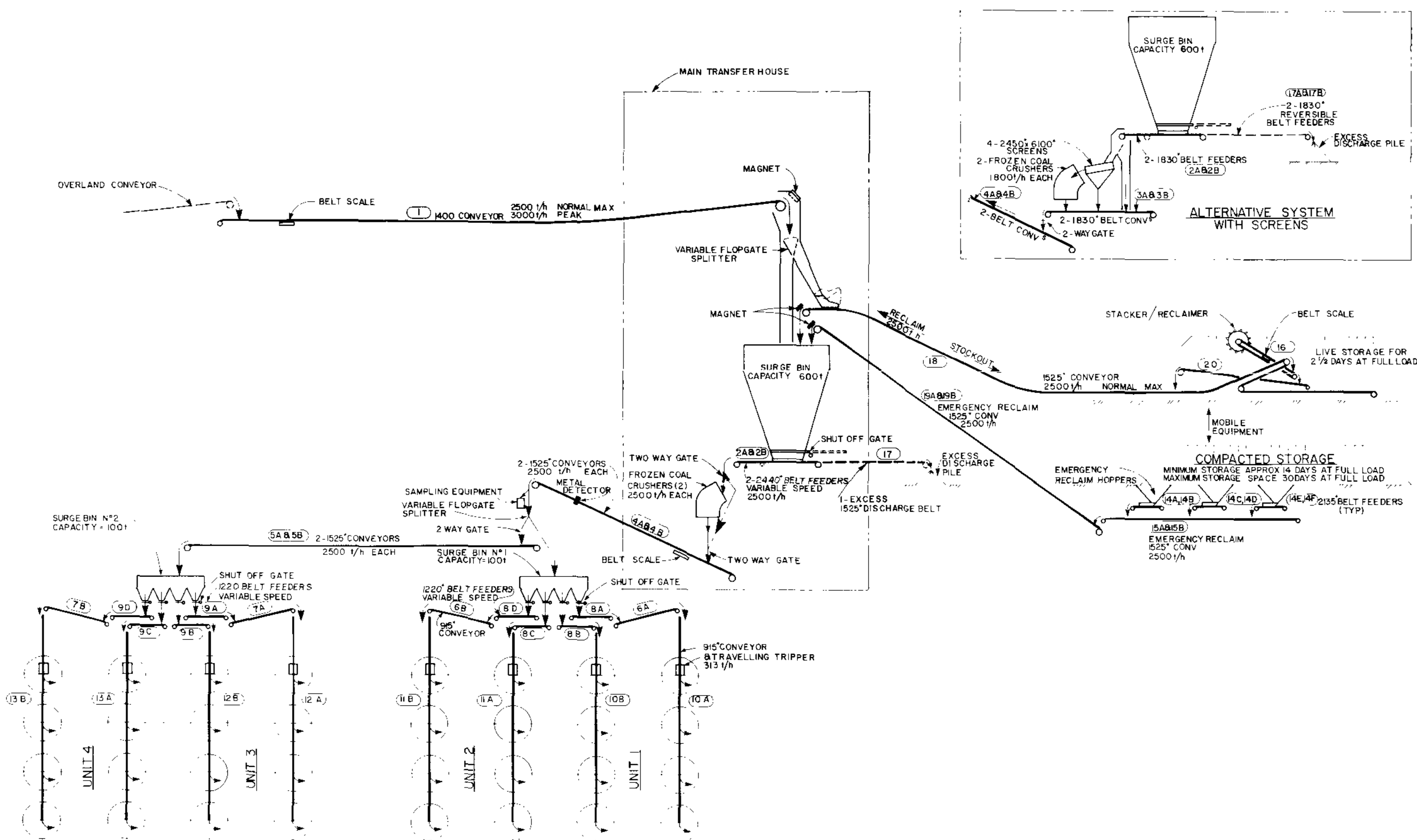
FIGURE 8-7

Pictorial View Waste Dump Development

SOURCE: British Columbia Hydro and Power Authority



NOTE
- ALL MEASUREMENT FIGURES ARE GIVEN IN MILLIMETRES

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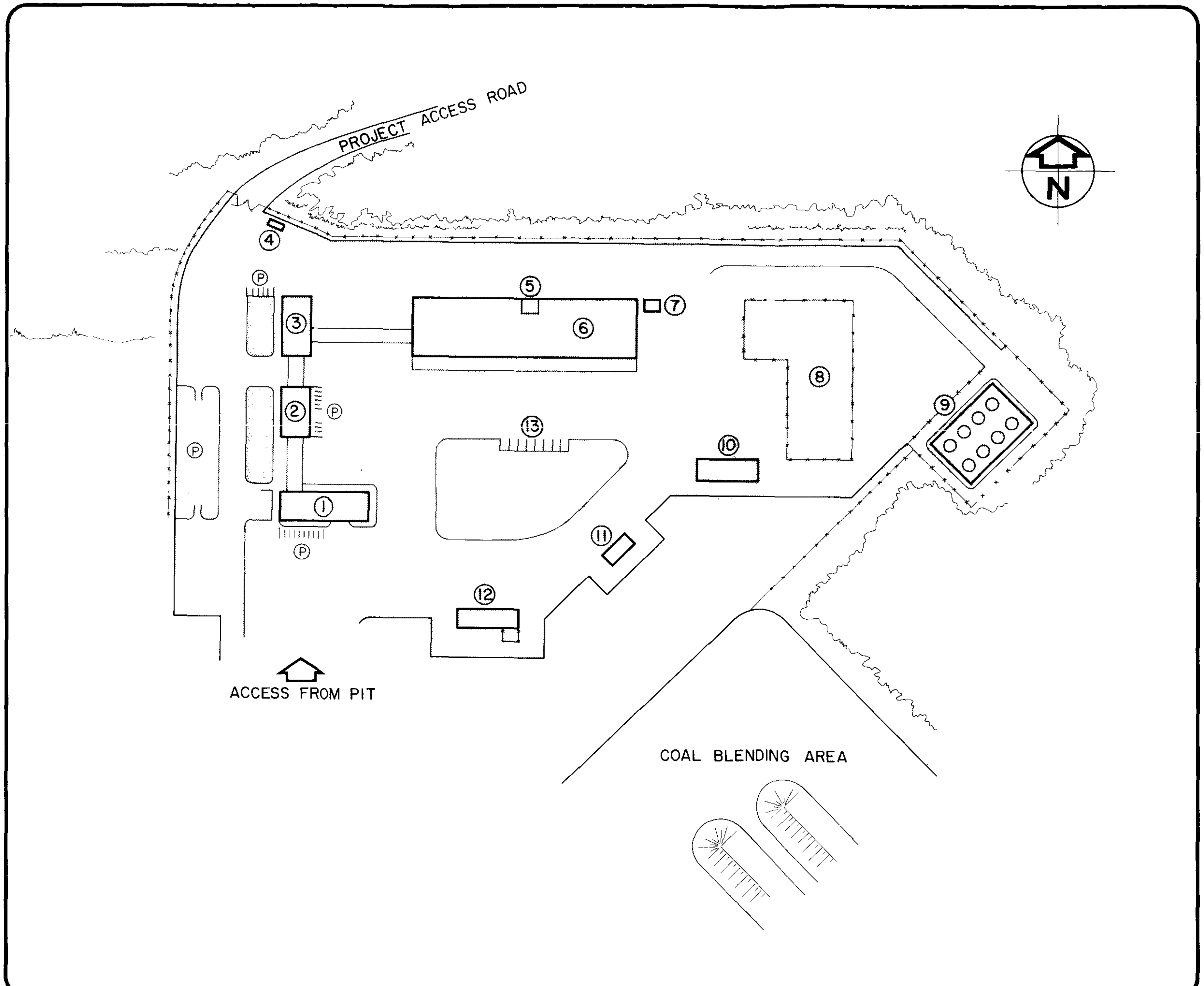
FIGURE 8-9

Powerplant Flow Diagram of Coal System

SOURCE: Integ-Ebasco



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

4 PRINCIPAL MINE DESIGN CONSIDERATIONS

The three tables marked with a star and all the figures in this Summary Report were originally numbered for the Main Report. These numbers could not be changed in the time available. The tables have therefore been inserted immediately following the first reference to them in the text, and the figures have been placed at the end of this section.

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SECTION 4

PRINCIPAL MINE DESIGN CONSIDERATIONS

This section presents the background to the principal features of the design of the mining operation. No attempt is made to present or to support all the detailed design criteria used. The intent is to provide some understanding of the key factors and decisions that have influenced the development of the plan.

The interaction of the mine and the powerplant has been fully recognized in the course of these studies. There are three principal interfaces between the two areas:

(1) The quality of the fuel supplied to the powerplant;

(2) The physical transfer of coal from the mine to the powerplant;

(3) The quantity of fuel required to meet powerplant production schedules.

4.1 RESOURCE EVALUATION

Coal in the Hat Creek Basin was first reported by G.M. Dawson of the Geological Survey of Canada in 1877. Several attempts were made over the years to develop the deposits on a small scale. A systematic program of exploration and evaluation was initiated by B.C. Hydro in 1974. Core-drilling programs were conducted each year from 1974-1978. With the exception of 1975, when the No. 2 Deposit was drilled, work has been concentrated on the smaller No. 1 Deposit.

To date, 206 core-holes totalling 54,000 metres have been drilled in the No. 1 Deposit. Geophysical logs were obtained on most of these holes. The cores obtained from drilling were carefully examined by geologists and their observations recorded. The coal intersections were sampled and submitted for proximate, ultimate, and ash analyses.

The data gathered in these drilling programs have been used to define the geological structure of the deposit and how the quality of the coal varies throughout the deposit. The No. 1 Deposit is now sufficiently well defined to permit the mine and powerplant design to proceed with an acceptable level of confidence.

4.1.1 Geology

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.

The unique thickness of the Hat Creek coal deposits and the wide range of coal quality encountered presented a difficult problem in establishing continuity in the deposit. The problem was ultimately overcome, and the deposit has been divided into 16 sub-zones (layers) - see Table 4-3. This was achieved using the geologists' description of the cores, the analytical results, and, most significantly, the geophysical logs.

Geophysical logs are obtained by raising a geophysical instrument containing an emission source through the length of a drill hole and recording the response electronically. The response is recorded continuously as a trace on a strip chart to provide a permanent record of the results. The most useful data on the Hat Creek deposits was provided by the Gamma Ray and Neutron Bulk Density logs. The interpretation of these logs identified characteristic signatures for marker horizons and various grades of coal, which could not be differentiated visually.

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.

Two of the 16 sub-zones identified are essentially waste bands with localized sections of coal. The remaining 14 sub-zones represent layers of varying coal quality, but demonstrate good continuity within the layers. The thickness of the sub-zones vary from five metres to 55 metres and typically average 20 metres.

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 (see Figures 4-3, 4-4A, and 4-4B).

In the North-West quadrant of the deposit, much of the coal is overlain by burnt material. 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. The burnt material is very good for building roads.

4.1.2 Coal Reserves

4.1.2.1 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.

4.1.2.2 Development of the Variable Block Model

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 metres 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 metres North and South.

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

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 metres in thickness, which reflects selective mining capability. Bands less than 2 metres 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 metres North-South and 500 metres 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:

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

Burn zone material was assigned a specific gravity of 2.16, and other waste 2.00.

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.

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

4.1.2.3 Reserves

The proven and probable coal reserves of the Hat Creek No. 1 Deposit have been computed to be 739 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-metre partings and a cut-off value of 9.3 MJ/kg. Table 4-1 shows the distribution of the reserves by sub-zones.

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 746 million tonnes coal at 16.72 MJ/kg, 37.73% ash, and 0.46% sulphur.

4.1.3 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. 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.

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

Estimates of the sulphur content of all the blocks contained within each sub-zone were produced by kriging. The kriged block values were input to the Variable Block Model for use in reserve and pit evaluation calculations.

Table 4-5 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;

TABLE 4-1

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	60561.	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 4-4

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 4-5

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

(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% (one standard deviation) 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 4-4.

The precision figures were calculated for 75 metres x 75 metres 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 metres x 150 metres 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.

4.2 GEOTECHNICS

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

4.2.1 Pit Slope Stability

The following design slope angles have been recommended by Golder Associates. These angles are steeper than the 16° recommended in earlier studies. 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.

The current design 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 can be gained within the deposit while slopes of modest height are 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 are 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.

4.2.2 Slide Areas

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 cubic metres 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^{o} from bedrock to surface perimeters.

4.2.3 Bench Height

For economic efficiency, a standard bench height of 15 metres 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.

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.

4.2.4 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.

4.2.5 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 metres 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.

4.3 SELECTIVE MINING

4.3.1 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 metres to 10 metres, have been identified. The D-zone coal was deposited during a stable period. Few waste partings were formed and the best, most consistent quality of coal, is contained in the D-zone. The B-zone was also deposited under relatively stable conditions although there were a few incursions of sediment-laden floods to produce some waste bands.

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

The larger waste and low-grade partings are simple to identify and easily mined as waste material. The smaller partings, up to 5 metres, are more readily mined with the coal. However, while this simplifies the mining process, it reduces the quality of the coal 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 0.5 metres to 5 metres. These studies indicated that significant improvements in fuel quality could be obtained with selective mining. This improvement would be particularly significant in the A-zone. In the C-zone the quality improvement would be small, but more coal would be recovered. Overall, the indications were that as much, or more, total heat content could be recovered depending on the size of parting that could be removed.

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 (0.5-1 metre) would be high and reduce equipment productivity significantly.

4.3.2 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 cubic metre bucket was able to segregate partings one metre thick. This separation is possible primarily because of the difference in the physical characteristics between the coal which is hard, and the partings which are soft. After exposure to the atmosphere for a week or two, sufficient drying of the coal face occurs to highlight the colour differences between coal and waste. This assists in the identification of the different materials. Observation of larger hydraulic shovels with 10 cubic metre buckets at other mining operations indicates that the wrist-like digging action of these machines will permit selective mining of partings 1.5 metres to 2 metres 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 metres thick and greater can be segregated effectively without significantly reducing equipment productivity or increasing mining costs. In practice, it will often be possible to mine selectively bands less than 2 metres, depending on their position and attitude.

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.

4.3.3 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.

4.4 PIT DESIGN AND PRODUCTION SCHEDULING

The function of pit design is to produce a practical pit with the most favourable economics attainable while meeting the design criteria. In production scheduling, the designed pit is sequenced to deliver the required quantity of coal at an acceptable quality with a minimum of waste removal for each time period.

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 4.1) to the completion of the production schedule.

4.4.1 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 metre 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).

4.4.2 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 metres square in plan and 15 metres high. A block of coal this size represents approximately 55,000 tonnes. 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 metres, 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.

4.4.3 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 metres 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°, South and West 19°, and North 15° (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° 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 4-6. 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.

4.4.4 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,

TABLE 4-6

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.[illegible]5	1.23	11449.	17.59	0.75	9.30
R2PM1.DAT	109441.	18.[illegible]3	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

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 4-7) 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).

TABLE 4-7

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	13	14	15
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	11.64	11.40	11.12
	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	124.21	135.61	146.73
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	17.0	17.2	17.6
	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	17.9	17.9	17.8
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	35.31	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	0.54	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	149.60	150.00	149.72
	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	1704.52	1854.52	2004.24
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	7.81	7.65	7.46
	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	83.36	91.01	98.47
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	10.62	9.84	9.09
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	6.79	6.29	5.81
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	17.41	16.13	14.90
	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	201.67	217.80	232.70
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	25.22	23.78	22.36
	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	285.03	308.81	331.17
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	1.50	1.41	1.34
	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	1.62	1.61	1.59

MATERIALS MINED Quantities in (10^6)		PRODUCTION YEARS																				TOTAL
		16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	
COAL MINED (Tonnes)	Annual	10.06	10.06	10.67	10.15	9.90	9.66	9.57	10.40	10.03	9.83	8.01	9.19	8.39	8.55	8.58	8.64	8.61	8.25	8.05	7.62	
	Cumul.	156.79	166.85	177.52	187.67	197.56	207.22	216.79	227.19	237.21	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	18.0	18.1	17.1	17.9	18.4	18.9	19.0	17.5	18.2	18.5	19.2	17.0	18.4	18.0	17.9	17.8	17.9	18.7	19.1	20.2	
	Cumul.	17.9	17.9	17.8	17.8	17.9	17.9	18.0	17.9	18.0	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	33.90	33.90	33.90	33.90	33.90	32.77	32.77	32.77	32.77	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	0.50	0.50	0.50	0.50	0.50	0.48	0.48	0.48	0.48	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	138.53	139.30	139.58	138.99	139.35	139.67	139.10	139.23	139.65	139.12	117.65	118.11	118.10	117.73	117.49	117.65	117.90	118.02	117.62	117.75	
	Cumul.	2142.77	2282.07	2421.65	2560.64	2699.99	2839.66	2978.76	3117.99	3257.64	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	6.75	6.75	7.16	6.81	6.64	6.48	6.42	6.98	6.73	6.60	5.38	6.17	5.63	5.74	5.76	5.80	5.78	5.54	5.40	5.11	
	Cumul.	105.23	111.98	119.14	125.95	132.59	139.07	145.50	152.48	159.20	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	8.80	8.07	7.35	6.70	6.57	2.16	2.16	2.10	2.10	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	5.40	4.95	4.50	4.10	4.02	8.12	8.12	7.89	7.92	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	14.20	13.02	11.85	10.80	10.59	10.28	10.28	9.99	10.02	10.00	9.83	9.85	9.84	9.85	9.86	9.86	9.85	9.82	2.35	1.96	
	Cumul.	246.90	259.92	271.77	282.57	293.16	303.44	313.72	323.71	333.73	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	20.95	19.77	19.01	17.61	17.23	16.76	16.70	16.97	16.75	16.60	15.21	16.02	15.47	9.58	15.62	15.66	15.63	15.81	7.75	7.07	
	Cumul.	352.13	371.90	390.91	408.52	425.75	442.51	459.22	476.19	492.93	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	1.41	1.29	1.11	1.06	1.07	1.06	1.07	0.96	1.00	1.02	1.23	1.07	1.17	1.15	1.15	1.14	1.14	1.19	0.29	0.26	
	Cumul.	1.57	1.56	1.53	1.59	1.48	1.46	1.45	1.42	1.41	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

SOURCE: British Columbia Hydro and Power Authority

4.5 EQUIPMENT SELECTION

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.

This section discusses the selection of the production 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.

4.5.1 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 kilometres per hour. On downhill hauls the speed limits were established to provide maximum braking capability.

4.5.1.1 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 metres was balanced against the economics of using larger equipment. The machine selected for loading coal is a hydraulic shovel with a 10.7 cubic metres 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 cubic metre hydraulic shovel. These trucks were rated at 77 tonnes, 91 tonnes, and 109 tonnes.

The economic analysis performed for the three selected critical periods showed a marginal cost advantage for the 77-tonne truck over the 91-tonne truck, with the 109-tonne truck ranked third. In reviewing the fleet size developed in the analysis, a better balance of trucks to shovels was obtained with the 77-tonne trucks, which were confirmed as the most suitable coal truck.

During the peak production years the number of 77-tonne trucks required ranges from nine to 11. The principal specifications of the truck are: 77 cubic metre coal box; 1,000 horsepower engine; 24.00 x 49 tires, and a 23:1 gear ratio.

4.5.1.2 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 cubic metre bucket is the only production unit in this size range currently available. Four of these units will be required at peak production levels.

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 cubic metre cable shovel with the 14.5 cubic metre hydraulic shovel. The results showed marginal cost savings using the 16.8 cubic metre 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 cubic metre 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 cubic metre hydraulic shovels were evaluated for waste haulage. These trucks were 109 tonnes, 136 tonnes, and 154 tonnes. Other truck sizes were eliminated in a preliminary evaluation.

The economic analysis was performed using the same three critical periods identified above. The 154-tonne truck showed the lowest unit production costs and was selected for waste haulage. The maximum requirement is for 14 154-tonne trucks between Years 10 and 14. The principal specifications for the truck are: 90 cubic metre rock box; 1,600 horsepower 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.

4.5.1.3 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-tonne waste trucks can be loaded by the 10.7 cubic metre 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-tonne trucks loaded either by 10.7 cubic metre 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.

4.5.2 Support Equipment

Mine support equipment is required for four principal tasks: *road construction and maintenance*; *mine support*; *materials-handling support*; and general service equipment. There is a considerable overlap in the application of specific types of equipment to the different categories.

A substantial fleet of support equipment has been provided for in the plan. This fleet includes: tandem-powered scrapers; 32-tonne trucks and front-end loaders; D-9, D-8, and D-7 bulldozers; rubber-tired dozers; graders; water trucks and a mobile crusher. In addition, the need for a fleet of emergency, service, and personnel transportation vehicles has been identified.

4.6 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.

4.6.1 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-metre 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 millimetres. The (20 millimetres 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-tonne 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.

4.6.2 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.

4.6.3 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 millimetres. 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 millimetres. 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.

4.6.4 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 cubic metres 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 kilometres. 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.

4.6.5 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.

4.7 FUEL QUALITY

4.7.1 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.

4.7.2 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 4.3, 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 4-9 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 4-9 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.

TABLE 4-9

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

Category 3, the low-grade coal, was considered marginal fuel for the boilers and is discussed further in Section 4.7.4. 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 metres 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%.

4.7.3 Boiler Fuel Specification Development

4.7.3.1 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.

4.7.3.2 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:

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

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.

4.7.3.3 Fuel Assessment

4.7.3.3.1 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 4-10).

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.

4.7.3.3.2 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 4-10

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 4-10 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.

4.7.3.4 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 are presented in Table 4-11.

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 4-12 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.

4.7.4 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

TABLE 4-11

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 4-12

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.

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 4.7.2, 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 millimetres or 20 millimetres 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. 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.

4.8 FUEL QUALITY CONTROL

4.8.1 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.

4.8.2 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 metres 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 powerplant will be monitored in a similar manner on the Overland Conveyor as a confirmatory check on quality.

4.8.3 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.

4.9 ENVIRONMENTAL PROTECTION

4.9.1 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 millimetres per annum, of which about half falls as snow. Winds behave according to the mountain/valley topography and are channelled predominantly upslope from the North to the South and 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.

4.9.2 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 at an elevation of 930 metres, a protective dike to 950 metres 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.

4.9.3 Noise

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

Present noise levels in the valley vary from about 30 to 40 decibels in the areas away from Highway 12. Adjacent to the highway, noise levels range from 44 to 51 decibels. By comparison, a soft whisper would produce a sound level of about 30 decibels, and a quiet wind through the trees would be around 50 decibels.

Noise from construction would, of course, be transitory, whereas noise from the mine operation essentially constant throughout the mine's productive life. The latter would stem principally from heavy equipment moving in and around the pit, with intermittent additional noise from the coal stacker-reclaimer, conveyors and crushers. Only two of the five Hat Creek ranches are expected to be affected by construction activity noise. Maximum noise levels on these ranches would reach 47 decibels which is close to the 45 decibels typically set 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 decibels; the next two, levels of between 45 and 49 decibels; and the two furthest away, levels of 41 to 42 decibels. As the natural background level is 35 to 40 decibels, the occasional level of noise from the mining operation is not expected to cause annoyance to anyone reasonably disposed.

4.9.4 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. 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 milligrams per litre;

(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.

4.9.5 Land Reclamation

On-site Reclamation Testing

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

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

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

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

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

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

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

Vegetation Species

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

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

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

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

Waste Dumps and Embankments

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

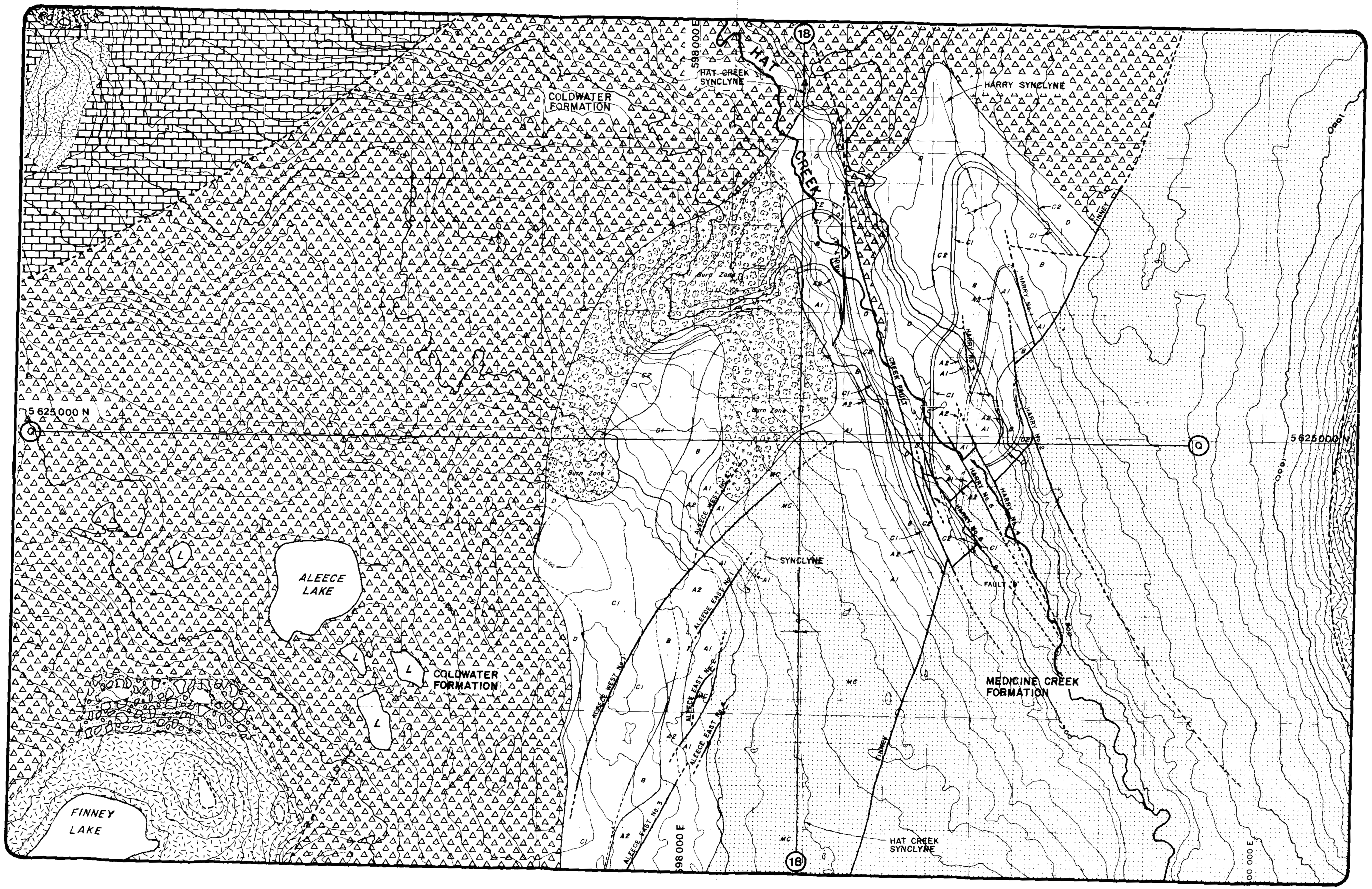
Areas disturbed due to construction of facilities such as the transportation corridors will be reclaimed as soon as possible following construction. Trees will be planted where appropriate to screen the development and enhance the aesthetic appearance of the project.

Reclamation upon Project Termination

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

The plan adopted for abandoning the open pit after 35 years provides for resloping the top three benches (about 115 hectares) from 45° to 26° to provide a safer perimeter and lessen the visual impact. No resloping will be done below this level. After resloping, fertilizer and seed will be aerially broadcast on all pit benches. In time revegetated overburden and slide areas may be expected to creep and to slump into the pit. Figure 11-1 shows the extent of reclamation in Year 45.

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.



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	60 - 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)

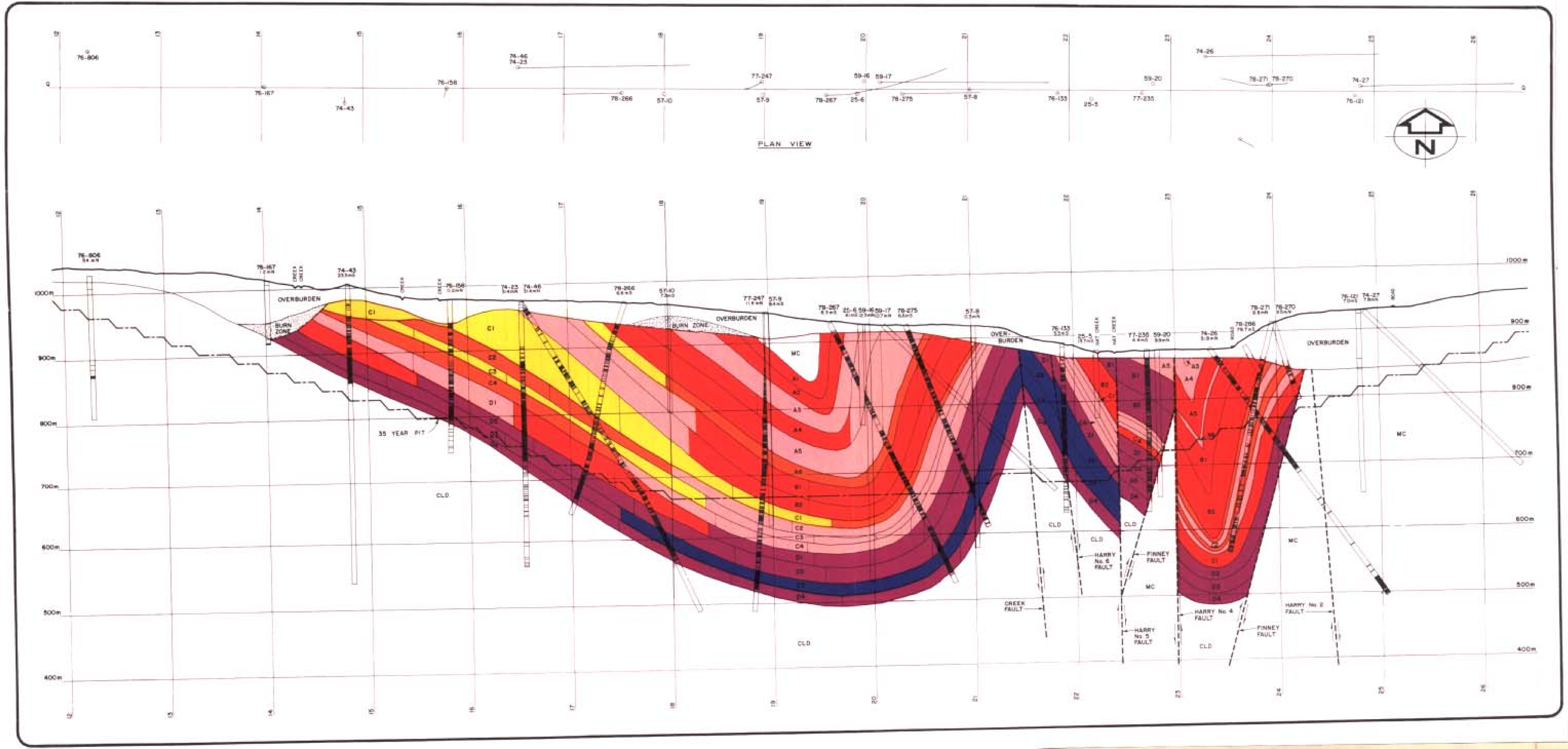
L LAKE

BURNT COAL ZONE

FIGURE 4-2

REGIONAL BEDROCK GEOLOGY NO. 1 DEPOSIT

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

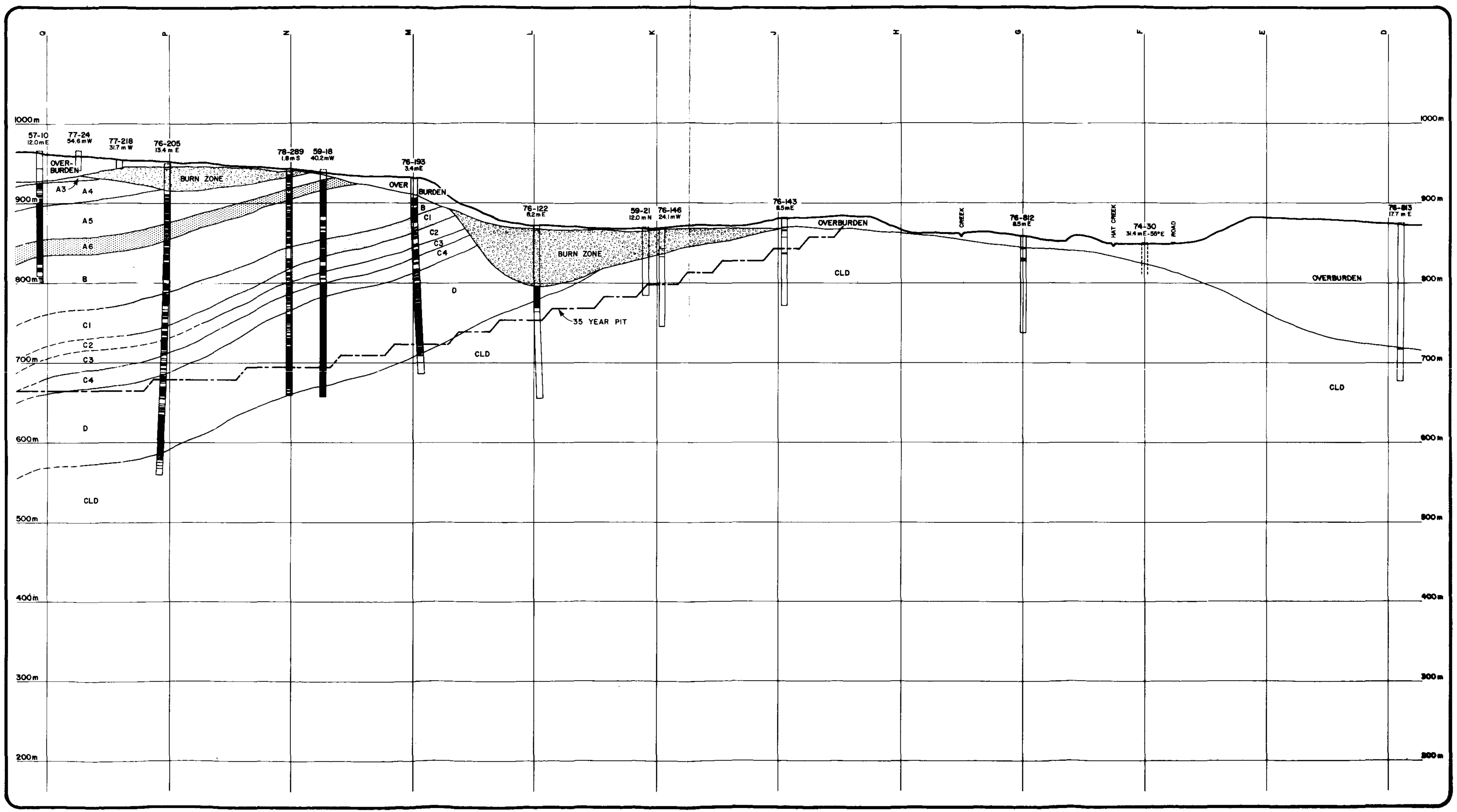
100m 0 100 200 300m

HAT CREEK PROJECT

FIGURE 4-3
GEOLOGICAL CROSS SECTION SECTION Q
SECTION DRAWN LOOKING NORTH

M084 E

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

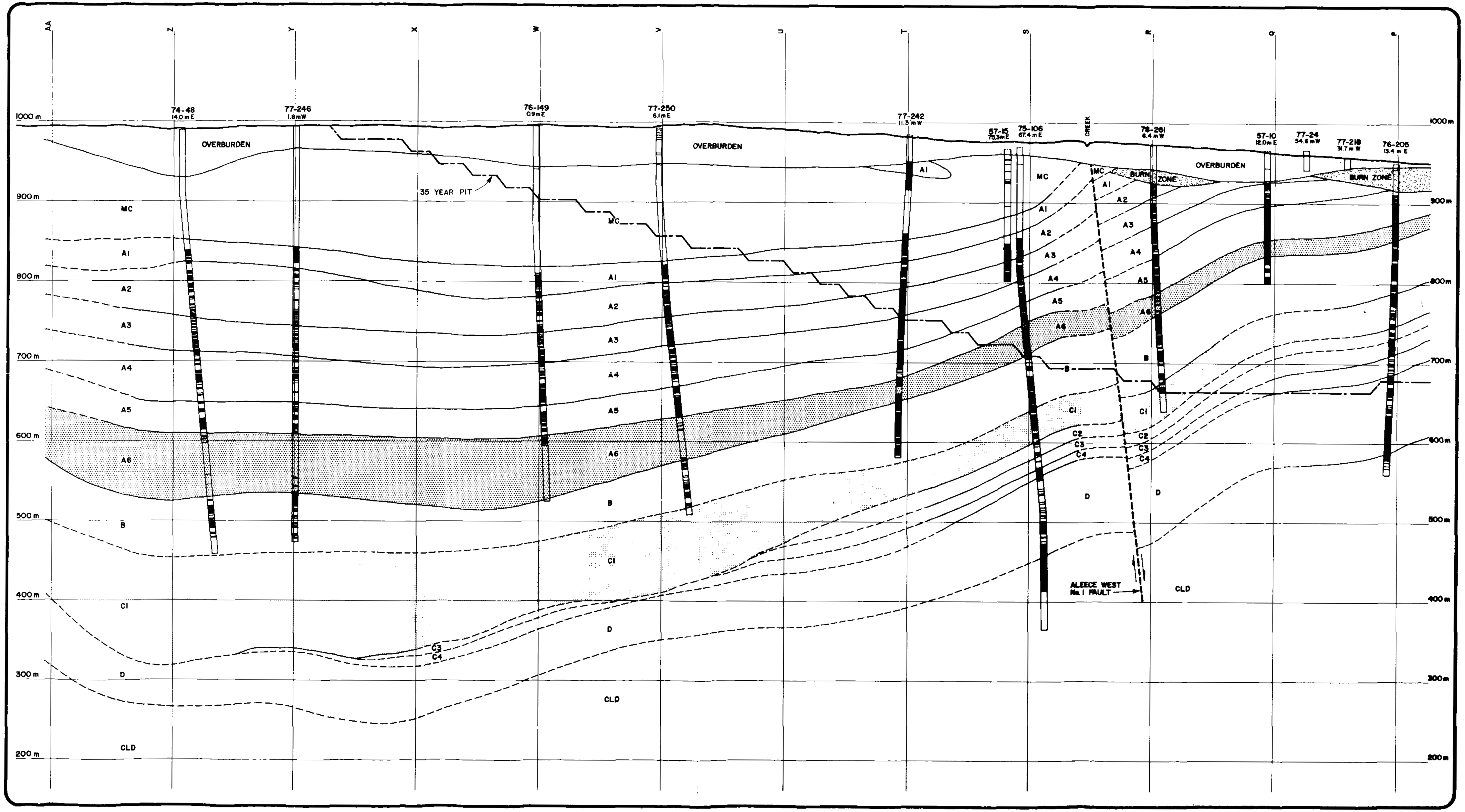
HAT CREEK PROJECT

FIGURE 4-4A

GEOLOGICAL CROSS SECTION 18 NORTH

SECTION DRAWN LOOKING WEST

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

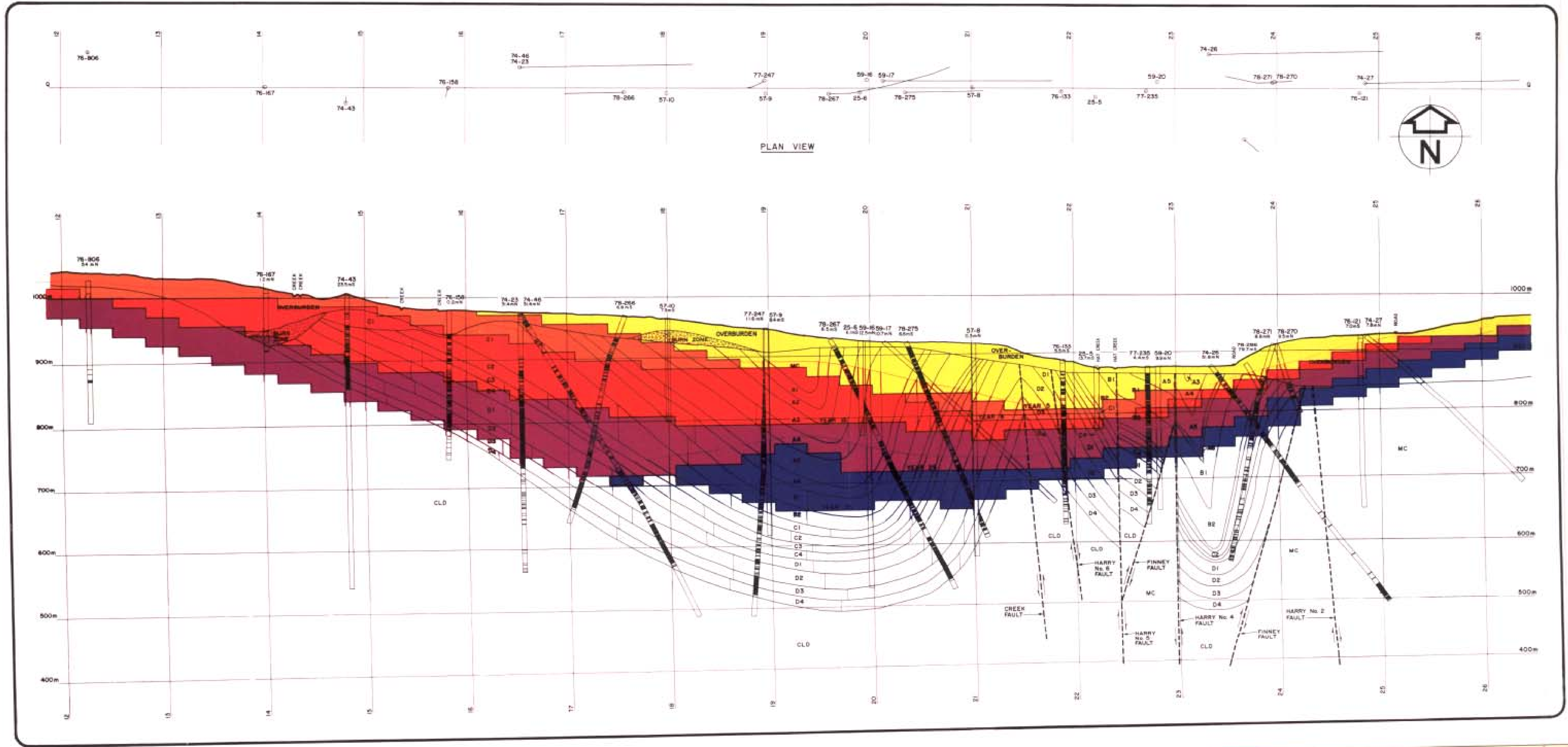
HAT CREEK PROJECT

FIGURE 4-4B

GEOLOGICAL CROSS SECTION 18 SOUTH

SECTION DRAWN LOOKING WEST

SOURCE: British Columbia Hydro and Power Authority



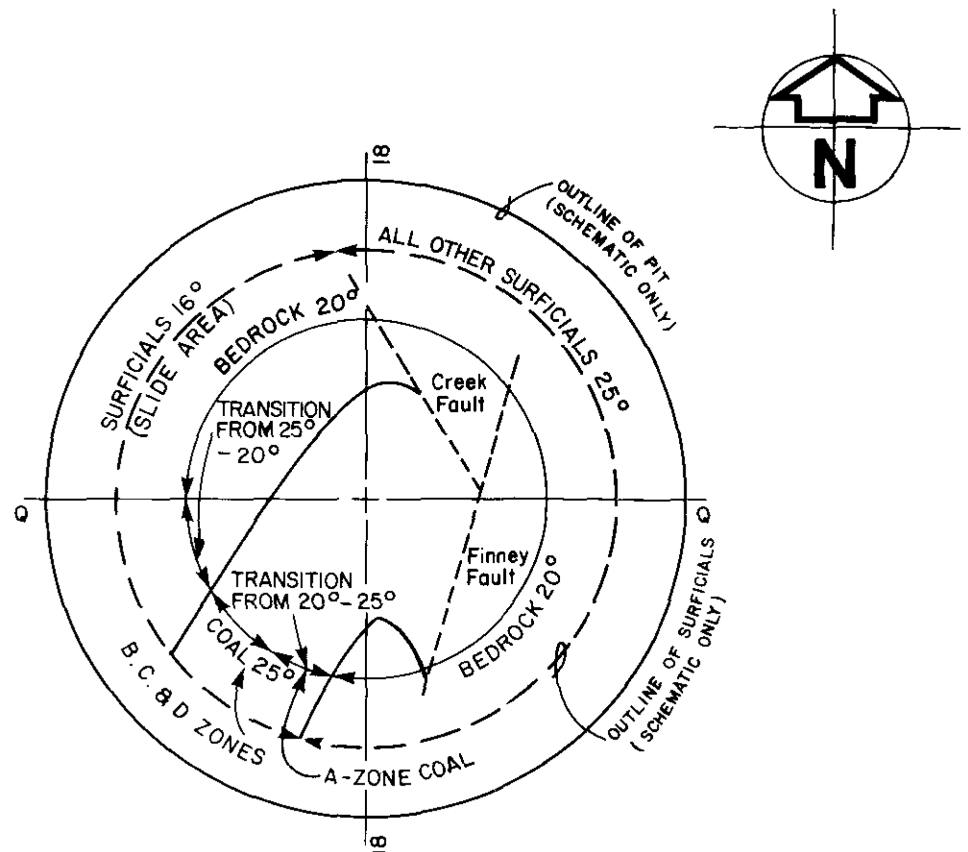
HAT CREEK PROJECT

FIGURE 5-1

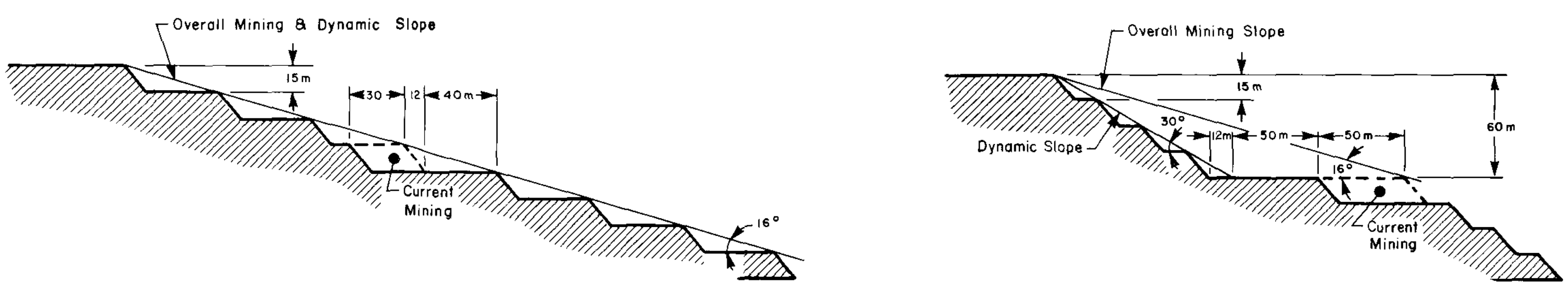
Interim Pit Development: Cross Section Q

M084 F

SOURCE: British Columbia Hydro and Power Authority



35 YEAR PIT SLOPE ANGLES



(a) FOR MINING COAL

(b) FOR MINING WASTE AND OVERBURDEN

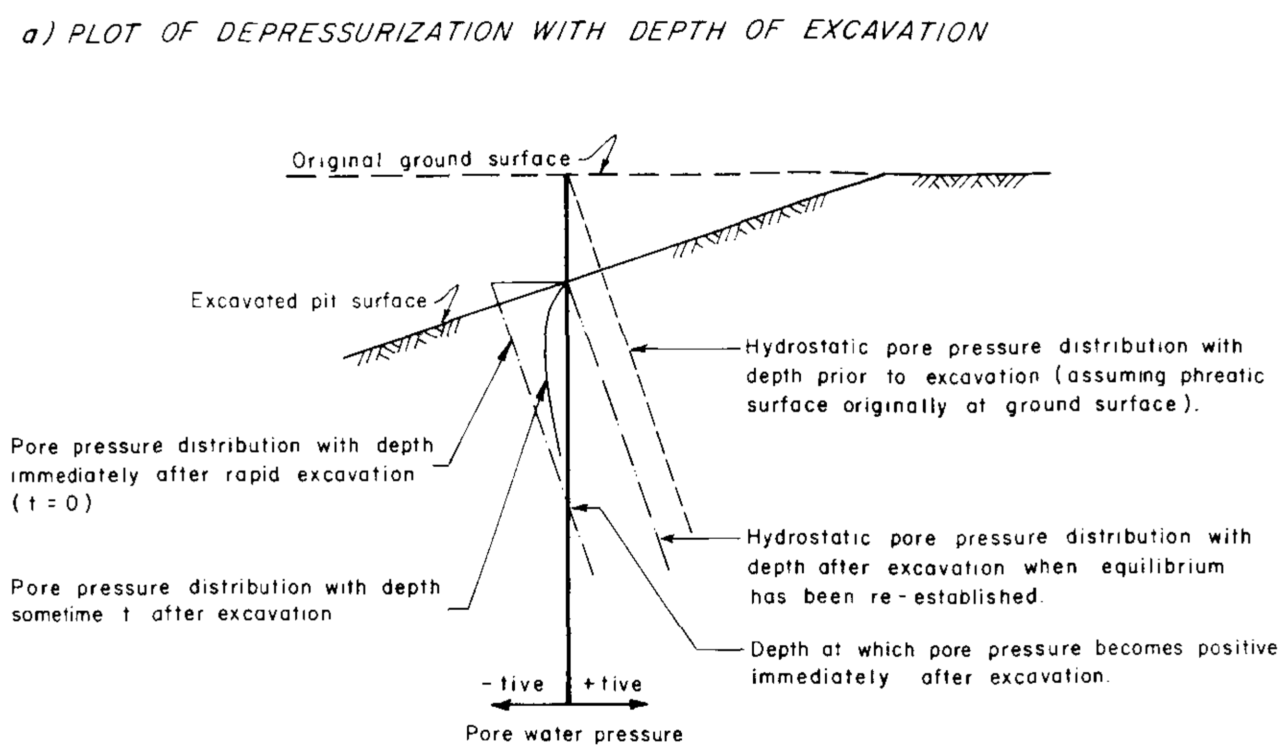
DYNAMIC SLOPE CROSS SECTION

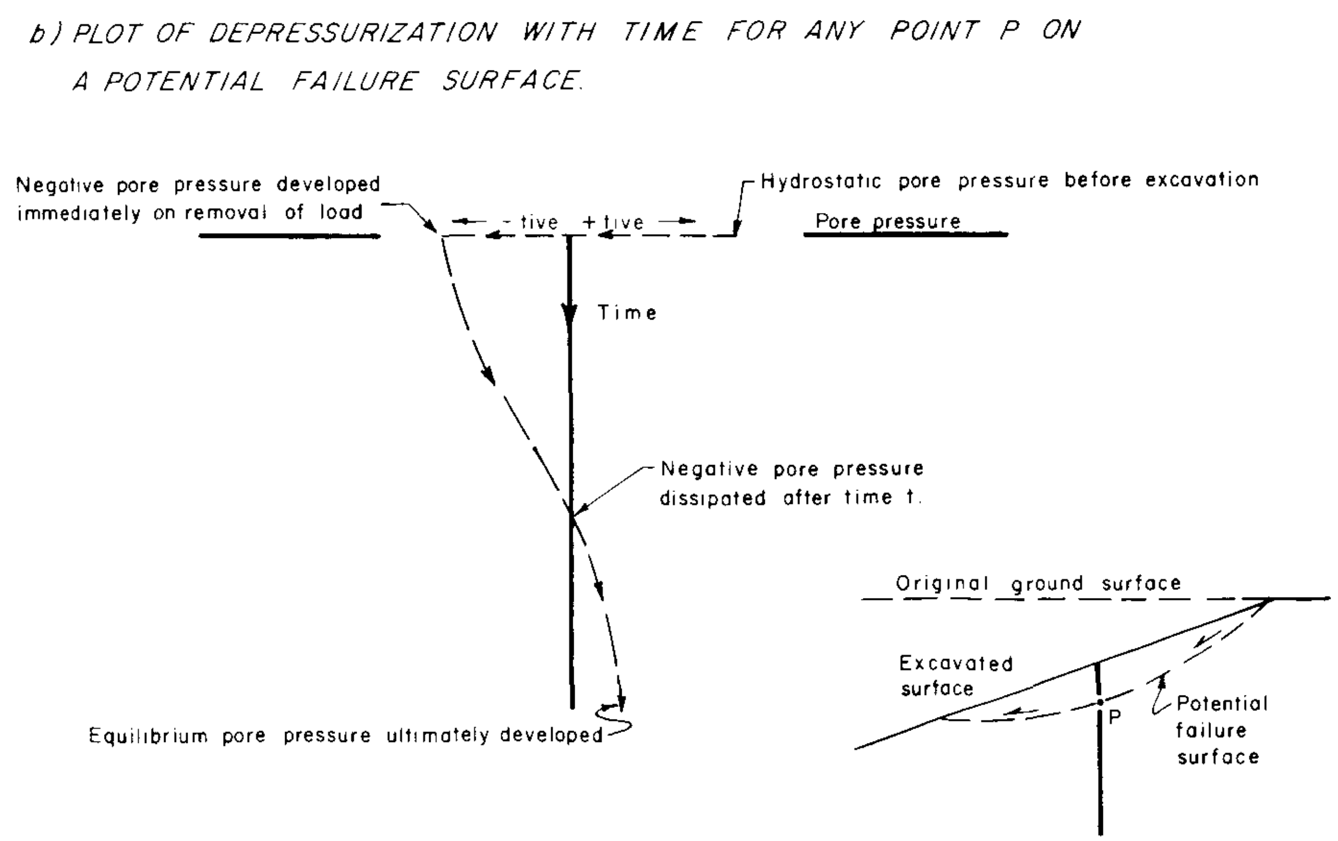
HAT CREEK PROJECT

FIGURE 5-3

PIT SLOPES

SOURCE: Golder Associates



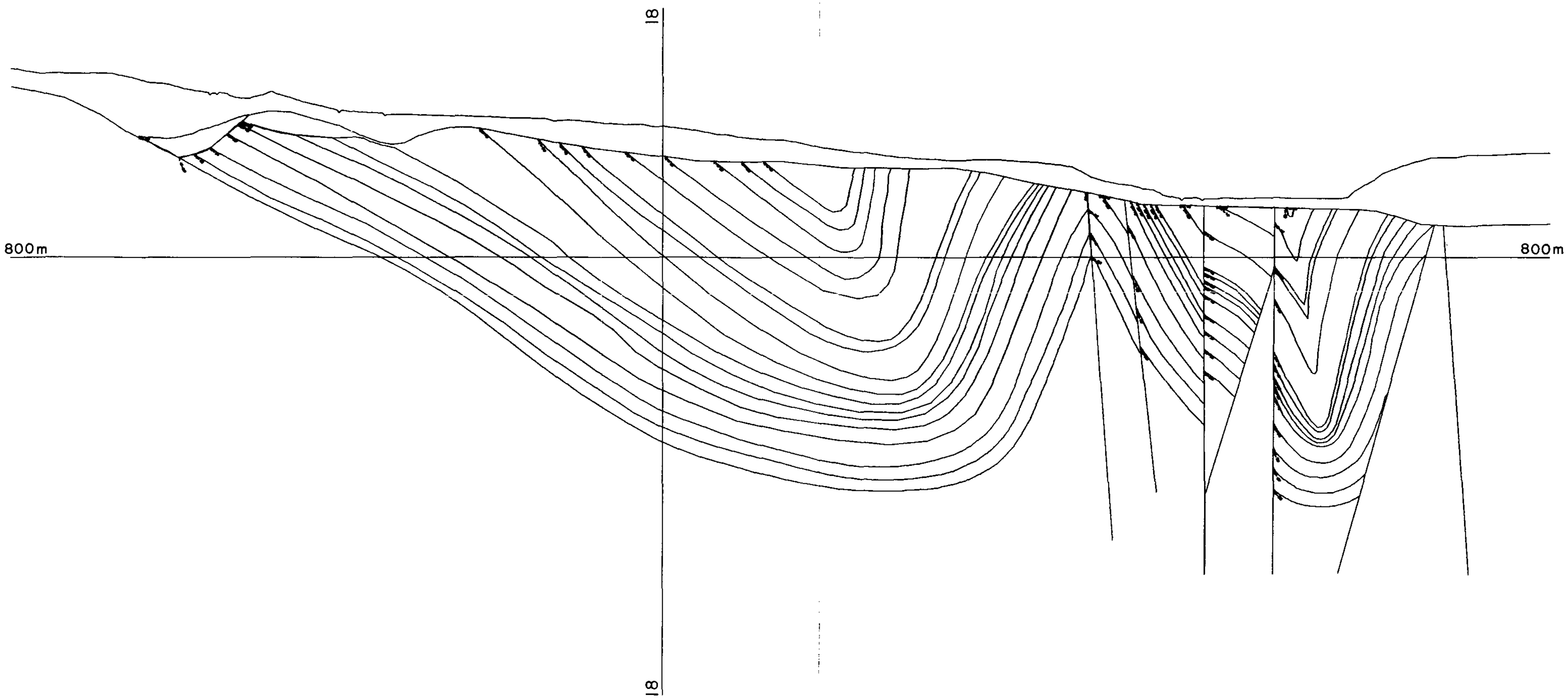


HAT CREEK PROJECT

FIGURE 5-8

DEPRESSURIZATION BY EXCAVATION

SOURCE: Golder Associates



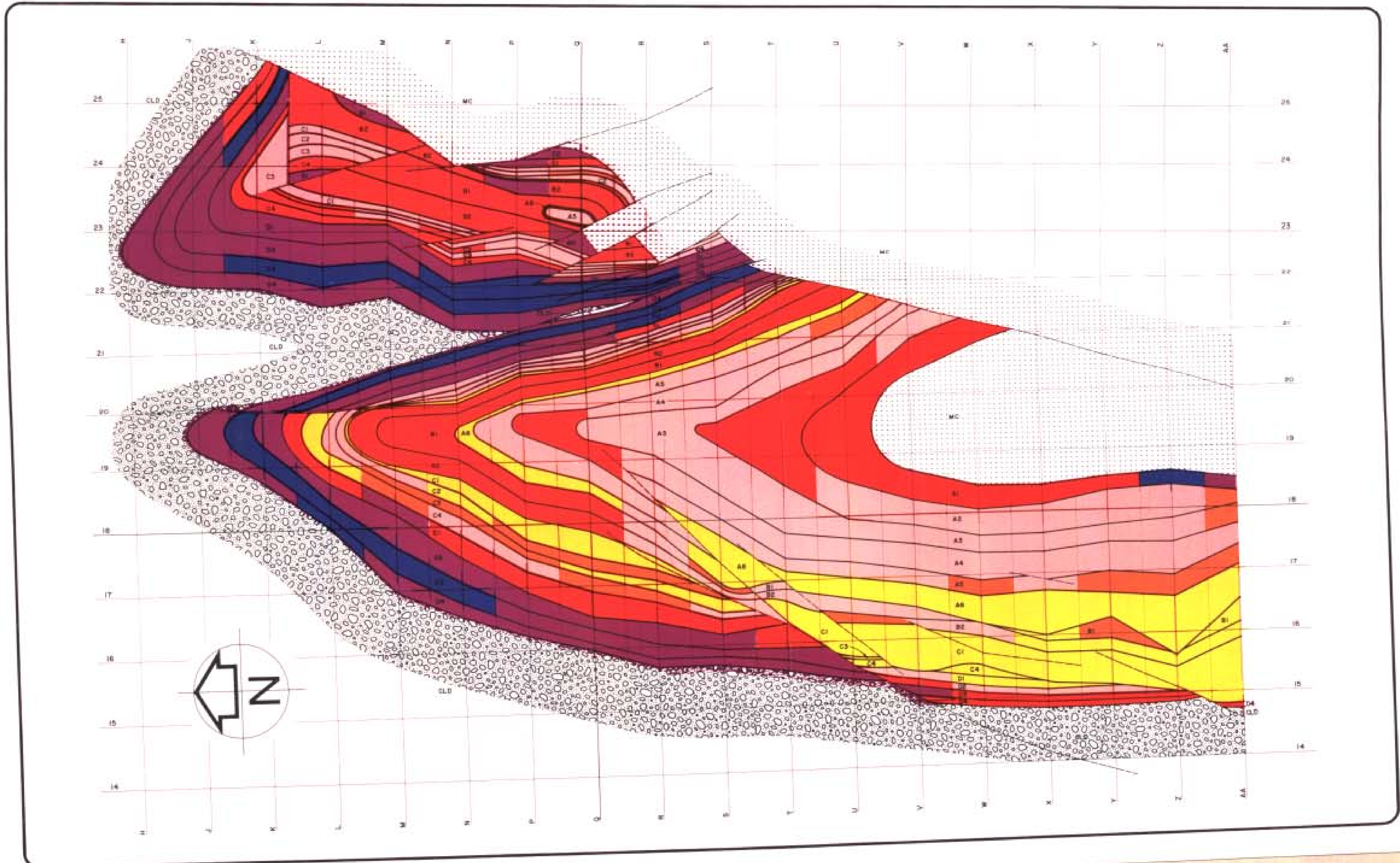
HAT CREEK PROJECT

FIGURE 5-9

Typical VBM Cross-Section Section Q

SECTION DRAWN LOOKING NORTH

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

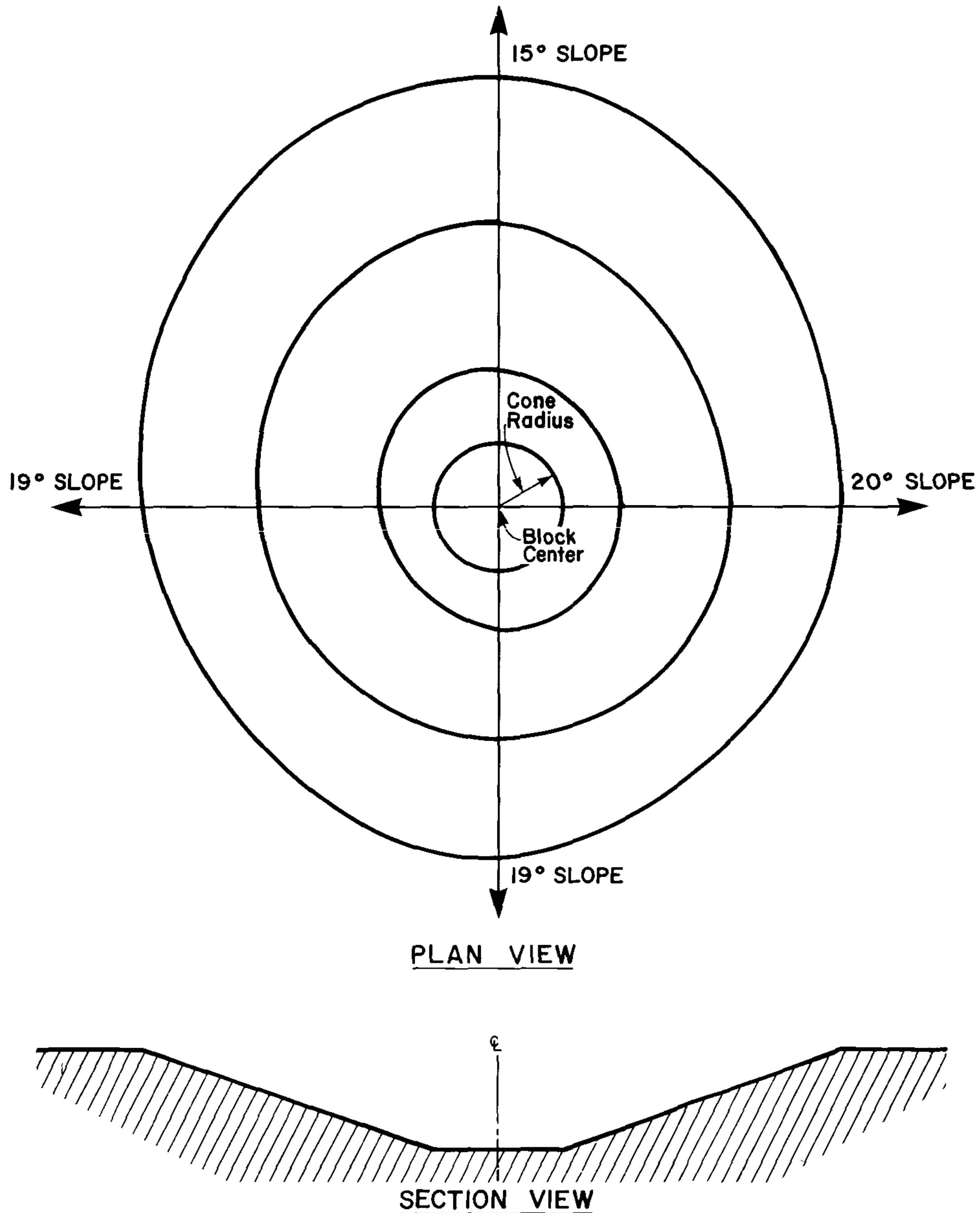
100m 0 100 200 300 400m

HAT CREEK PROJECT

FIGURE 5-10

Bench Plan-Elevation 775

M084 G

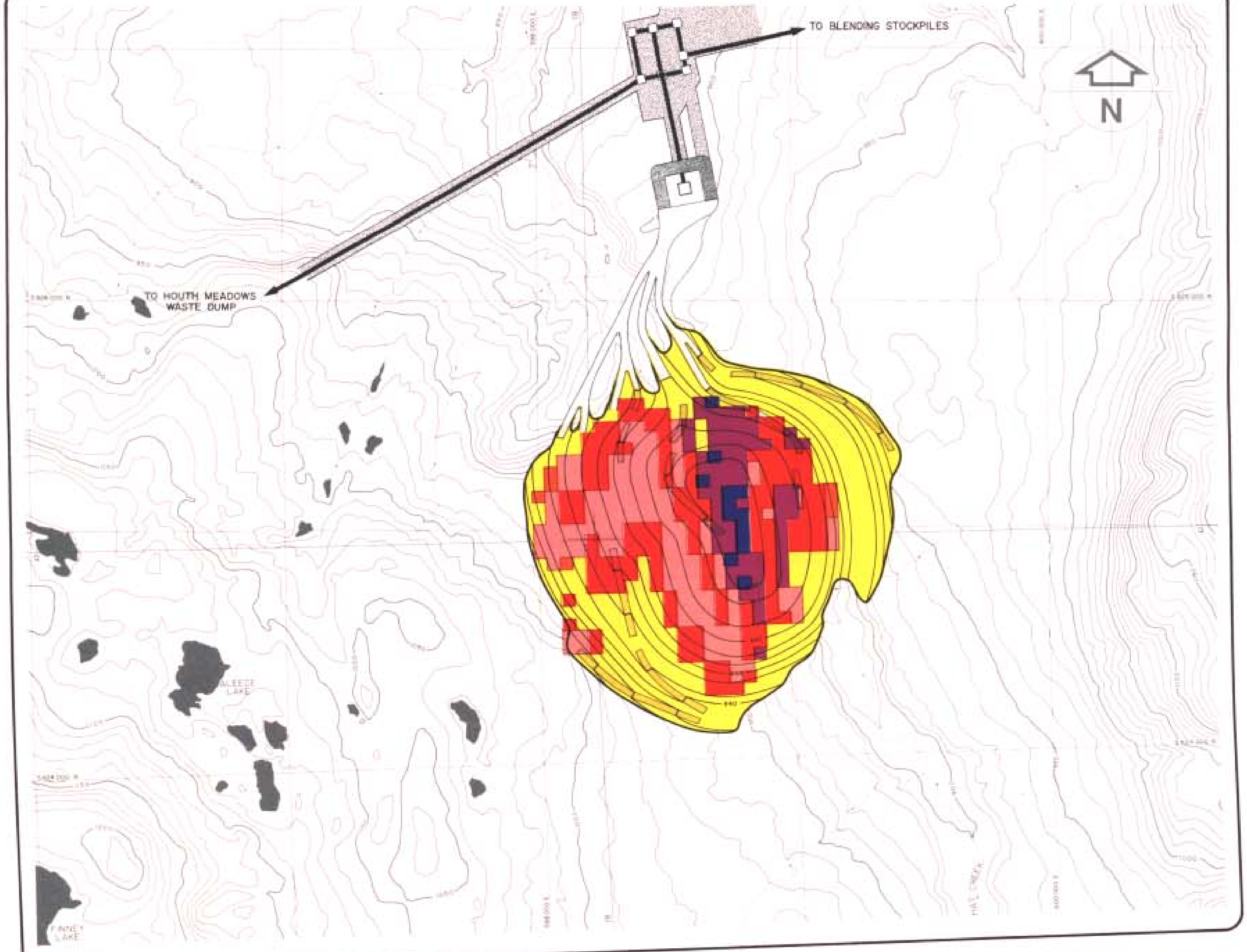


HAT CREEK PROJECT

FIGURE 5-11

Cone Geometry

SOURCE: Mintec, Inc.



LEGEND

SPECIFIC ENERGY RANGES
MJ/kg (DRY BASIS)

(yellow)	< 9.29
(orange)	9.3 — 12.99
(pink)	13.0 — 16.49
(red)	16.5 — 19.99
(purple)	20.0 — 23.49
(blue)	23.5 >

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

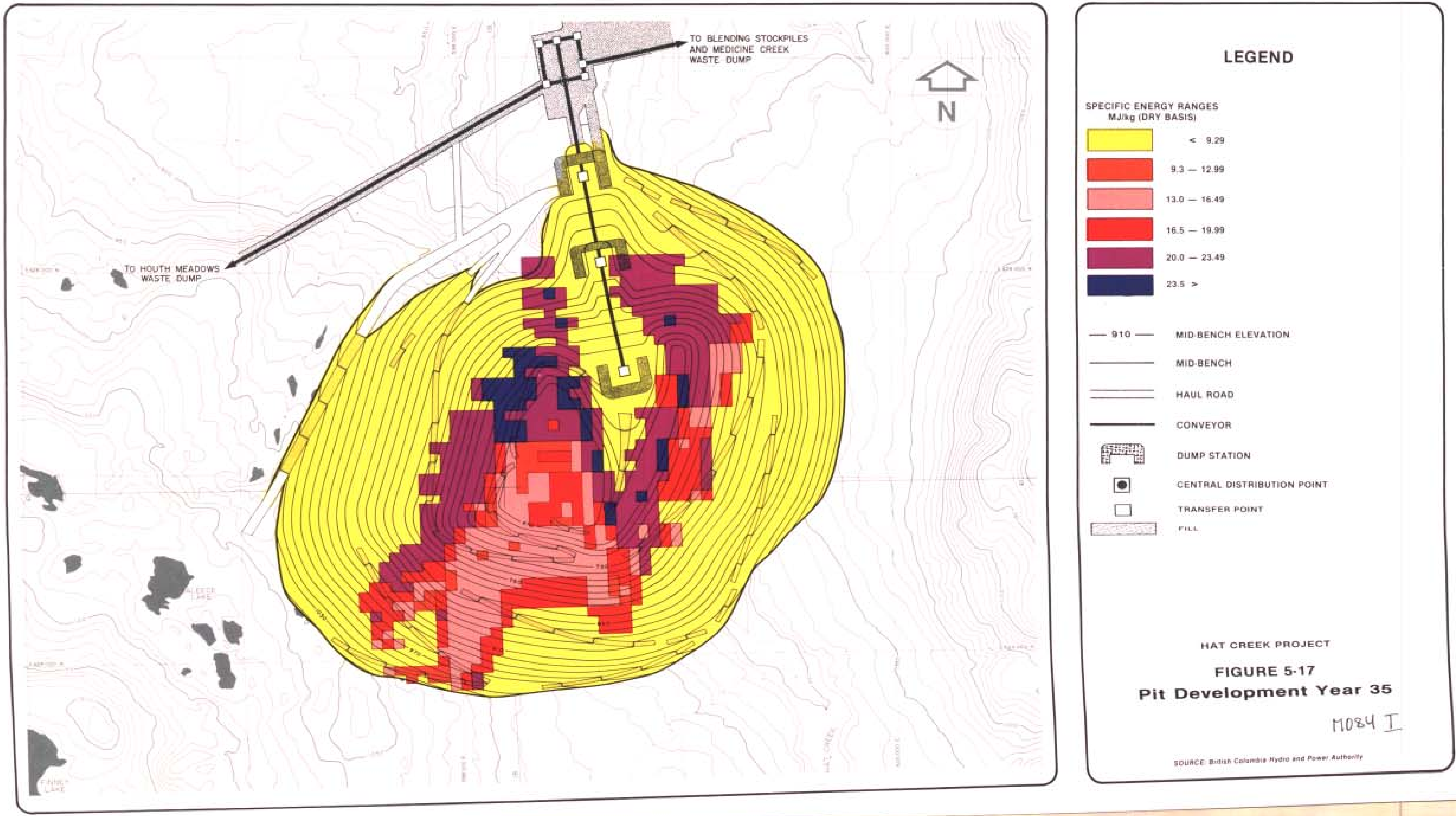
HAT CREEK PROJECT

FIGURE 5-13

Pit Development Year 5

M084 H

SOURCE: British Columbia Hydro and Power Authority

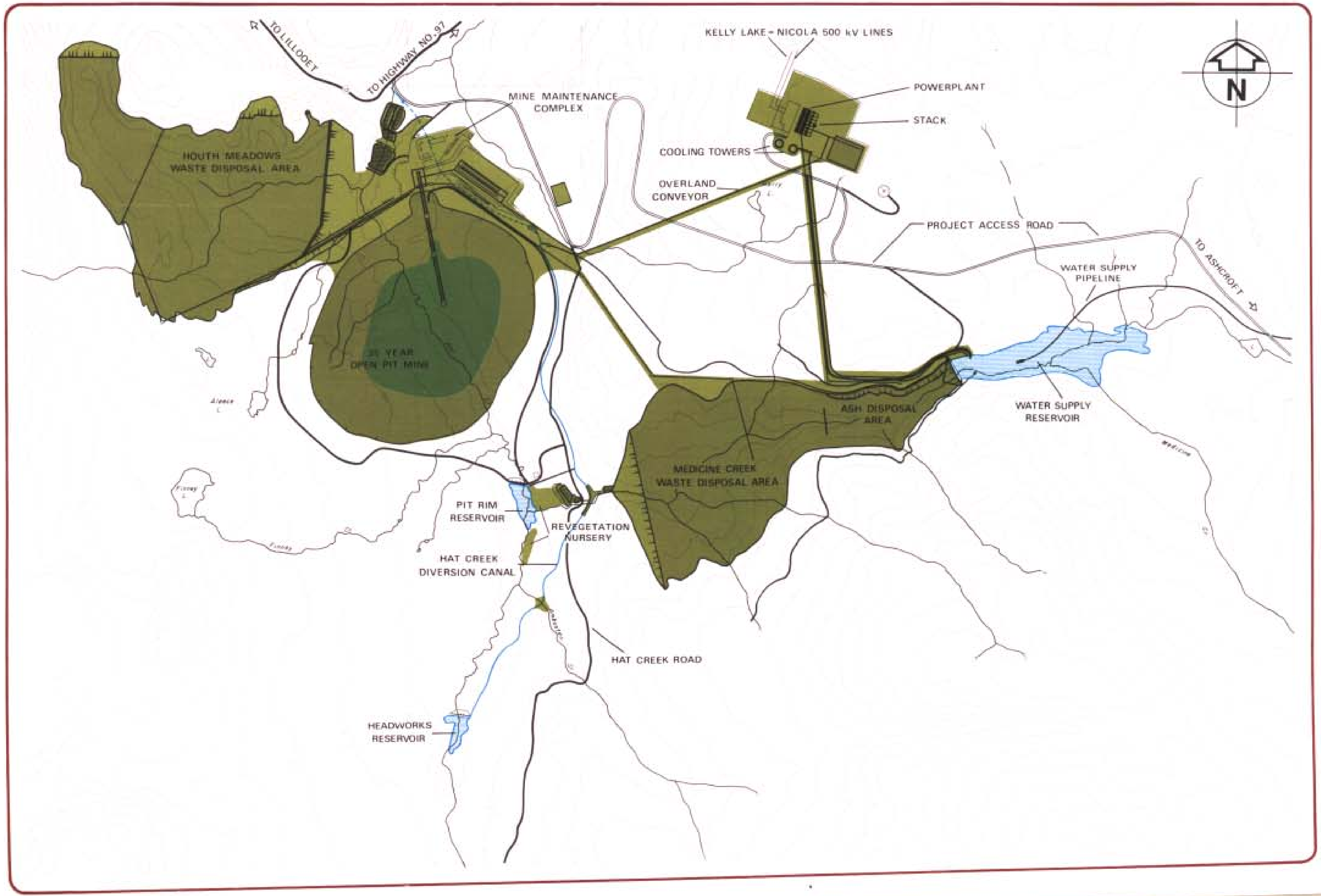


HAT CREEK PROJECT

FIGURE 5-17
Pit Development Year 35

M084 I

SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 11-1

RECLAMATION — YEAR 45

M084 J

SOURCE: Cominco — Monenco Joint Venture (5)

5 COST ESTIMATES

All the figures in this Summary Report were originally numbered for the Main Report. These numbers could not be changed in the time available. The figures have therefore been placed at the end of this section.

LIST OF TABLES

LIST OF FIGURES

SECTION 5

COST ESTIMATES

5.1 DEVELOPMENT OF CAPITAL AND OPERATING COSTS

Capital and operating costs have been developed for the 50-year period covered by the mining plan. These costs, in 1979 dollars, provide for all aspects of the mining operation through to delivery of the coal to the powerplant.

Table 5-1 presents a summary of the capital and operating costs on an annual basis, and Table 5-2 shows these costs by major cost centre.

The Annual Production Schedule (discussed in Section 4.4) is the starting point for costing. When combined with the equipment productivity standards, the equipment requirements for capital cost estimating, and the annual equipment utilization for operating cost calculations, are quickly determined. Standard operating costs were developed for each type of equipment. The major components of the standard costs are: operating labour, operating supplies, maintenance labour, maintenance supplies, and overhead. The standard costs combined with the annual equipment utilization gives the production costs. Figure 14-1 is a flowchart showing the development of cash flow. As a by-product of the standard costs, the manpower requirements can be established - see Table 5-3 "Summary of Manpower Requirements".

Additional capital and operating costs for fixed facilities, the Materials-handling System, environmental protection, overhead, and general mine expense, are developed separately by cost centre.

Appropriate allowances were made for royalties, insurance, taxes, and contingencies. A contractor's allowance of 10% was made in the operating costs to provide for the contractor's overhead and profit should operation of the mine be contracted out. No additional allowance has been made for the additional staff the owner would require for monitoring and control.

5.2 ESTIMATING CRITERIA

The following criteria were also used in developing the project costs:

1. Capital

(1) Major equipment costs were based on manufacturers' budget prices in 1979 dollars and include freight, import duty where applicable, insurance, Provincial sales tax, and erection costs;

(2) Equipment service lives were based partly on manufacturers' recommendations and from a survey of operating mines;

(3) An exchange rate of $1.15/$1.00 U.S./Canadian was used where applicable;

(4) Labour rates were taken from prevailing agreements in the B.C. construction industry.

2. Operating

(1) Staff salaries were developed from a survey conducted in 1978 for the Mining Association of B.C.;

(2) Hourly wage rates were developed from a review of current labour agreements in the B.C. mining industry;

(3) The mine operating schedule is based on 24 hours per day operation for 354 days per year;

(4) Operating shifts are eight hours per day.

5.3 FINANCIAL ANALYSIS

A financial analysis was performed to establish a uniform selling price for coal over the life of the project. The uninflated cash flows were discounted at 3% to arrive at a price for Hat Creek performance coal delivered to the powerplant. The calculation was done twice:

(1) Excluding cost of power consumed by the mine: $7.80 per tonne delivered, which is equivalent to $0.567/GJ or 6.19 mills/kW.h;

(2) Including mine power consumption of 20 mills/kW.h: $8.27 per tonne delivered, which is equivalent to $0.601/GJ or 6.56 mills/kW.h.

TABLE 5-1

SUMMARY OF ANNUAL COSTS, CANADIAN $ OCTOBER 1979

HAT CREEK PROJECT MINING REPORT 1979

Year	Annual Coal Production tonnes $\times 10^6$	Annual Coal Production MJ $\times 10^9$	$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 5-2

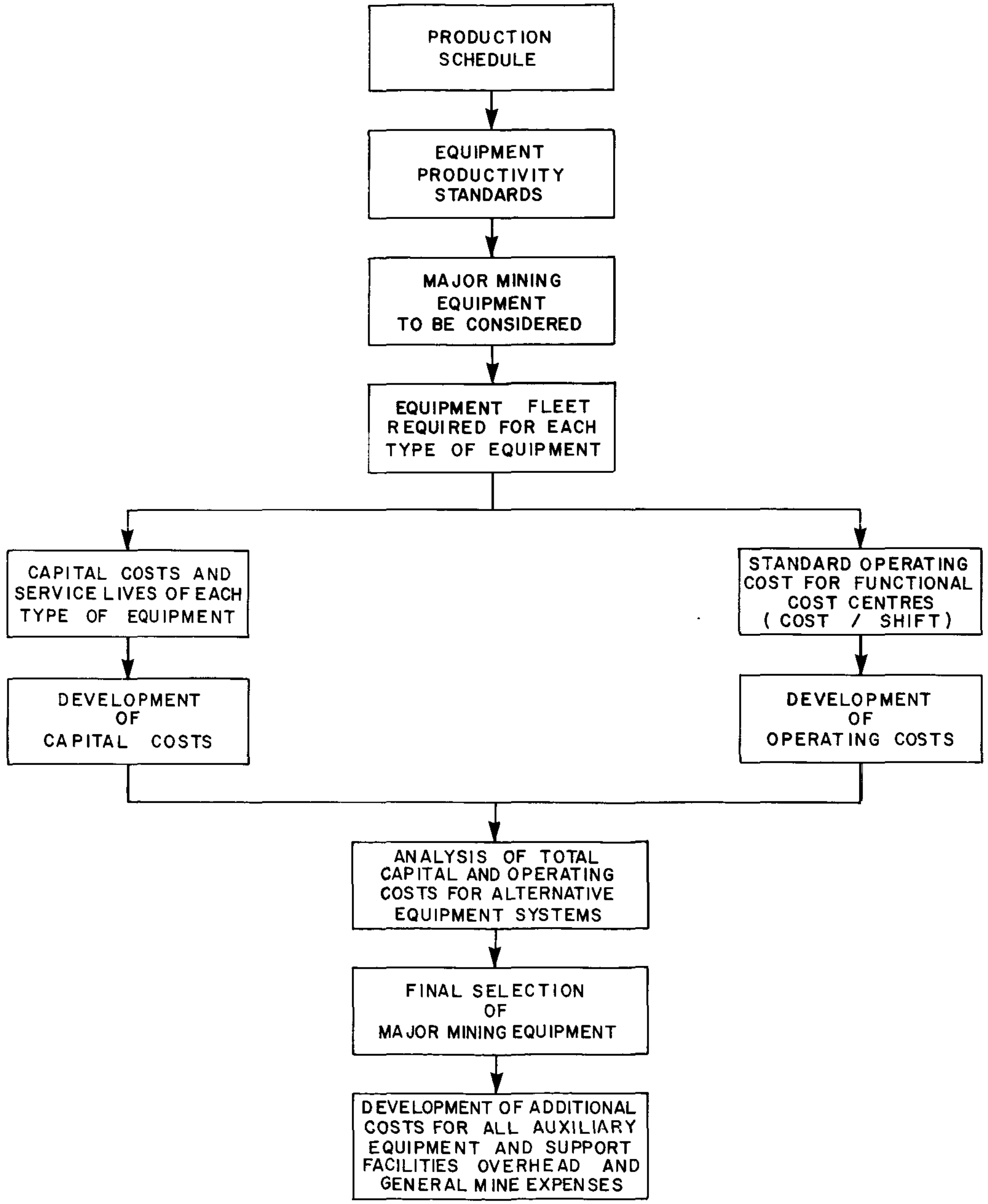
BREAKDOWN OF TOTAL ESTIMATED CAPITAL AND
OPERATING EXPENDITURES BY MAJOR COST CENTRES
($000's October 1979)
Hat Creek Project Mining Report 1979

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 Equipment†	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

TABLE 5-3

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

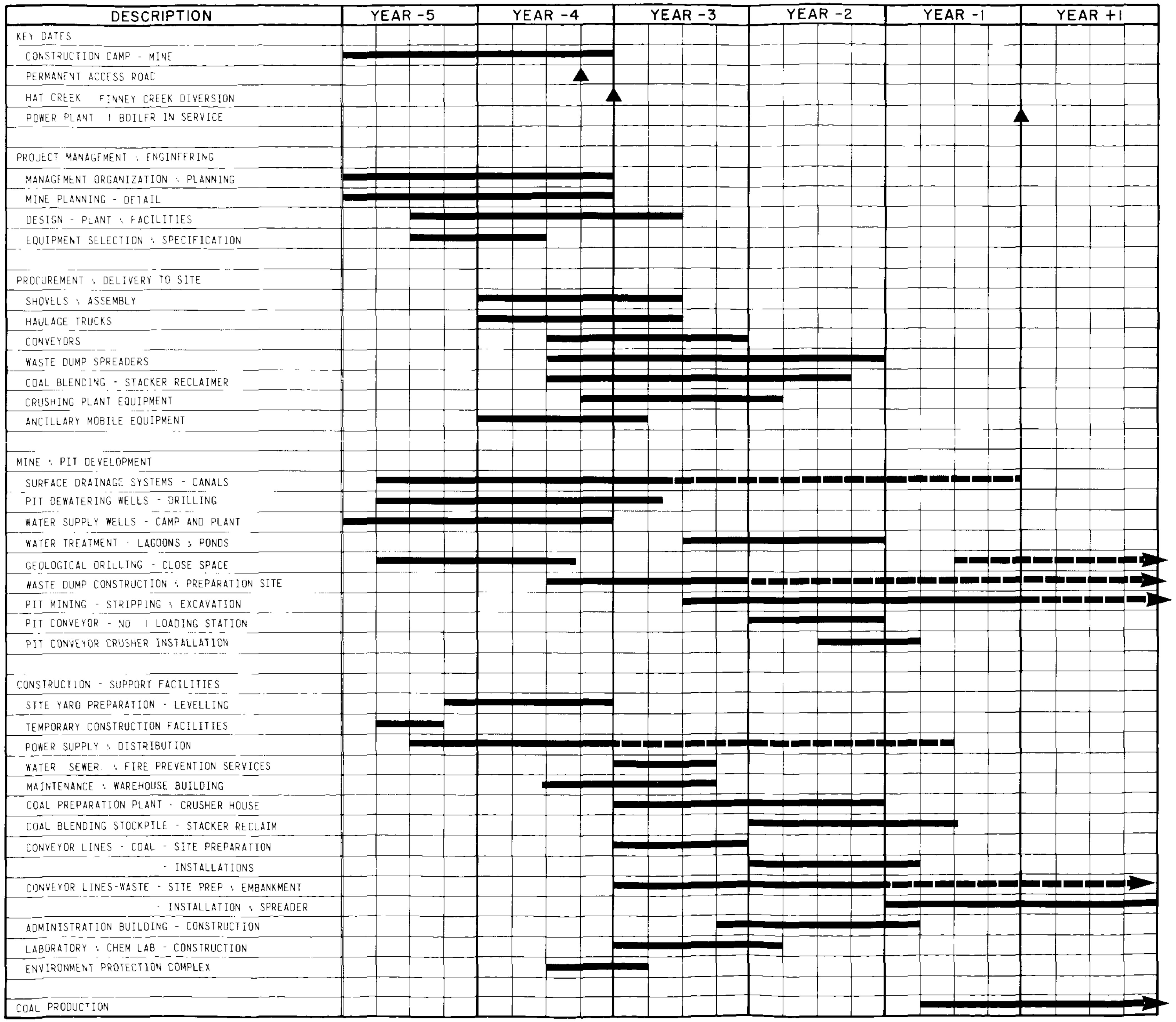


HAT CREEK PROJECT

FIGURE 14-1

Overall Approach to Development of Cash Flow

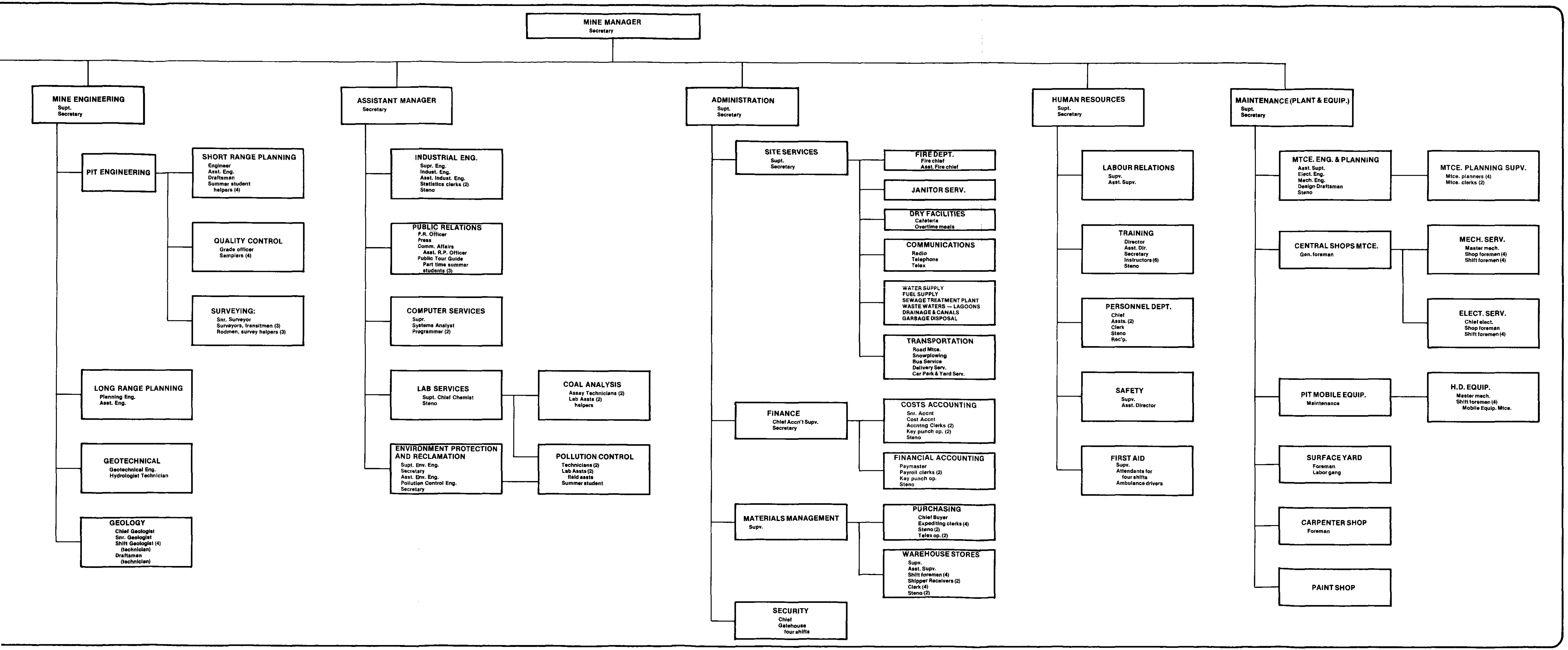
SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 12-1
CONSTRUCTION SCHEDULE

SOURCE: British Columbia Hydro and Power Authority



HAT CREEK PROJECT

FIGURE 13-1

Organization Chart

SOURCE: *British Columbia Hydro and Power Authority*

SECTION 6

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