HAT CREEK PROJECT

MINING FEASIBILITY REPORT

VOLUME III

MINE PLANNING

prepared for
British Columbia Hydro and Power Authority

by
Cominco-Monenco Joint Venture

1978

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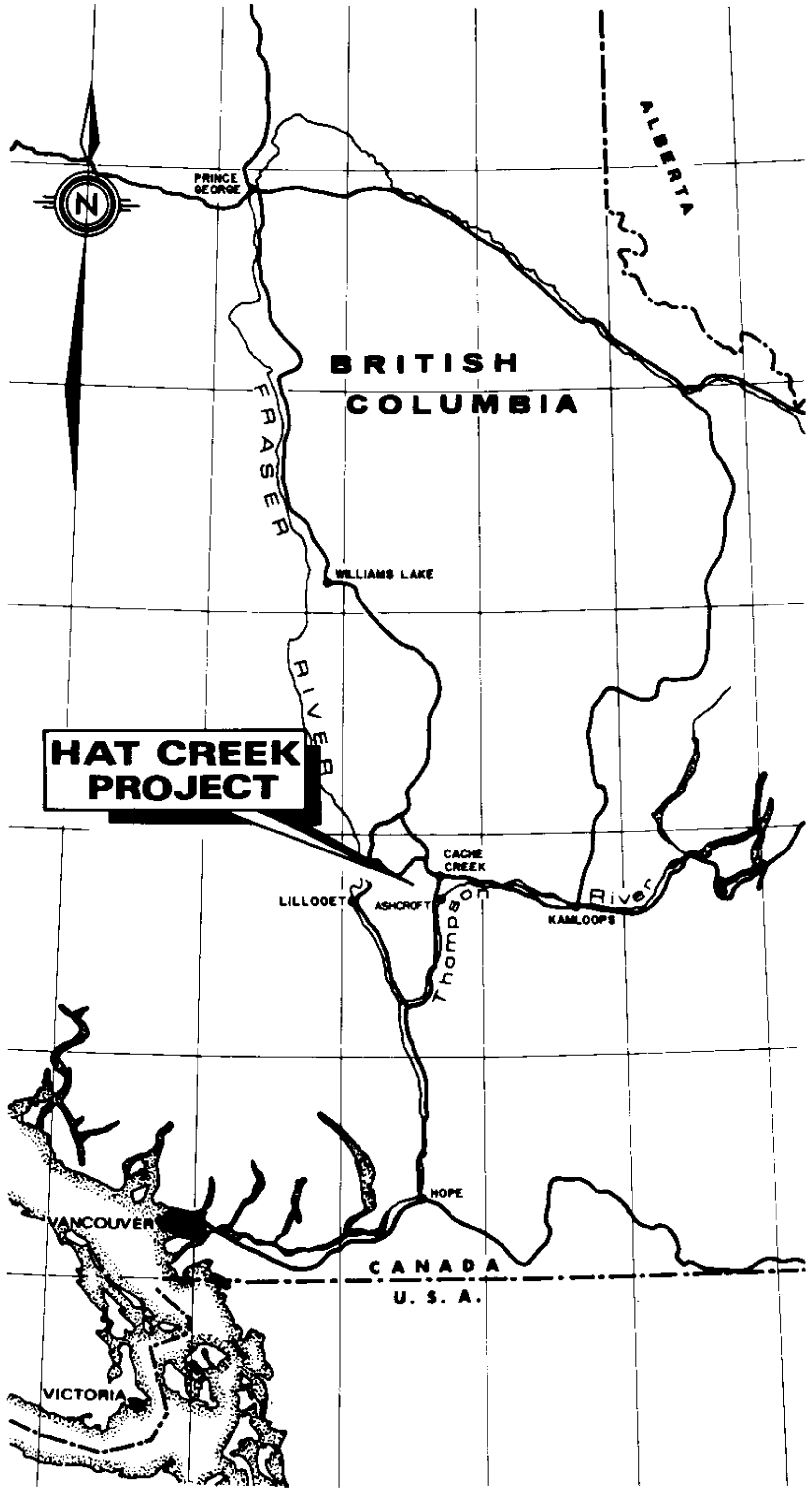

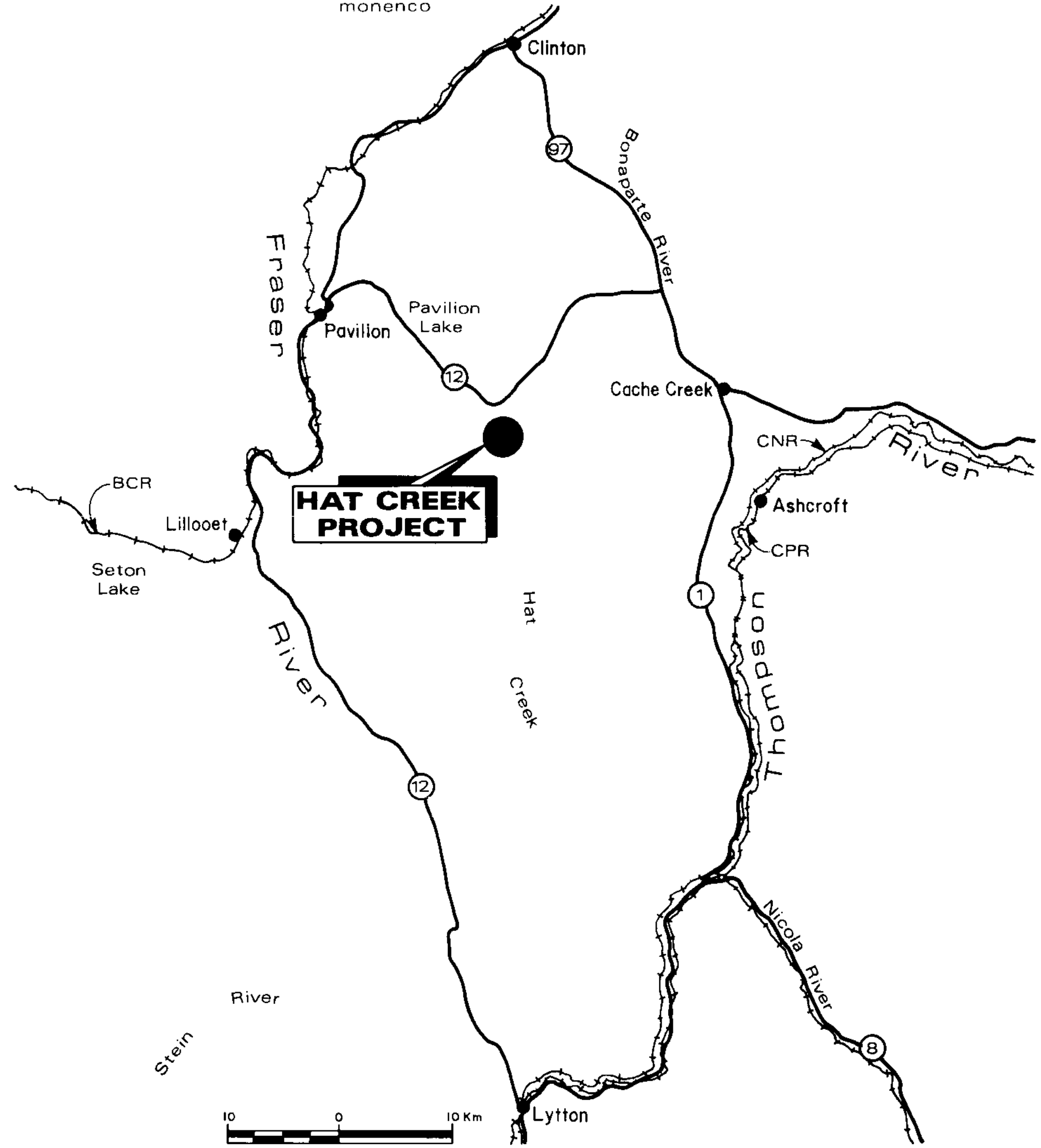

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
PROJECT LOCATION

VOLUME III

MINE PLANNING

TABLE OF CONTENTS

TABLE OF CONTENTS (continued)

TABLE OF CONTENTS (continued)

Page

TABLE OF CONTENTS (continued)

TABLE OF CONTENTS (continued)

LIST OF TABLES

LIST OF TABLES (Continued)

LIST OF FIGURES

LIST OF FIGURES (Continued)

SUMMARY

1. On the basis of studies completed to date, it is recommended that a shovel/truck/conveyor system be adopted for mining of the No. 1 coal deposit at Hat Creek.

   In total, six alternative mining systems were studied at various levels of detail. On the basis of practical operating considerations at Hat Creek, three of the systems were judged to be impractical. Three remaining systems - the recommended shovel/truck/conveyor, the shovel/conveyor, and the bucket weel excavator/conveyor - were evaluated by an order of magnitude comparison for the complete mining facility on the basis of cost advantage and operating efficiency. Results of this comparison indicated that a detailed study be conducted for a shovel/truck/conveyor system and a bucket wheel excavator/conveyor system, the latter being modified to include a combination of shovel/truck. The bucket wheel system was eventually ruled out on the basis that it appeared to have no economic advantage and that it would be relatively inflexible to change should the coal quality or geological structure be different than expected.

2. The mining system recommended for the development of the Hat Creek open pit coal mine would utilize the following major units of equipment at peak production

   7 - electric shovels with 16.8 cubic metre bucket size

   9 - rear dump coal haul trucks of 109-tonne capacity

   18 - rear dump waste haul trucks of 136-tonne capacity

   26,435 metres of conveyors with a belt width of 1200 mm

   2 - waste spreaders with 40 metre discharge booms

   2 - bridge mounted bucket wheel reclaimers

   2 - rail mounted stackers with 53 metre discharge booms

   50 - various sized front-end loaders, bulldozers, graders, drills, scrapers, and small end-dump trucks

Coal and waste, excavated by electric shovel, would be transported by rear dump haulage trucks to one of three unloading stations located at various levels in the pit. The materials would then be transferred onto a centrally located system of three mine conveyors and transported to various destinations out of the pit. Coal would be transported to a crushing and blending facility prior to transfer by conveyor to the generating station. Low-grade coal would be conveyed directly to a stockpile located in an area which would permit easy access for future recovery. Waste materials, transported by conveyor to the Houth Meadows or Medicine Creek disposal sites, would be placed by spreaders behind engineered retaining embankments.

3. The No. 1 Deposit at Hat Creek contains more than adequate mineable reserves of coal to support the proposed 2000 MW generating station over a 35-year period. A comparison of coal reserves, coal quality, and stripping quantities required to support the 35-year period versus that considered available from an expanded pit are summarized as follows:

	Recommended 35-year Pit	Available if Pit is Expanded
Mining depth	632 metres ASL	450 metres ASL
Coal tonnes greater than 4000 Btu/lb	349×10^6	676×10^6
Percentage of total estimated geological reserves in the No. 1 Deposit	48%	93%
Quality of coal		
- calorific value (Btu/lb dry basis)	7327	6804
- ash content (% dry basis)	36.3	39.9
- sulphur content (%)	0.48	0.52
Moisture content of R.O.M. coal (%)	25%	25%
Stripping ratio - bank cubic metres of waste to tonnes of coal	1.3	2.0

4. The mining approach adopted in the early years placed a special emphasis on developing the mine downward in the northeast corner of the deposit to uncover adequate quantities of the highest grade coal available. This approach enabled the mine to supply an average fuel quality through each of the 35 years as close as possible to the overall mine average. The lateral development of the mine was restricted as much as possible to maintain a level, annual material movement over the mine life.

5. A geotechnical evaluation of the Hat Creek site was conducted by a specialist consultant. This, in close consultation with the mining project team, led to the formulation of specific geotechnical parameters which were incorporated in the development of the mine plans and associated waste dumps. The mine wall slope angles adopted for planning of the 35-year pit range from 16 degrees in the area of less materials, to 25 degrees in the competent coal and surficial materials. The waste dumps are proposed to be constructed in 20 metre lifts below the spreader and 15 metre lifts above the spreader, with an angle of repose for the waste materials varying from 27.5 to 34.5 degrees.

   In addition, the geotechnical consultant has suggested that extensive depressurization of groundwater, as originally contemplated by conventional deep wells and pit pumping, may not be possible. However, the concept of stress relief rebound after excavation was investigated, and is considered to offer reasonable potential for the effective depressurization of mine wall slopes. For cost estimating purposes, however, the deep well method of depressurization was adopted as recommended.

6. Coal quality studies resulted in a recommendation that only coal whose calorific value is greater than 4000 Btu/lb should be delivered to the generating station. Coal whose calorific value is less than 4000 Btu/lb but greater than 3000 Btu/lb would be classified as low-grade coal and stockpiled separately from waste materials. This would allow for possible future recovery, should improved boiler technology allow the use of this as fuel. Material whose calorific value is less than 3000 Btu/lb would be placed in waste dumps. A summary of the various quantities and qualities of coal for the 35-year pit is as follows:

R.O.M. coal greater than 4000 Btu/lb	82%	- 344 million tonnes
Low-grade coal from 3000 to 4000 Btu/lb	4%	- 16 million tonnes
Waste coal less than 3000 Btu/lb	14%	- 59 million tonnes
Total	100%	419 million tonnes

On the basis of in situ heat value, 95% occurs within the run-of-mine coal over 4000 Btu/lb, 2% occurs in the low-grade coal, and 3% occurs within the waste materials.

7. Preliminary studies were carried out to determine the potential fuel quality improvement resulting from the removal of waste partings less than five metres thick in the A-Zone coal. The results of the work demonstrated that the benefits of partings removal were small and should not be incorporated in the current study. They should however be further evaluated after the mine development has commenced and more detailed information is available as to the location, attitude, thickness, colour, and density of the partings. The evaluation would also include an economic analysis of partings removal, tested under operating conditions.

8. The recommended mining system offers a potential to achieve over the life of the mine an average level of coal quality of 7327 Btu/lb dry basis. The variation in the coal quality over the life of the mine, on a weekly basis, is forecasted to fall between 7000 and 7800 Btu/lb. To assist in achieving this objective an extensive crushing, sampling, and blending facility has been provided. In addition, however, careful scheduling of coal excavation and grade control in the pit will be required.

SECTION 1

GENERAL APPROACH

1.1 OBJECTIVES

The objectives of the mine planning portion of the feasibility study were to develop an optimum mine plan which would:

(a) provide a reliable source of fuel over the projected 35-year life of the 2000 MW thermal plant;

(b) deliver a consistent fuel quality at least cost over the 35-year life;

(c) efficiently utilize the total coal resource, with particular regard to coal conservation and future development;

(d) utilize a mining system that is proven and dependable;

(e) place special emphasis on the safety of men and equipment, with particular regard to the stability of mine wall slopes and waste dumps; and

(f) be optimized to the point where detailed engineering could immediately proceed upon project commitment.

1.2 ALTERNATIVES CONSIDERED

A number of alternative mine development methods, equipment combinations, and coal quality assessments were evaluated. Principal aspects considered were:

(a) pit configurations;

(b) two alternative exits from the pit - north and south;

(c) six alternative equipment systems comprising:

- shovel/truck
- shovel/truck conveyor
- shovel/conveyor
- bucket wheel excavator/conveyor
- continuous excavator/truck and/or conveyor
- dragline/truck and/or conveyor;

(d) differing degrees of partings removal and the effect on delivered coal quality;

(e) selection of coal quality cut-off grades; and

(f) alternative coal beneficiation systems.

1.3 PLANNING PROCESS

Following an initial review of all available data, interpretations of geological and coal quality data were completed; basic geological data were then incorporated into the MEPS computer program. As these data were progressively refined and results of the 1977 drilling program became available, three different computer models were developed and labelled Phase I, II, and III, respectively.

The Phase I computer model presented the deposit by four coal zones, designated A, B, C, and D and two waste zones, A-2 and C-1.

The Phase II computer model subsequently refined the interpretation by dividing the A, B, C, and D coal zones of the Phase I model into four, two, two, and four coal subzones, respectively. This resulted in a total of 12 subzones of coal and two subzones of waste. The model was further subdivided into 14 geographic sectors to aid in detailed manual mine planning.

The Phase III computer model incorporated the 1977 drilling and quality data, new topographic data, revised coal densities, and smaller grid calculation size, and identified two additional coal subzones within the previous C-1 waste zone.

Most detailed mine planning of the alternative systems utilized the Phase II computer model data. Detailed mine plans of the recommended system were adjusted to reflect Phase III data.

The output from the Phase II computer model was used to prepare bench plans at 15-metre vertical increments throughout the deposit showing sector and subzone boundaries, fault zones, and surficial/bedrock boundaries, with quantities and qualities for coal and waste reported for each 75 metre block. It was then possible to develop manual pit designs to meet grade criteria, and to investigate the effects on grade and annual materials quantities of different mine development approaches. Once a pit outline had been developed, it was digitized and detailed final quantities were calculated by the computer.

Layouts, including waste dumps, conveying, and spreading arrangements were prepared for the alternative mining systems. Three of the original six alternative systems were eliminated from further investigation following an evaluation of the operating and economic considerations. Of the remaining systems - shovel/truck/conveyor, shovel/conveyor, and bucket wheel excavator - an order of magnitude comparison for operating difficulty and minimum cost was conducted. The bucket wheel system was ruled out on the basis of operating considerations and it was decided that a detailed cost comparison would be carried out for a shovel/truck/conveyor scheme and a combined bucket wheel/shovel/truck/conveyor scheme.

In parallel with the economic studies being carried out, work also proceeded on the determination of: the sensitivity of coal quality to selective mining, cut-off grades, short term quality variation, and blending and washing systems. As a result of these studies, it was possible to adopt quality criteria for final mine design and to develop a delivered fuel specification.

Coal beneficiation was found to be uneconomic and final, annual rates of production for the mine were estimated on the calorific value of blended run-of-mine (R.O.M.) coal.

Other aspects considered in the identification of a preferred mining system and selection of equipment included geological and geotechnical criteria, quality constraints, material types, operating regime, environmental protection and reclamation.

1.4 RECOMMENDED SYSTEM

Following extensive study of the various alternatives described previously, it was concluded that a shovel/truck/conveyor combination be recommended as the preferred mining system for use on the Hat Creek No. 1 Deposit. In light of the unique physical conditions present at Upper Hat Creek, it is believed that the shovel/truck/conveyor system would best suit the required operating demands and that a cost advantage would be realized.

The recommended system (refer Figure 5-9) incorporates the following major concepts.

(a) a northern exit from the mine with three conveyor systems for waste, coal, and construction material/low-grade coal, respectively;

(b) a shovel/truck system working 15-metre high benches to feed into unloading stations located over the central conveyor system;

(c) an approach to pit development which allows the mining of an average grade of coal and an average annual quantity of material over the 35-year life;

(d) the sufficient and continuous exposure of better-grade D-Zone coal necessary for coal blending;

(e) conveying and spreading systems to dispose of waste in the Houth Meadows and Medicine Creek dump areas;

(f) a crushing, sampling, and blending facility adjacent to the mine services area close to the northern pit exit;

(g) separate low-grade coal stockpile ranging from 3000 to 4000 Btu/lb (dry basis) for future use; and

(h) delivering blended coal to the generating station by overland conveyor.

A description of the geological setting, geotechnical considerations and fuel requirements is provided within Sections 2, 3, and 4, respectively of this volume. Mine development planning and a detailed description of the preferred mining system are contained within Sections 5 and 6; evaluation of alternative mining systems considered is detailed within Section 7.

SECTION 2

GEOLOGICAL CONSIDERATIONS

2.1 GEOLOGICAL STRUCTURES

A complete description of the No. 1 Deposit is contained within Volume II, Geology and Coal Quality, of this report. The salient features that influence mine planning include:

(a) the series of parallel north/south trending open folds that plunge southwards at a shallow angle, but to great depths;

(b) daylighting of some of the better-quality coal at the northeast end of the deposit as outcrops and subcrops under relatively shallow overburden;

(c) two major faults that first dislocate and then truncate the coal structures on the east side of the deposit;

(d) the location of the better-quality coal in the stratigraphically lowest coal unit in the deposit;

(e) large quantities of construction grade surficial materials which overlie coal on the east side of the deposit; and

(f) slide materials on the surface and in the footwall rocks on the northwest side of the deposit. The active slide has apparently also developed in the beds immediately above the conglomerate which is present at depth in the area. This slide below the current active slide is probably still actively receding.

2.2 DISTRIBUTION OF COAL BY GRADE

Investigation early in the study suggested a general east/west trending of calorific values within each of the three lowest coal zones. Trending in A-Zone was found to be less significant. It is also recognized that sample data from individual drill holes may not represent the most realistic local values. A volume and grade calculation procedure was therefore developed to handle the analytical data in a manner considered to be statistically acceptable (refer Section 5.6, of this volume, and Section 3, Volume II).

The following conclusions have been drawn from analysis of the distribution of coal by grade throughout the vertical and horizontal extent of the deposit:

(a) The best grade coal (D-Zone) lies at the bottom of the four-zone stratigraphic sequence. In the portion of the total deposit mined in this project, it is estimated to contain about 42% of the coal tonnage and about 55% of the heat value.

(b) Coal quality improves significantly from west to east in the three lower zones (B-1, C-2, D-1). The trending is less pronounced in A-Zone. A statistical approach termed Trend Surface Analysis was used to determine this trending.

(c) There is abundant D-Zone coal of higher than average quality east of the central anticlinal structure. The coal underlies the existing bed of Hat Creek in the northern part of the deposit. East of Hat Creek in this region good grade coal subcrops below surficial materials which are more than 100-metres thick.

(d) On the west limb of the syncline, a significant portion of the coal in the middle zones, B-1 and C-2, is shaled-out and is considered unusable. The western portions of these zones are therefore categorized as either "seam waste" or low-grade coal, depending on their relative coal quality.

Consideration of these grade factors, together with structural characteristics and distribution of waste materials, has resulted in the mining approach outlined in Section 5.

SECTION 3

GEOTECHNICAL CONSIDERATIONS

3.1 INTRODUCTION

A review of the geotechnical data and recommendations available at the commencement of the feasibility study indicated that a conservative approach had been adopted, i.e., relatively risk-free, shallow pit slope angles. This related in part to the incomplete state of field studies at that time. Geotechnical consultants had not been able to firmly determine the likelihood of ground subsidence, or other anticipated operating problems involved in excavation and maintenance of the pit, or in handling of the variety of mine production materials.

During the course of the feasibility study, a close liaison was maintained with the geotechnical consultants, who continued the field study program. The conservative geotechnical approach originally adopted was modified and the concept of acceptable risk introduced. Details of consultant recommendations and the subsequent effects on mine planning are presented in the following Sections 3.2 to 3.6 which deal with identification, material parameters, and handling characteristics of waste materials; slope stability in pit walls and waste dumps; localized ground conditions; and hydrology.

3.2 IDENTIFICATION OF MATERIALS

Descriptions of typical surficial and bedrock materials that could be encountered over the life of the 35-year project are provided on Tables 3-1 and 3-2, respectively. Estimated percentages of each category by volume compared to the total volume of waste to be removed are also shown to indicate their relative significance.

As heavy localized concentrations of large boulders will tend to lower the productivity of excavating equipment, it was considered necessary to estimate the frequency and size range of boulders contained in the surficial materials. For purposes of investigation, a boulder was defined as a discrete mass of hard material exceeding 0.3 cubic metres in size. Estimated frequency in the mine area is (Golder Associates):

(a) East of Hat Creek the volume of boulders compared to all other surficials is 3 in 100,000. The frequency in till is higher than in sands and gravels. Boulders of significant size will also be encountered near the surface in colluvium, particularly at the base of the volcanic cliffs at the eastern limit of the pit.

(b) West of Hat Creek the ratio of boulders by volume is estimated to be 40 to 100,000 in till and slide debris. A frequency increase in the order of one magnitude is expected in local surface areas and buried stream channels.

These estimates are considered to be conservative and are expected to be somewhat less during actual mining operations.

In general, boulders may be encountered in all surficial deposits except lacustrine sediments. The investigative analysis included glacial tills, glaciofluvial deposits, slide debris, colluvium, alluvium, and burn agglomerates. Data were derived from the following sources:

- 1976 Becker hammer drilling program
- 1976/77 Golder Associates rotary cone drilling program
- 1977 mapping in "Trench A"
- 1977 B.C. Hydro report entitled "Surface Boulder Frequency Survey of the No. 1 Coal Deposit at Hat Creek"

TABLE 3-1

Description of Surficial Materials as Supplied by Golder Associates
Hat Creek Project Mining Feasibility Report 1978

Type	Description	Location	Range of Permeability cm/sec	Moisture Content on Dry Weight Basis	Unlaxial Strength	Atterberg Limits	General Materials Classifications and Volumes by CMJV	Total Materials
Alluvium	Rounded sands and gravels probably with silt interbeds as seen in Trench B. Mostly reworked glacials.	Predominantly in Hat Creek Valley bottom.	10^{-4} - 10^{-2}	Depends on drainage.	Usually not cohesive.	Usually N.P., but could go up to about LL = 40 PL = 15 (No test results.)	Pervious	
Colluvium	Coarse, angular, roughly bedded perhaps with variable proportion of fines formed on slopes by erosion. May comprise volcanics, limestone, or granodiorite.	Widespread at base of steeper slopes.	10^{-5} - 10^{-2}	11 - 60% Highly dependent on composition. Average 30%.	100-500 kPa, depending on composition.	Varies over full range because of composition variability.	Material .00 183 x 10^6 BCM	27
Glacio-fluvial Deposits	Interbedded rounded/subrounded sands and sandy gravels with cobbles and boulders up to 24" dia. (approx.). Much variation in grading. Some interbedded tills. Glacial meltwater deposit.	East side of valley.	10^{-4} - 10^{-3}	Depends on drainage.	Non-cohesive.	N.P.		
Baked Zone	Varies from an irregular mass of red-brown partly-fused claystone and siltstone with some coal to well bedded slightly baked in situ Coldwater materials.	Dry Lake area. May be obscured by glacial or slide deposits in subcrop on west side.	Highly variable.	Insufficient data for characterization; properties highly variable.			Baked Zone 9.37 x 10^5 BCM	1
Lacustrine Deposits	Bedded silts with coarse sand and occasional gravel may be also clayey, laminated and/or highly disturbed. Overconsolidated. Glacial origin.	Locally throughout glacial deposits. South Meadows embankment foundations.	10^{-5} - 10^{-4}	18. - 32 Average 25	200-500 kPa	LL = 48 PL = 26 (Average from a small number of tests)	Hard Pan &	
Till	Glacial deposit composed of cobbles and gravels with occasional boulders up to 36" dia. maximum but generally much less, in a matrix of sand, silt, and clay. Locally variable, depending on matrix. Seen in base of Trench C.	West side of valley.	10^{-8} - 10^{-6}	15. - 50 Average 26%	0-300 kPa	LL = 86 PL = 42 (Average from a small number of tests)	Consolidated Till 22.00 x 10^6 BCM	3
Slide Debris (Active)	As above, but some softer zones. Currently unstable.	Active slide in NW and minor slides elsewhere in west.	Not known.	11 - 60% Highly dependent on composition. Average 30%.	100-500 kPa, depending on composition.	Varies over full range because of composition variability.	Impervious	
Slide Debris (Stable)	Composed of variable assortment of till and Coldwater sediments often in a bentonite matrix. Seen in upper part of Trenches A and C. Post glacial.	West side of valley especially NW.	Not known.	11% - 60% Highly dependent on composition. Average 30%.	100-500 kPa, depending on composition.	Varies over full range because of composition variability.	Material 66.02 x 10^6 BCM	10

TABLE 3-2

Description of Rock Materials as Supplied by Golder Associates
Hat Creek Project Mining Feasibility Report 1978

Type	Description	Location	Range of Permeability cm/sec	Moisture Content on Dry Weight Basis	Uniaxial Strength	Atterberg Limits	General Material Classification	% Total Materials
Upper Claystone/Siltstone	Very weak to moderately weak clay-rich rocks in which bedding often hard to discern. Rock breaks along joints. Where softened or reworked, material highly plastic and tenacious. Zones of shearing and brecciation. Possibly tuffaceous near margins of basin. Generally dark green or dark brown colour.	Stratigraphically above the coal. Subcrops in an arc from NE to SW in final pit slopes.	10^{-8}	13% - 32% Average 24% May tend to decrease with depth from ~29% at subcrop to ~18%, 150 m deeper.	400 - 12 000 kPa Average 3700 kPa. May tend to increase from ~1000 kPa to ~8000 kPa after 150 m.	LL = 94 PL = 35 (Average)		
Lower Claystone/Siltstone	As upper claystone/siltstone, but contains many interbeds of sandstone and conglomerate. Generally light greenish colour.	West and NW pit slopes. Stratigraphically below the coal. Also occurs as interbeds in the conglomerate.	10^{-9}	23% - 70% Average 31%	600 - 3500 kPa As interbeds in conglomerate, 3500 - 7000 kPa.	LL = 143 PL = 34 (Average)		
Sandstone	Varies from weak silty sandstone through to moderately strong fine grained conglomerate. Matrix usually composed of silt/clay and granular material may be tuffaceous and weak. Locally cemented especially immediately below the coal. Generally greenish.	West and NW pit slopes. Stratigraphically below the coal. Forms interbeds in lower claystone/siltstone and in conglomerate.	10^{-8}	19% - 32 Average 25%	Some tendency to increase from ~1000 kPa to ~10 000 kPa and vary similarly with depth.	LL = 80 PL = 30 (Based on only a few results)	BEDROCK WASTE 133.63×10^6 BCM	20
Conglomerate	Highly variable in character depending on relative proportions of granular material and matrix. Coarse gravel fragments rounded to subrounded but also angular where tuffaceous. Matrix may be bentonitic. Often calcite cemented. Not yet seen in outcrop or excavation. Contains interbeds of siltstone and sandstone.	South abutment of Houth Meadows Embankment. Forms ridge between Houth Meadows and pit. Also occurs as interbeds in lower claystone/siltstone.	10^{-8}	Average 15%, based on few test results. Note that interbeds will raise overall average.	Depends on cementation; up to 43 600 kPa has been measured locally. Some zones almost uncemented.	LL = 60 PL = 27 (Based on very few results)		
Volcanics	Includes an assortment of basalts, dacites, Rhyolites, agglomerates, breccias and tuffs. Closely jointed.	East and west of pit.	10^{-8} - 10^{-5}*	No data.	Up to 23 000 kPa has been measured. Strength may often be much greater.		IN-SEAM WASTE 27.52×10^6 BCM; IMPERVIOUS WEAK MATERIALS 161.15×10^6 BCM	4
Coal Interbeds	Generally thinly bedded claystone/siltstone of moderate plasticity. Some bentonitic material in A-Zone and near margins of basin. May be highly sheared or brecciated.	Centre of pit and limited area in SW wall.	10^{-8}	32% - 36% Average 23%	No data.	LL = 59 PL = 33 (Average)	COAL 243.28×10^6 BCM	35
Coal	Thinly bedded moderately strong but highly fractured. Interbedded with siltstone partings and beds, often highly sheared. Some cleating. Much variation from clean - dirty coal except in D-Zone. Some zones of complete fragmentation.	Centre of pit and limited area in SW wall.	10^{-7} - 10^{-4}		1000 - 17 000 kPa			

* Locally may be as high as 10^{-4} cm/sec.

3.3 MATERIAL PARAMETERS

331 SPECIFIC GRAVITY

A linear relationship was drawn between ash (as a percentage of coal on a dry weight basis) and the specific gravity of Hat Creek coal. The following linear regression equation was established for all types of coal in the deposit and was used to estimate the density of each reserve block.

$$\text{Specific gravity of coal} = 1.1704 + (0.009577 \times \%\ \text{ash})$$

The specific gravities used in this study for waste materials are:

- waste above bedrock 2.2
- bedrock waste (including major partings between coal zones 2.0

332 SWELL FACTORS

The swell factors used in this study for the three primary materials are as follows:

	As Mined	Dumped in Stockpiles	Machine Compacted
Coal	35%	35%	20%
Waste above bedrock			
- Granular surficials	20%	15%	0%
- Cohesive surficials	30%	25%	-
Bedrock waste	30%	25%	-

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

333 MATERIAL CUTTING RESISTANCE

Uni-axial compression tests and tri-axial shear tests were carried out to determine the cutting resistance of the various surficial and bedrock materials. The averaged test results for uni-axial strengths are provided on Tables 3-1 and 3-2. Tri-axial shear test results were not utilized since the breakdown of materials classification did not correspond to those adopted within the mining study.

334 THE COMBINED EFFECT OF MOISTURE AND CLAY-LIKE MATERIALS

The relative in situ clay-like mineral and moisture contents of in-pit materials exert a major influence on the behaviour characteristics of mined materials and subsequent equipment productivity during operations. The moisture contents by type of material together with the resulting derived values for Atterburg limits (plastic and liquid) are provided on Tables 3-1 and 3-2. The relationship between moisture content and Atterburg limits is of particular significance to trafficability characteristics which are discussed in the following section.

335 BEARING CAPACITY OF MATERIALS

Investigations were conducted to examine the suitability of in situ pit materials to support both fixed mine facilities and mobile equipment traffic.

At the locations recommended for the mine conveyor and its truck unloading stations, the crushing house, and ancillary facilities, the in situ strengths of both the surficial materials and the bedrock, could be expected to exceed the minimum foundation support specification of 5 kg/cm^2. Further soil strength tests are required to provide more precise, localized data for final foundation design of these structures. It was determined that locating semi-permanent conveyorways close to the northwest slide would not be practical due to the high risk of "soil creep".

An evaluation was made of roads founded on surficials, bedrock, and coal surfaces with regards to their stability to support large mobile equipment working at high productivity rates. Roads on granular surficial materials were considered to require minimal preparation, construction activities consisting of filling excavations or other hollows with adjacent materials to attain a uniform gradient, and providing for drainage. Normal road topping would be applied to the graded surface. Special road-building technology is only considered necessary in the northwest slide area.

Roads on waste rock and in situ coal are considered capable of supporting the traffic of 136-tonne trucks provided an adequate sub-base is constructed. As the effective moisture in most of the bedrock materials is below the derived values for plastic limits, geotechnical conclusions indicate that heavy traffic is likely to compact rather than liquify the materials.

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

3.4 HYDROLOGY

341 GENERAL

The surface and groundwater hydrology of the mine area will be modified for reasons of safety, continuity of operations, and material stability. The modifications will consist of three primary activities: diversion of surface water away from the various mine operating areas, removal and treatment of surface water within the mine operating areas, and withdrawal of groundwater primarily to stabilize mine wall slopes. The first two activities are relatively straightforward and are dealt with extensively in Volume IV. Groundwater removal is more complex however, particularly when considered in relation to bank stability, slope gradients, and stripping ratios.

342 GROUNDWATER WITHDRAWAL

A number of practical drilling problems were encountered in establishing well holes and test holes during the 1977 dewatering test program that had not been anticipated from earlier drilling programs. However, from an assessment of the limited results and previously available data, the following conclusions with regard to the withdrawal of groundwater have been drawn (Golder Associates, 1978):

(a) "It will be extremely difficult to achieve depressurization in most areas; in the claystone/ siltstone sequences both above and below the coal it could alternatively prove to be impossible or uneconomic to attempt depressurization. The pumping test of 1976-77 may only be representative of a local or anomalous situation."

(b) "Some areas within the coal may drain very poorly, and wet operating conditions and residual pore pressures could result. Further testing will assist in proving the extent of this problem."

(c) "Even if the permeability of the underlying conglomerate or the overlying surficials permitted these formations to be depressurized, the permeability of the adjacent claystone/ siltstone formations is too low for any significant assistance in depressurization to be made."

(d) "Consolidation coefficients have been calculated from the pumping test results and from laboratory tests. It appears that the rebound from stress relief on unloading by excavation could produce greatly reduced pore pressures - even negative pore pressures - within the claystone/siltstone formations. The low permeabilities of these rocks could mean that the pore pressures would be very slow to equalize. In fact equalization would only occur some considerable time after the end of mining."

(e) "Due to the marked variations in lithology, structure, topography and mineralogy that are known to exist at Hat Creek, groundwater conditions could vary significantly across the site."

In summary, extensive groundwater depressurization of mine slopes is not likely to be feasible.* Pumping from wells would be effective only in localized areas of better-than-normal permeability, and rebound resulting from stress relief should be relied upon for effective depressurization of the siltstone/ claystone rocks. The possibility of slope failure in the developing pit is somewhat greater than would be the case with successful depressurization. This risk has been accommodated in the design of slopes.

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* Memo from Golder Associates to CMJV dated 27 March 1978.

3.5 SLOPE STABILITY

351 GENERAL

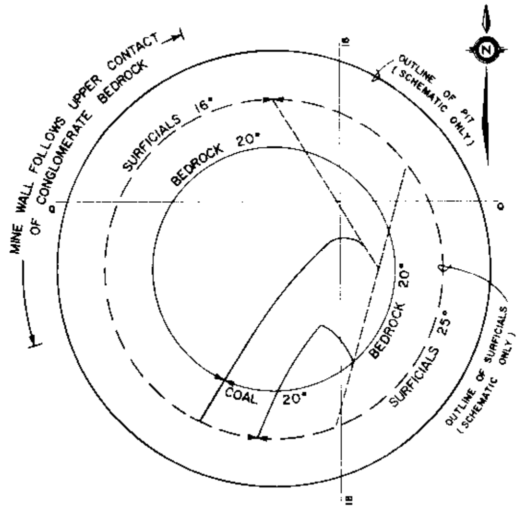
Determination of acceptable pit slopes for mine stability was one of the most complex geotechnical factors considered in the feasibility study. Although some field experience had been gained in handling weak materials and observing excavated mine wall slopes within the perched water table during the summer of 1977, none of the test excavations reached the deep-seated groundwater aquifers where most problems are anticipated. When considering the clay content of the majority of materials, the complex goundwater conditions, and the introduction of depressurization difficulties, the complexity in establishing slope stability criteria is self-evident.

Four main categories of slopes were investigated - final pit walls, dynamic slopes, waste dumps, and total resource pit slopes - and are dealt with below. Recommended slopes for each of these categories are shown graphically on Figure 3-1 which follows this page and on Figure 3-2, page 3-14.

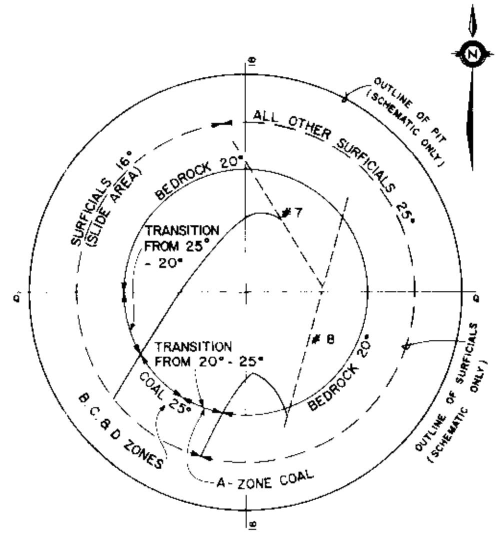
352 FINAL MINE WALL SLOPES

Based on the concept of rebound from stress relief and assuming stability monitoring, the following pit slopes were found geotechnically acceptable and have been adopted in mine planning.

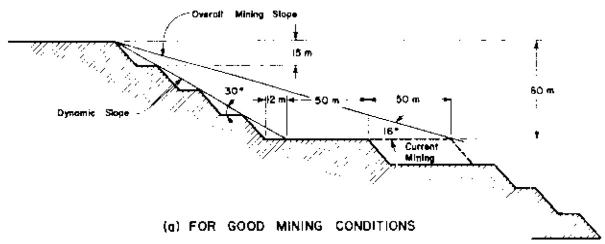
- bedrock in the pit, clockwise from Section Q on the west around to A-Zone coal in the southwest .... 20°
- B, C, and D-Zone coal in the south west .... 25°
- on the west side of the D-Zone coal, between Section Q and the D-Zone, a transition from .... 25-20°
- in the A-Zone coal on the west side of B-Zone coal, a transition from .... 25-20°



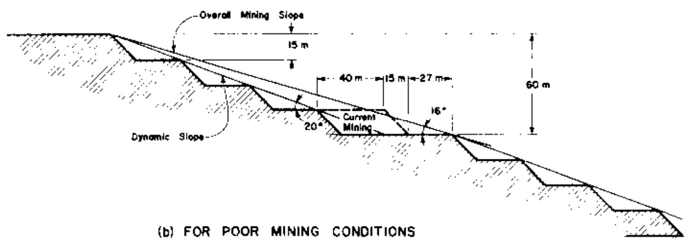
RECOMMENDED EXPANDED PIT
TO 450m ASL SLOPE ANGLES



35 YEAR PIT SLOPE ANGLES



(a) FOR GOOD MINING CONDITIONS



(b) FOR POOR MINING CONDITIONS

DYNAMIC SLOPE X SECTION

FIGURE 3-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

PIT SLOPES

- surficials, except in active slide areas .... 25°

- surficials in active slide areas .... 16°

These slopes are somewhat less conservative than those recommended by the geotechnical consultant earlier in the study and a certain element of risk has been introduced. This is particularly true for the slide area, which has been defined as a high maintenance area. Due to limited available data, a further breakdown of slope variations cannot be made at this time; further geotechnical assessment is required, particularly in the early stages of excavation, which will permit some refinement of these slopes.

353 DYNAMIC SLOPES

As the mine develops, combinations of benches would be advanced during excavation at steeper overall angles than either the final or the interim pit slopes. These are referred to as the dynamic slopes. These slopes would be temporary and therefore a lower factor of safety against failure would be tolerated. Failures in coal would generally follow joint patterns or bedding planes whereas in the bedrock waste and surficial materials, failures would tend to be circular because the material strength is not high in comparison to the strength along discontinuities.

The dynamic slopes recommended are dependent on the structural features of the rock being excavated. As these features vary substantially within the pit, specific recommendations to cover each operating condition encountered cannot be made at this time. However, the following guidelines were used in the planning:

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

(b) Dynamic slope angles of 30° are considered to be acceptable in strong bedded material where the bench alignment follows the above guideline.

(c) In the event the dip of the bedding is less than 30° and out of the face, and the strike of the bedding is parallel to or within 20° of the face alignment, the dymanic slopes should be reduced to the slope of the bedding. This precaution is not necessary for bedding less than 20°.

(d) Dynamic slopes of 30° in weak bedrock waste materials are considered to have adequate short-term stability up to a period of one or two years.

354 WASTE DUMPS

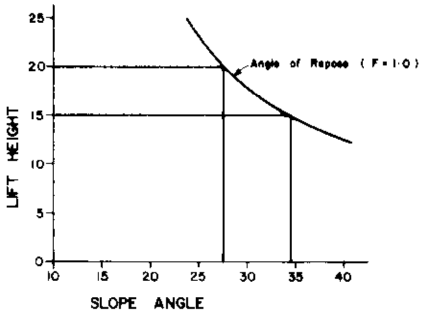
As a large portion of the waste to be disposed of is made up of very weak materials, construction of retaining embankments was recommended at the Houth Meadows and Medicine Creek dumping areas.

The geotechnical criteria used in assessing stability factors of the waste dumps are discussed below with regards to embankment construction materials, general waste dumping, and bench height. Waste dump slopes and other geotechnical design criteria are shown on Figure 3-2.

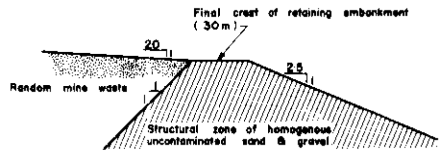
354.1 Embankment Construction Materials

The retaining embankments should be constructed of free-draining, homogenous sands and gravels: it is imperative that the glacial sands and gravels used for this purpose be uncontaminated by bentonitic clay mine wastes. Some size gradation of the materials would be required, control being provided by routine testing. Provided that contamination is prevented, adequate strength should be maintained without any special compaction measures.

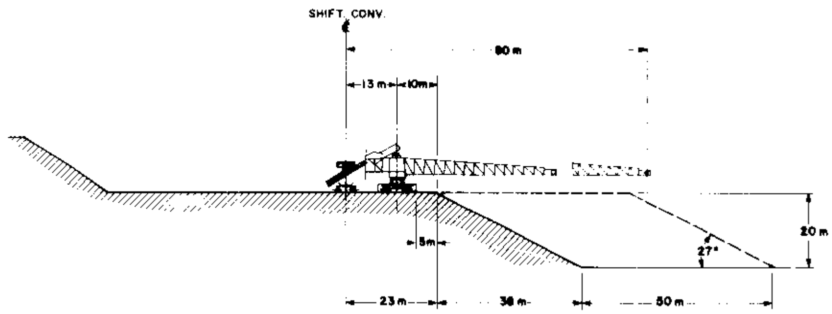
In constructing an embankment, the sands and gravels could either be dumped in lifts from a spreader, or placed in thin



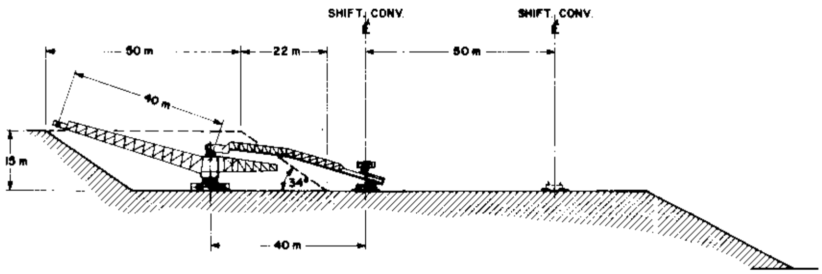
(a) WASTE MATERIAL ANGLE OF REPOSE



WASTE EMBANKMENT SLOPE ANGLES



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



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

FIGURE 3-2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

MINING FEASIBILITY REPORT

WASTE DUMP SLOPES

layers using trucks and bulldozers. By placing the materials in thin layers, increased quantities of poorer-grade silts and fine sands could be tolerated in reaching the given density.

354.2 Disposal of General Mine Wastes

To prevent weak waste materials which are dumped at heights exceeding the crest of the retaining embankment from sloughing over the embankment, the following gradient criteria were used:

- where dumped materials exceed the retaining embankment height by less than 80 metres, slopes should be not greater than 10 horizontal to 1 vertical
- where dumped materials exceed crest height by more than 80 metres, slopes should be not greater than 20 horizontal to 1 vertical.

354.3 Bench Height and Spreader Design

The predicted behaviour of dumped materials is a major factor in the design of the spreading equipment. The relationships between bench height and angle of repose for dumped Coldwater rock wastes have been estimated and are illustrated on Figure 3-2. The curve reflects a safety factor of 1.0, and represents the angle of repose of the material as it comes off the spreader.

No significant increase in overall stability would result from the mixing of any stronger materials from the mine, such as sand and gravel surplus to construction requirements or volcanics, with weaker materials excavated from the Coldwater claystone/siltstone/sandstone sequence. Similarly, no significant benefit would derive from using these "Coldwater-free" materials to stabilize dump benches while spreading is in progress: the stronger materials would have to blanket the dump to a thickness at least equal to the proposed bench height to provide any substantial support and foundation shear resistance.

355 EXPANDED PIT TO 450 M ASL

Based on the assumptions that an expanded pit to 450 m ASL will be backfilled with waste materials from the No. 2 Deposit and that either the pit slopes will be depressurized or that pore pressure reductions are possible from rebound of materials, the following pit slopes* (shown graphically on Figure 3-1) were adopted for planning purposes.

- bedrock coal and waste for the total circumference of the pit except where the conglomerate formation on the northwest side is encountered .... 20°
- in the northwest quadrant, the 20° bedrock slope angle would intersect the contact between the conglomerate and the lower siltstone/claystone sequence. The mine slope would be brought up to the contact with the conglomerate.
- east side surficials in till, sand, and gravels .... 25°
- east side volcanic deposits .... 25°
- west and south side surficials .... 16°

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* Recommended by Golder Associates.

SECTION 4

FUEL REQUIREMENTS AND QUALITY FACTORS

4.1 GENERATING STATION REQUIREMENTS

At the commencement of the study, coal requirements for the generating station were based on a specification that reflected the concept of beneficiating run-of-mine coal to an average calorific value of 7875 Btu/lb dry basis. At this level, the quantity of washed coal required was 282 million tonnes.

Following an economic assessment, the wash plant was eliminated from further consideration and the concept adopted whereby run-of-mine coal is blended and delivered directly to the generating station. Revised target qualities of 7375 Btu/lb dry basis and 24% moisture were established. Using these revised target qualities, thermal plant consultants set out the current schedule of plant needs as shown in column 5 of Table 4-1.

It was recognized that despite planning efforts to achieve a constant average grade of coal, periodic variations would result. Graphs were prepared showing yearly total Btu input requirements based on the target quality, together with the adjustments to be applied when variations occur. Column 6 of Table 4-1 reflects the quantity of R.O.M. coal, 349.49 million tonnes, required by the generating station after the variations for coal quality have been considered. The estimated "as delivered" moisture content was also revised upward from the target of 24% to 25%. Using the graphs shown on Figure 4-1 and applying the following formula, the estimated actual tonnes of coal required by the generating station may be calculated:

$$\text{Required Tonnes Coal} = \frac{\text{Yearly Btu Input Requirement} \times \text{Btu Input Multiplier Factor}}{\underset{\text{(dry basis)}}{\text{Average Coal Quality}} \times 0.75 \times 2205}$$

The average coal quality after blending for 35 years, although slightly below target, is listed below. It should be realized, however, that the total heat requirements of the generating station have been maintained by an increase in the overall coal tonnage.

ash content dry basis	36.3%
calorific value dry basis	7327 Btu/lb
moisture as delivered	25%
calorific value as delivered	5495 Btu/lb
ash content as delivered	27.2%

TABLE 4-1

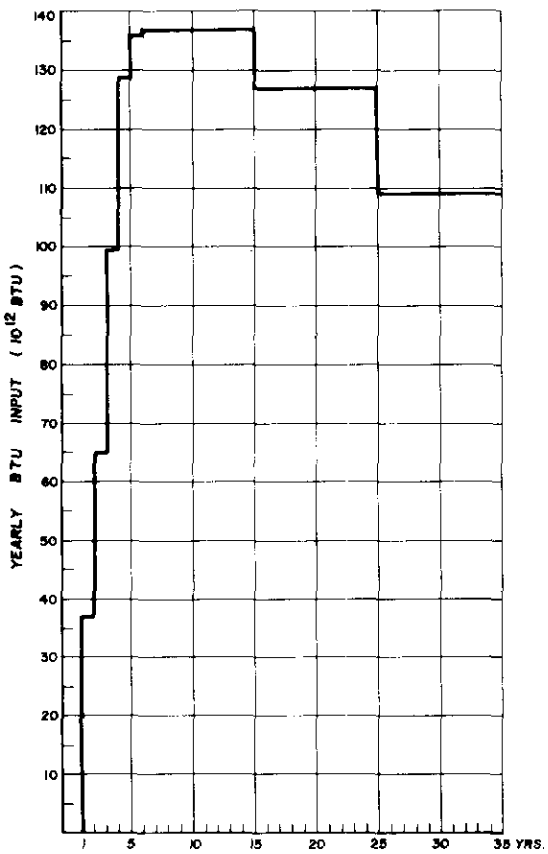
Requirements for Fuel and Heat Value
R.O.M. Coal at Thermal Plant

Hat Creek Mining Project Feasibility Report 1978

Year	Boiler Units in Service	Net Plant Capacity in MW	Average Capacity Factor %	Plant Needs at Target Quality tonnes x 10^6	Plant Needs* at Predicted Quality tonnes x 10^6	Total Heat* Required at Predicted Quality Btu's x 10^{12}
1	2	3	4	5	6	7
Pre-Production	1	500	-	1.0	1.03	12.5
1	1	500	69	3.0	3.08	37.0
2	2	1000	60	5.3	5.43	65.5
3	3	1500	60	8.0	8.20	99.0
4	4	2000	61	10.4	10.66	128.5
5	4	2000	65	11.0	11.30	137.0
6-15	4	2000	70	111.0	113.76	138.0
16-25	4	2000	65	103.0	105.96	128.0
26-35	4	2000	55	88.0	90.07	109.0
Totals:				340.7	349.49	4234.2 x 10^{12}

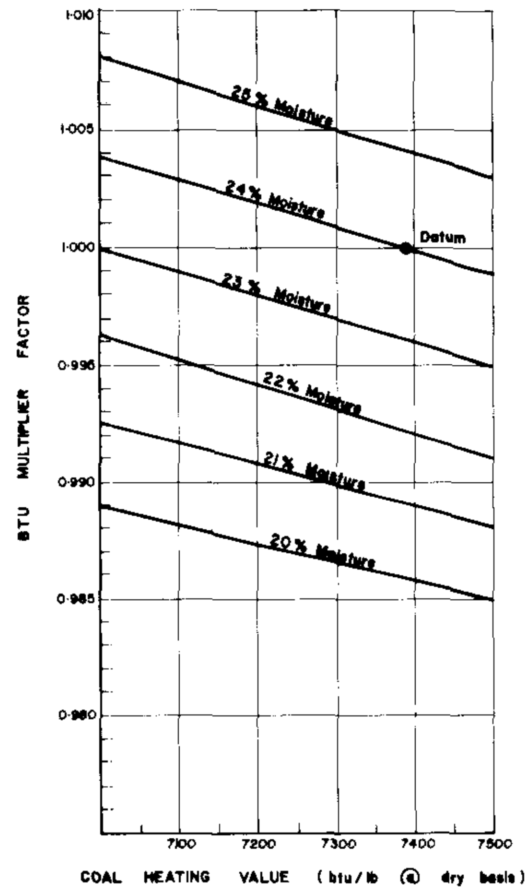
* Requirements are based on a specific R.O.M. quality for each period of pit development.

cominco-monenco joint venture



(a) YEARLY BTU INPUT REQUIREMENT

Note: Heat input is based on a datum of 7375 btu/lb. (dry basis) @ 24% moisture.



(b) BTU INPUT MULTIPLIER FACTOR

Note: used to modify yearly BTU input when coal quality deviates from datum.

FIGURE 4-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

BTU MULTIPLIER FACTOR VS IN SITU COAL & BOILER FEED QUANTITIES

411 ALLOWABLE COAL QUALITY VARIATIONS

Taking into consideration the ability of existing blending systems to accommodate variations in run-of-mine coal, the allowable variations in the quality of coal delivered to the generating station were mutually agreed upon and are summarized as follows:

(a) instantaneous fluctuations of $\pm$ 150 Btu/lb without notice are acceptable; and

(b) fluctuations of greater than 150 Btu/lb with a minimum of one hour's notice are also acceptable provided the quality of the blended coal is not less than 7000 Btu/lb or greater than 7800 Btu/lb dry basis.

4.2 PARTINGS REMOVAL INVESTIGATIONS

421 INTRODUCTION

Partings removal investigations were undertaken in recognition of the considerable quantity of partings and the importance of fuel quality to the generating station. In the investigations, geological cross-sections were utilized which showed partings interpretations developed independently of, but in parallel with, the Phase II and Phase III computer models. The principal objectives were to assess possible improvement in coal quality by selective partings removal, and to establish criteria for mine design in this regard.

The partings considered in this investigation are limited to waste materials lying within the A-Zone coal bed. Specifically excluded and dealt with in other sections of this report are those waste materials contained in the sub-zones A-2 and C-1, and dilution or waste materials developing from the foot wall and the hanging wall contacts with the coal seams. Also excluded from removal consideration are any partings in the B, C, and D Zones; these had been estimated in a preliminary study to have an insignificant effect on potential fuel upgrading, as compared to removal of partings from the A-Zone.

The study of partings removal was carried out in two stages. The first stage used pre-1977 drilling data, comprising only 19 holes penetrating the A-Zone, and developed order of magnitude estimates of quantities, improvement in coal quality, and economics of achieving this benefit; the results were treated as inconclusive and are not specifically outlined here. The second stage was based on all the latest available drill hole data, including the 1977 drilling program with 31 holes penetrating the A-Zone. The results of these studies are discussed in the following sections.

422 PARTINGS QUANTITIES

Partings quantities, estimated by a manual calculation and forecast for the 35 year period of mining, are provided on Table 4-2. Partings quantities, by the thickness and attitude (or angle), are also given in Table 4-2. It should be

TABLE 4-2

Summary of Partings in A-Zone

Hat Creek Project Mining Feasibility Report 1978

Time Period	Total Quantity ($m^3 \times 10^3$)		Summary of Quantity by Attitude ($m^3 \times 10^3$)				Summary of Quantity by Thickness ($m^3 \times 10^3$)			
			0-20°	20-90°	90-180°	Total	<3 m	3-5 m	>5	Total
Pre-production	-	38	-	26	12	38	38	-	-	38
Years 1 - 5	(4%)	429	23	114	292	429	274	155	-	429
Years 6 - 10	(17%)	1931	240	514	1177	1931	979	670	282	1931
Years 11 - 15	(19%)	2204	193	520	1491	2204	1343	543	318	2204
Years 16 - 25	(33%)	3747	790	782	2175	3747	2034	983	730	3747
Years 26 - 35	(27%)	3004	1345	1085	574	3004	1869	439	696	3004
TOTAL	100%	11 354	2591	3041	5722	11 354	6538	2790	2026	11 354
% Distribution			23	27	50	100	58	25	17	100

noted that 50% of the partings are estimated to be less than 3 metres thick and that 58% lie in the most adverse mining attitude: 90° to 180°.

423 POTENTIAL COAL QUALITY IMPROVEMENT

It was considered impractical to remove 100% of these partings materials because of the following factors:

- irregular occurrence and correlation of partings even as encountered in close-spaced development drilling
- recognizing and differentiating wastes from coal
- mining thin versus thick seams
- mining adversely sloping versus relatively flat seams.

The sensitivity of coal quality to partings removal was assessed using three alternative sets of assumptions: impractical (100% removal), conservative, and optimistic. These are summarized as follows:

	CASE A Impractical	CASE B Conservative	CASE C Optimistic
% of partings mineable if seam thickness is:			
> 5 m	100%	100%	100%
3 to 5 m	100%	85%	90%
< 3 m	100%	50%	65%
% of partings mineable if waste material in relation to direction of mining is:			
0 to 20°	100%	80%	95%
20 to 90°	100%	60%	75%
90 to 180°	100%	20%	50%

The percentages of partings considered mineable were applied to the originally calculated quantities for five-year periods and quality improvements calculated using the following formula.

$$Q = \frac{M \times D}{M - (P \times 2.00)}$$

where:

Q = quality of coal after partings removal
M = original quantity of coal (tonnes)
D = quality of run-of-mine coal (dry basis) before partings removal
P = quantity of partings (m^3)

The results of calculations using this formula are somewhat optimistic in that the formula assumes 100% ash content and zero calorific value in the partings.

Benefits to coal quality that could accrue from partings removal in the A-Zone coal during the first 10 years of operations are estimated to be in the range of 2 to 10%. The coal quality improvement over the 35-year period of mining is estimated to be in the range of 8 to 14% for the A-Zone coal (refer Table 4-3). It should however be recognized that the A-Zone coal represents only about 22.5% of the total coal reserves and hence the maximum coal quality improvement over the total coal reserves are estimated at approximately 0.5 to 2.3%.

424 METHODS OF PARTINGS REMOVAL

Partings greater than 5 metres in thickness are generally considered mineable with 16.8 m^3 shovels and 136 tonne end-dump trucks. Partings less than 5 metres in thickness, however, will require smaller units of equipment with greater flexibility. The mining techniques and equipment have been broadly grouped as follows:

(a) for seams lying between 0°-20° from the horizontal - generally small scrapers and bulldozers provided the seams are fairly flat-lying

TABLE 4-3

Summary of Estimated Partings Removal
Investigation Results

Hat Creek Project Mining Feasibility Report 1978

Parameter	Case A		Case B		Case C	
Rating	impractical		conservative		optimistic	
% of Total Partings Considered Removable	100 %		29 %		45 %	
QUANTITIES TO BE REMOVED ($m^3 \times 10^3$)						
Year						
Pre-production		38		8		17
0 - 5		429		73		129
6 - 10		1931		425		464
11 - 15		2204		510		1060
16 - 25		3747		1050		1910
26 - 35		3004		1227		1830
TOTAL		11 354		3293		5110
ESTIMATED IMPROVEMENT IN BTU VALUE IN A-ZONE COAL						
Year						
Pre-production to Year 5	(10%)	547	(2%)	88	(3%)	159
6 - 10	(60%)	3240	(9%)	486	(10%)	535
11 - 15	(45%)	2485	(8%)	427	(17%)	969
16 - 25	(33%)	1808	(6%)	309	(14%)	792
26 - 35	(80%)	4434	(22%)	1227	(41%)	2255
TOTAL	(40%)	2169	(8%)	461	(14%)	775

(b)	for seams lying between 20°-90° from the horizontal	-	bulldozers, small front-end loaders, with small end-dump trucks combined with occasional Gradall or backhoe
(c)	for seams lying between 90°-180° from the horizontal	-	probably backhoe and Gradall excavators emptying into small end-dump trucks

Because of the nature of the operation and geological conditions, final equipment selection to accommodate partings removal should not be made until experience has been gained during actual trial mining operations.

425 CONCLUSIONS

(a) For the purposes of this study the benefits of partings removal in the A-Zone are considered "potential" only and would have a minor effect on the overall coal quality. Detailed cost estimates of mining partings or of the coal quality improvements that might result were not considered to be warranted at this level of study.

(b) During the early years of mine development a closely monitored test program to determine the economics and best methods of removing partings should be undertaken. Major capital expenditures should be contemplated only on completion of the testing program. The types of equipment required are all readily available on short delivery notice at the present time.

(c) The possible mining of partings does not have a critical or even significant impact on the decision to proceed with the project. The majority of partings that could be effectively removed will only be encountered in the later years of mine development.

4.3 CUT-OFF GRADES

431 INTRODUCTION

In determining a policy of cut-off grades for utilization of Hat Creek coal, consideration was given to several factors including the unusual geological structure, massive shaling-out of certain seams on the west limb, and conservation of potential resources for future use. The largest constituent of the deposit and that containing the best-quality coal is D-Zone which lies at the bottom of the basin. Above this are other, separate layers of coal, C-Zone, B-Zone, and A-Zone, each of unique character and containing varying amounts of coal. In order to provide sufficient quantities of fuel to the required specifications of the generating station all of these seams and their sub-seams must be mined.

A two-stage cut-off grade has been implemented, resulting in three categories of mine production. The higher cut-off differentiates fuel from non-fuel; the lower cut-off further subdivides non-fuel according to its ultimate disposal in either waste dumps or low-grade coal stockpiles. To estimate the relative quantities of materials included in each category, a computer-generated summary of coal reserve blocks was prepared. Reserve block data were analyzed and cut-off grades for fuel, non-fuel, and waste estimated as discussed in Section 432 and 433 following. The distribution of the low-grade coal material below fuel cut-off and above waste cut-off is discussed in Section 434. Sensitivity studies on the effects of varying fuel cut-off grades were also carried out and are outlined in Section 435.

It should be noted that while the cut-off grades used in this study are appropriate for this level of mine design, it is not intended that they be adopted as fixed operational constraints. Close-spaced bench sampling is required to more completely delineate the quality distribution throughout each sub-zone for the day-to-day selection of fuel grade coal.

432 NON-FUEL CUT-OFF GRADE

For this study coal blocks are computer-generated vertical columns 25 x 25 metres in plan, based on a 14 sub-zone model. Each column has been assigned a tonnage and a quality interpolated from drill hole data. The sum of all blocks in the 35-year pit, by category of grade quality, is listed in Table 4-4. Ninety-five percent of the resource heat is contained in 82% of the coal reserve blocks whose average value exceeds 4000 Btu/lb. The remaining 5% of the resource heat is found in the 18% of the coal reserve blocks that are classed as non-fuel coal.

Table 4-4 and Figure 4-2 show that variations in both coal quality and quantity are relatively small in the groups of blocks ranging in value from 3000 to 4500 Btu/lb.

The distribution of non-fuel coals was assessed by examining the computer displays of heat values on plan and section. It is apparent that most of the non-fuel blocks (less than 4000 Btu) are located in sub-zones A-2 and C-1 which can be described as waste bands with minor coal inclusions. The significant portion of the remaining low value blocks is located on the western flank of the syncline and probably represents the result of shaling-out during coal deposition and slide-like incursions of Coldwater rocks into the younger coal measures. The actual breakdown by zone follows:

Zone	Coal less than 4000 Btu/lb
A-1	0%
A-2	23%
B-1	4%
C-1	34%
C-2	39%
D-1	0%

TABLE 4-4

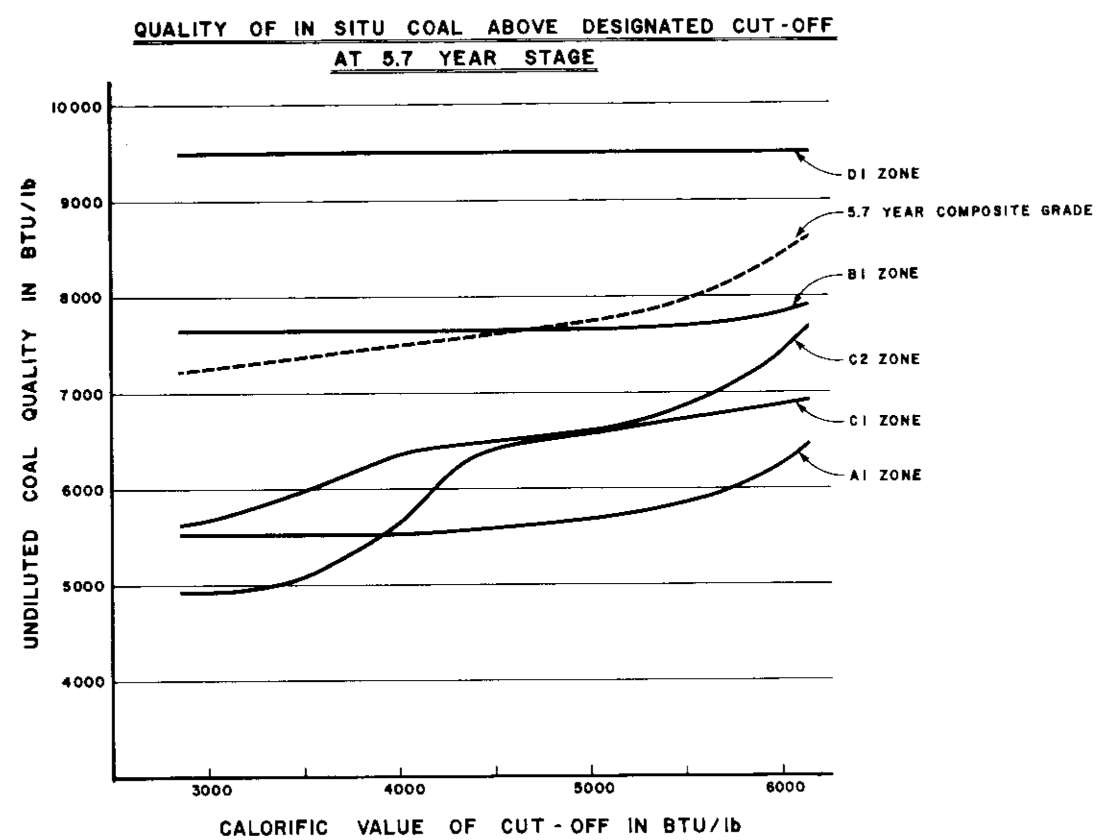
Estimates of Undiluted Coal Block Tonnages
by Range of Heat Value

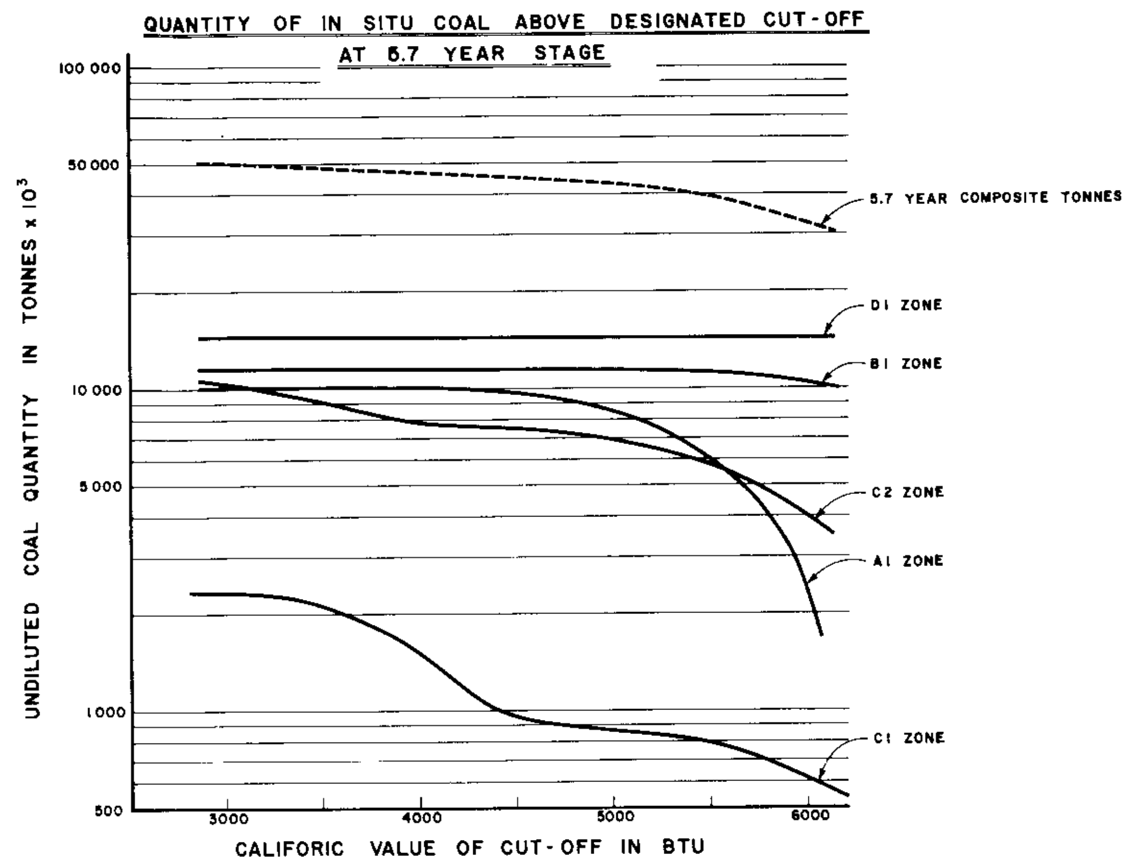
Hat Creek Project Mining Feasibility Report 1978

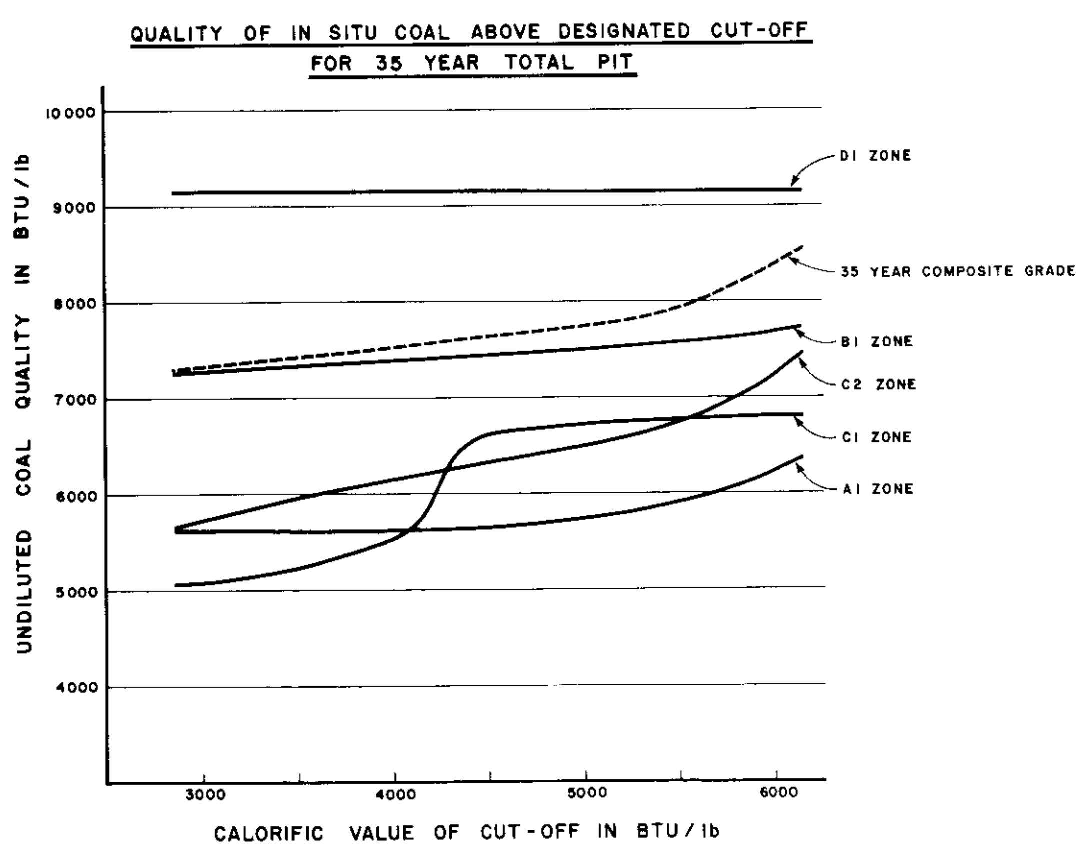
Quality Range (Btu/lb)	Coal Blocks (tonnes)	Average Heating Value (Btu/lb)*	Quality of Remaining Coal (Btu/lb)	Total Heating Value (Btu x 10^{13})**	PERCENTAGE Block Tonnages by Range	PERCENTAGE Block Tonnages Cumulative Remaining	OF TOTAL Heating Value by Range	OF TOTAL Heating Value Cumulative Remaining
0-2000	40 991	1000	6505	6.78	9.8	100.0	1.5	100.0
2000-3000	18 258	2449	7102	7.40	4.3	90.2	1.6	98.5
3000-3500	8143	3255	7344	4.38	1.9	85.9	1.0	96.9
3500-4000	7594	3745	7434	4.70	1.8	84.0	1.0	95.9
4000-4500	12 210	4265	7515	8.61	2.9	82.2	1.9	94.9
4500-5000	13 641	4784	7635	10.79	3.3	79.3	2.4	93.0
5000-6000	70 549	5522	7757	64.41	16.8	76.0	14.3	90.6
+6000	247 450	8394	8394	343.44	59.2	59.2	76.3	76.3
Total :	418 836	6505		450.51	100		100	

* dry basis
** assuming 25% moisture as delivered

Statistics reporting remaining coal or heat relate to lower value of range.







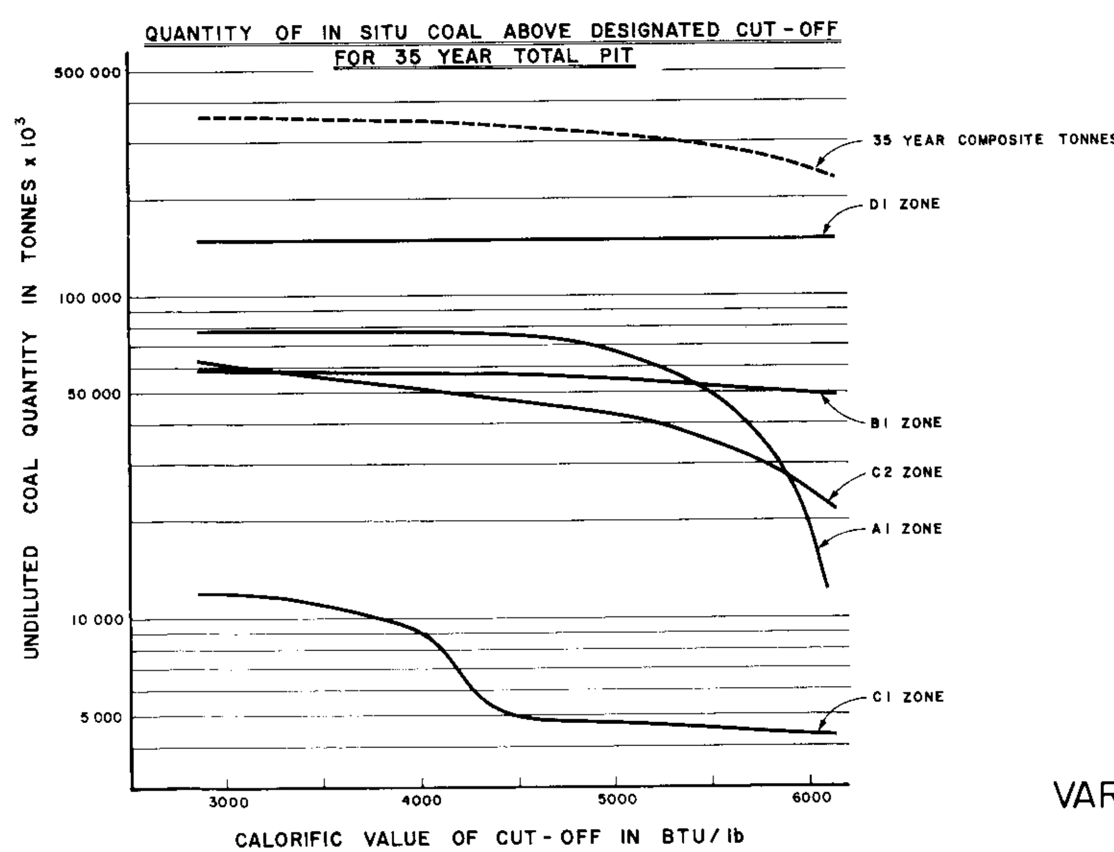


FIGURE 4-2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

VARIATIONS IN CUT-OFF GRADE
VS
COAL QUALITY & QUANTITY

As a result of the sensitivity studies, 4000 Btu/lb was considered to be a realistic bench mark grade for differentiating fuel from non-fuel in mine production. The average diluted fuel quality of 7327 Btu/lb, calculated from all blocks above 4000 Btu/lb, was acceptable for current boiler design. A large proportion of the deposit heat resources are consumed in the boiler and only minor amounts of heat are lost to the waste dumps or tied up in low-grade stockpiles. The 4000 Btu/lb cut-off value has therefore been used in all scheduling of fuel quantities, qualities coming from the mine, and in the preparation of the project fuel specification.

433 SEAM WASTE CUT-OFF GRADE

Seam waste is defined as the coal/waste mixture mined within the coal seam boundaries that has insufficient coal content to merit retaining it in low-grade coal stockpiles. It is economically advantageous over the 35-year mining period to set the cut-off value for seam waste relatively high, as it is costly to retain each increment of tonnage in the low-grade stockpile. However, more energy potential is conserved at a lower cut-off value. A reasonable compromise between costs and energy conservation is judged to exist at about 3000 Btu/lb level, in which the mix would probably be 20% true coal and 80% waste. In addition, this was considered to be a practical, recognizable bench-mark for the actual grade control procedures in the mine. Therefore, 3000 Btu/lb was adopted as the cut-off between low-grade coal and waste.

On this basis, the estimated tonnage of waste produced from each zone at the various stages of mine development are shown in Table 4-5. The major points are as follows:

(a) 65% of the total seam waste material comes from the two major interseam waste zones, A-2 and C-1. Despite some localized coal inclusions, these zones are essentially broad waste bands.

(b) Most of the remaining seam waste (33%) is in the C-2 zone and is located in the shaled-out west limb of the deposit.

TABLE 4-5

Estimated
Production of Wastes and Low-Grade Coal
over Stages of Mine Development

Hat Creek Project Mining Feasibility Report 1978

Period	Tonnage (x 10^3) by Zone						All Zones
	A-1	A-2	B-1	C-1	C-2	D-1	
SEAM WASTES (calorific value < 3000 Btu/lb)							
Pre-production to Year 5	0	2549	0	1700	685	0	4934
Years 5 - 9	0	1282	0	1054	696	0	3032
Years 9 - 15	0	2669	127	5980	3691	0	12 467
Years 15 - 20	0	1921	46	1187	277	0	3431
Years 20 - 25	0	2328	1066	5802	7458	0	16 654
Years 25 - 35	0	4701	184	7163	6873	0	18 921
Total Waste	0	15 549	1421	22 844	19 536	0	59 350
LOW-GRADE COAL (calorific value between 3000 to 4000 Btu/lb)							
Pre-production to Year 5	0	33	0	813	2560	0	3406
Years 5 - 9	0	231	20	370	473	0	1094
Years 9 - 15	0	220	379	566	2994	0	4159
Years 15 - 20	0	536	100	70	192	0	964
Years 20 - 25	0	160	435	118	1179	0	1892
Years 25 - 35	0	331	402	886	2684	0	4303
Total Low-Grade	0	1512	1342	2823	10 060	0	15 737

(c) The minor contribution from B-1 zone is from the west limb.

(d) No contribution from D-1 or A-1 zones is evident.

(e) Two-thirds of the seam waste production occurs after Year 15. By that time a practical approach towards cut-off grades will have been developed.

(f) The average calorific value of seam waste is less than 1500 Btu/lb (dry basis).

434 LOW-GRADE COAL

Low-grade coal is defined as that portion of mine production with a heating value between 3000 to 4000 Btu/lb (dry basis).

The tonnage distribution of low-grade coal by zone is listed in Table 4-5. The major points are as follows:

(a) 64% of the low-grade coal is estimated to come from the C-2 zone on the western fringe of the deposit.

(b) The combined major waste zones, A-2 and C-1, represent 27%. Low-grade coal will be found in random areas of these zones, but concentrated in the easterly portions.

(c) Approximately 30% of the total low-grade coal is estimated to be produced within the first 10 years of operation.

(d) The average grade of the low-grade stockpile is in the order of 3500 Btu/lb. If the stockpile is not utilized prior to the completion of the project, it will have accumulated only 2% of the heat value contained in that portion of the resource mined.

435 IMPLICATIONS OF VARYING FUEL CUT-OFF GRADE

Selection of the optimum cut-off grade for run-of-mine coal must attempt to satisfy two opposing viewpoints:

(a) mine operations would, ideally, favour the establishment of a cut-off grade which matched the run-of-mine coal excavated at least cost to the mine, thereby reducing waste volumes;

(b) generating station operations would, ideally, favour the establishment of a cut-off grade which matched manufacturer design requirements and kept operating costs to the minimum.

Maximum economic utilization of the resource lies in a compromise of these opposing factors. In developing the recommended Btu/lb cut-off grade, sensitivity studies were carried out taking into consideration the implications for both the mine and the generating station of varying the cut-off grades, although in some cases the available data on boiler costs were insufficient.

Four aspects of cut-off grade considered were:

- the heat value utilized
- effects on boiler fuel demand
- effects on sulphur values in the run-of-mine coal
- economics

Heat Value Utilized The effects on the proportion of resource heat utilized in the boilers that result from varying the mine cut-off grade are estimated and shown graphically on Figure 4-3. The graph shows that:

(a) At all realistic cut-off grades the overall utilization for this deposit is expected to be in the order of 95%.

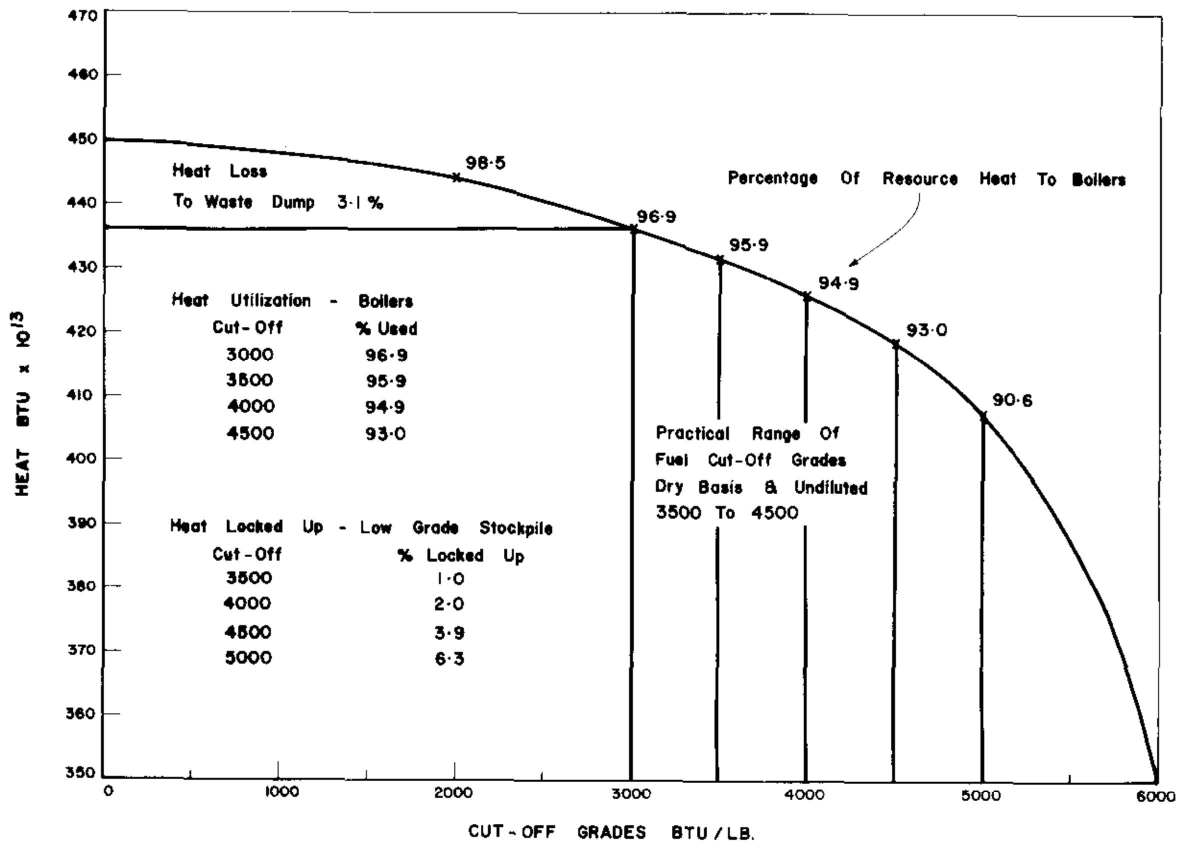


FIGURE 4-3

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

UTILIZATION OF THE HEAT RESOURCE AT VARYING CUT-OFF GRADES

(b) The incremental change in heat utilization over the range of cut-offs from 3000 to 4500 is about 1% per 500 Btu's of change in cut-off. However, as the cut-off grade increases beyond 4500 Btu/lb the heat utilization begins to decrease rapidly. A cut-off of 4000 Btu/lb results in high utilization and is close to the centre of the "flat" zone of minimal change.

Effects on Boiler Fuel Demand Table 4-6 shows the estimated effects of cut-off grade variations on boiler fuel demands. Increasing the cut-off grade from 4000 to 4500 Btu/lb requires the mining of an additional 6.6 million tonnes of coal and increases the quality of the run-of-mine coal from 7327 Btu/lb to 7444 Btu/lb dry basis. Quantity and quality variations for other cut-off grades show the variation in tonnes of coal to be mined versus the changes in quality. A cut-off value of approximately 4000 Btu/lb provides for the best combination of minimum quantity of coal to be mined and optimum heat value.

Effect on Sulphur Values The average level of sulphur in the fuel over the 35-year project is estimated to be in the order of 0.65 lbs of sulphur per million Btu's or 0.48% by weight of diluted fuel. No significant difference in the percent sulphur is evident when the fuel cut-off is varied 500 Btu/lb from the recommended cut-off. There is however, a small variation in the parts of sulphur per million Btu. The following results indicate how little sulphur is affected by varying the cut-off grade:

Fuel Cut-Off in Btu's/lb	Diluted Fuel % Sulphur	Coal Quality Heat Content	Lbs of Sulphur Per Million Btu's
3500	0.48%	7248	0.66
4000	0.48%	7327	0.65
4500	0.48%	7444	0.64

These variations relate more to combustion efficiencies than to fuel dilution or the addition of high or low sulphur coal blocks in the mining process. It was concluded that general variations in cut-off grades would not have a significant, direct effect on sulphur levels. Rather it will be necessary to isolate production areas of high sulphur content and handle them independently of calorific cut-off considerations.

TABLE 4-6

Estimated
Effects of Cut-Off Grade Variations
on Boiler Fuel Demand

Hat Creek Project Mining Feasibility Report 1978

Cut-off Grade (Btu/lb)	Coal Reserves - Undiluted: Tonnage ($t \times 10^6$)	Coal Reserves - Undiluted: Calorific Value (Btu/lb)	Deliverable Fuel*: Tonnage ($t \times 10^6$)	Deliverable Fuel*: Calorific Value (Btu/lb)	Estimated Boiler Demand ($t \times 10^6$)	Present 35 Year Pit Production: Shortfall ($t \times 10^6$)	Present 35 Year Pit Production: Excess ($t \times 10^6$)
6000	247.5	8394	251.3	8184	310.7	59.4	-
5000	318.0	7757	322.9	7563	337.2	14.3	-
4500	331.6	7635	336.7	7444	343.3	6.6	-
4000	343.8	7515	349.1	7327	349.1	- DATUM	-
3500	351.4	7434	356.8	7248	353.2	-	3.6
3000	359.6	7339	365.1	7156	357.9	-	7.2
2000	377.8	7102	383.7	6924	370.7		13.0
0	418.8	6505	425.2	6342	406.6	-	18.6

* reflects dilution and mining losses

Economics While it was possible to examine the economic effects on the mine operations of varying cut-off grades, the results were found to be misleading as they naturally favoured as low a cut-off grade as possible, which is not necessarily in the best interest of the total project. Further work, in cooperation with the generating station, is required to determine the optimum economic cut-off grade. A cut-off grade of 4000 Btu/lb would appear to represent the most favourable compromise of all economic factors that can be determined at the present time.

436 IMPLICATIONS OF VARYING WASTE CUT-OFF GRADE

The sensitivity of the 3000 Btu/lb waste cut-off level was also examined by evaluating the effects on quantity and quality of raising and lowering this value. These are summarized in Table 4-4. Raising the cut-off grade to 3500 Btu/lb may commit an additional 8.1 million tonnes of coal , representing about 1% of the resource heat, to the waste dumps. On the other hand, lowering the cut-off to 2500 Btu/lb would add about 9 million tonnes to the low-grade stockpile at considerable cost. This would also reduce the average quality of the low-grade coal from 3491 Btu/lb to 3200 Btu/lb. The low-grade stockpile must be viewed in context with other future energy resources of the Hat Creek deposit. After 35 years of operation an energy resource far larger and less expensive to recover than the low-grade stockpile would still remain in situ. In considering these factors and the following sensitivities analyses a 3000 Btu/lb cut-off for seam waste was selected.

437 IMPLICATIONS OF VARYING LOW-GRADE COAL CUT-OFF GRADE

Assuming the waste cut-off grade of 3000 Btu/lb is maintained, the effect of raising and lowering the cut-off grade on the low-grade coal stockpile is summarized as follows:

Coal Cut-Off Grade (Btu/lb)	Low-Grade Coal Stockpile Quantity of Coal (tonnes x 10^3)	Overall Grade (Btu/lb)	% of Total Heat Value
3000	Nil	N/A	0.0
3500	8143	3255	1.0
4000	15 737	3491	2.0
4500	27 947	3829	3.9

The additional cost of placing and maintaining low-grade coal in a stockpile is $0.90/tonne when compared to placing the material in waste dumps and $0.80/tonne when compared to blending it with other coal.

438 CONCLUSIONS

(a) The following grade definitions for mine production are recommended:

Waste	-	less than 3000 Btu/lb
Low-grade coal	-	between 3000 and 4000 Btu/lb
Fuel	-	greater than 4000 Btu/lb

(b) Selection of the optimum fuel cut-off grade for the overall project is dependent on the resolution of conflicting mine and generating station economic priorities. In this regard continuing studies are warranted.

(c) From purely a mining point of view it appears difficult to justify the costs of creating, maintaining, and eventually recovering coal from a low-grade coal stockpile. The additional cost is in the order of $0.80/tonne of low-grade coal or $0.11/million Btu of heat value. By the time this heat value is recovered from the low-grade stockpile, it is anticipated that the cost in 1977 dollars would approach double the normal cost of mining this same heat value from coal in the expanded pit.

If the low-grade coal is placed in the waste embankments with other material it can never be recovered. Conversely, if the low-grade coal is incorporated into the blended fuel over the life of the project, the average fuel quality would be reduced from the present 7327 Btu/lb to 7156 Btu/lb. With this lower average quality it is unlikely that coal delivered to the generating station would always be above the minimum of 7000 Btu considered acceptable.

The average quality of 3491 Btu/lb presently forecasted for the low-grade stockpile is sufficiently low as to warrant re-examining the merits of maintaining the low-grade coal stock-pile.

(d) The practicalities of cut-off grades lie in the ability to identify and sort waste/coal mixtures prior to or during excavation. Problems will be encountered predominantly in C-Zone coals on the west limb of the deposit, and to a lesser degree, in the A-2 and C-1 waste bands where they contain significant amounts of coal whose calorific value is greater than 4000 Btu/lb.

4.4 DILUTION AND LOSSES

441 INTRODUCTION

Studies were undertaken to determine the factors of dilution and mining losses. The final factors used are:

> Dilution at 2.5% by weight of diluted coal. The diluent is assumed to have zero heat value.
>
> Mining Loss at 1% of the diluted coal as mined.

Conversion of undiluted coal reserves to R.O.M. tonnes is accomplished by dividing by 0.975 to express the diluted state and by multiplying by 0.99 to express loss. Coal reserve grades are multiplied by 0.975 to express R.O.M. qualities. Conversions were applied manually to computerized statistics which describe withdrawal of fuel coal from in situ reserve blocks at any period in the mine development.

$$\text{R.O.M. Tonnes} = \frac{\text{Reserve Tonnes In Situ} \times 0.99}{0.975}$$

$$\text{R.O.M. Quality} = \text{Reserve Quality In Situ} \times 0.975$$

Within the context of this study, diluents are defined as unavoidable waste materials mined with fuel-grade coal. Specifically excluded are:

(a) partings within those sub-zones classified as coal; and

(b) the shaled-out coal, contained primarily in C-Zone, occurring in the west limb of the deposit which is classified as low-grade coal or waste.

442 SCOPE

The studies into dilution and mining loss were undertaken to estimate the effect of day-to-day operating practices on these losses. A practical balance must be maintained between the cost effect on the product resulting from dilution and loss, and the operational cost of minimizing these losses. The studies view the effects on dilution and loss by types of excavators as being equal. Some of the operating conditions that may be peculiar to Hat Creek include:

- problems in identifying the diluent, as it may resemble coal in colour, etc., while being dug, and because the line demarcating coal and waste is irregular due to drag folding
- problems in trying to prevent mixing of coal and diluent due to the low inherent strength in both of these materials
- problems in interrupting the scheduling of haulage vehicles.

443 DILUTION QUANTITIES

The study into dilution applied the concept that the percent weight of diluent that will be mixed with the coal is a function of the area of the contact surface of the coal/waste interface and the angular attitude of this interface. All contact areas were measured and classified as to attitude - either less than or greater than 15°. This process was repeated for several stages of the pit development to determine if there was a significant variation with time.

With regard to operating difficulties in effecting a clear separation of waste products from coal, the following dilution values were assumed:

Angle of Interface	Thickness of Waste Mixed into Coal
less than 15°	0.3 metres
greater than 15°	0.7 metres

The combination of waste thickness, interface area, and specific gravities of waste products in each stage of the mine development, as represented by a series of nested pits, yielded an estimated tonnage of diluent that could be compared to the tonnes of coal mined in the same stage. Additional tonnages of diluents judged to be representative of this particular operation are as follows:

- all sloughs of waste onto coal benches and not completely cleaned up .... 20 000 tonnes/year
- all road building materials laid down on coal benches and incompletely removed .... 10 000 tonnes/year

The results of diluent quantity calculations are provided below. Nominal time periods are not directly related to the generating station schedule.

Period of Pit Development	tonnes x 10^6 Coal	Diluent	% Dilution
-1 - 5 years	22.1	.6	2.6
6 - 10 years	44.8	1.2	2.6
11 - 15 years	62.2	1.5	2.4
16 - 25 years	115.4	2.5	2.1
26 - 35 years	98.3	2.2	2.2
Total	342.8	8.0	2.3

An additional 10% allowance was made to reflect human error. It was concluded that, in practice, the periodic variation in dilution proportions would not be great and that a simple life-of-the-operation factor could be applied without reservation; thus the factor of 2.5% was adopted.

444 QUANTITIES OF MINING LOSSES

A similar approach was taken towards the evaluation of potential mining losses of the coal reserves. The following day-to-day operating situations were considered to constitute mining losses:

- a token loss of oxidized coal
- coal lost with waste removed at coal/waste interfaces
- errors in dispatching coal to waste dumps
- degrading of coal during ground sloughs to such an extent that it would be dispatched to the waste dumps
- losses from dusting of fine coal and spillages during transportation

Certain of the above-listed losses have been estimated as follows:

Oxidized Coal

West side of pit - a 1 metre thick loss over the areas of A and D coal subcrops (B and C-Zone coals subcrop as non-fuels).

East side of pit - a 0.25 metre loss over all coal subcrops.

Coal at Interfaces with Waste

A 0.25 metre loss was assumed where coal interfaces with hanging wall rocks, foot wall rocks, A-2 and C-1 Zone waste bands.

Operational Losses

The combined loss from all other causes, i.e., human errors, wind losses, admixing with waste during sloughs, etc., was estimated at 25 000 tonnes per year.

The study results are shown in Table 4-7; it was decided that the average value of 1% loss would be applied as a life-of-the-project factor to all stages of the pit development.

TABLE 4-7

Potential Coal Losses
During Pit Development

Hat Creek Project Mining Feasibility Report 1978

Development Period	Coal Produced	Coal Loss ($t \times 10^6$)				% Loss
		Oxidized	Dilution	Other	Total	
Years 0 - 5	22.1	0.2	0.1	0.1	0.4	2.1
Years 5 - 10	44.8	0.2	0.3	0.1	0.6	1.3
Years 10 - 15	62.2	0.2	0.3	0.1	0.6	1.0
Years 15 - 25	115.4	0.3	0.5	0.3	1.1	1.0
Years 25 - 35	98.3	-	0.6	0.2	0.8	0.8
Total	342.8	0.9	1.8	0.8	3.5	1.0

4.5 R.O.M. COAL GRADE CONTROL INVESTIGATIONS

451 INTRODUCTION

To determine the anticipated run-of-mine grade variations and subsequent stockpile and blending requirements for the mine, three short-term equipment simulations for production grade control were carried out. These studies were based on the shovel/truck/conveyor mining system.

This section of the report discusses the simulation studies, their results, and recommends the mine operating practices considered necessary to achieve the grade control and thus supply satisfactory run-of-mine coal to the proposed blending facility. Security of coal supply over the short-term is also discussed.

The three studies are described as follows:

The first study attempts to demonstrate the quality variation in R.O.M. coal delivered to the blending facility over a one week period of time. This study simulates coal production from specially prepared mining bench plans that are based on the degree of variation in coal quality found in the basic bore hole data. The second study demonstrated the level of grade control that should be achieved over the short term. This investigated weekly and monthly coal quality variations over a period of one year by scheduling equipment to mine selected blocks of coal. The study utilized the Phase II mine model information projected on bench plans, and was carried out for Year 3 of the 35-year mining sequence. The third study was a repeat of the second study with the exception that the Phase III mine model data and Year 1 were used for the analysis.

452 MINE OPERATING PRACTICE

The bench configuration used in the mine plans for the 35-year development of the mine permitted concurrent shovel production from four to six benches; however, each operation of one shovel would be separated vertically by three to five benches, thus permitting the mine to develop on 10 to 20 benches over the long-term. By using this approach it should be possible to mine coal in 100-metre wide strips on each of the operating benches. Figure 3-1 in Section 3 of this report shows a profile of the bench layout discussed here and referred to as the dynamic slope.

To obtain the operating flexibility desired for selecting blocks of coal, the 100-metre wide strip can be mined as two 50-metre wide strips. The outer strip would be mined in a straightforward, unselective manner, with the shovel taking all grades of coal available in the mining face. The inner strip would be selectively mined by moving the shovel back from the face of the outer strip to a location along the length of the inner strip containing a desired grade of coal. Figure 4-4 depicts a typical mining operation on bench 850 as taken from the Year 5 mine plan. This mine operating practice has been incorporated in the mine plans and was also used in the second and third simulation studies carried out.

For the weekly and monthly studies, the mine model information available was used to schedule equipment to excavate the desired coal by moving shovels short distances. In practice, however, additional "fill-in" bore hole drilling and sampling would be carried out to provide a detailed breakdown of the coal available on each bench. This is further discussed in Section 453 following. In addition, it has been found that a six-month supply of coal is exposed ahead of the excavating equipment such that detailed grade control planning can be done and equipment scheduled according to a six-month or yearly mine plan.

453 PRODUCTION PLANNING

A number of operating procedures are important to the process of predicting short-term production grades that result from the simultaneous mining of coal blocks of various qualities. The simulation studies carried out were based on the broader information available in the mine model, without the benefit of information from fill-in drilling and grade control procedures that would normally be available in an operating mine. However, the available data were considered adequate for this feasibility study, and it is recommended that the following control procedures be incorporated into the mining plan.

cominco-monenco joint venture

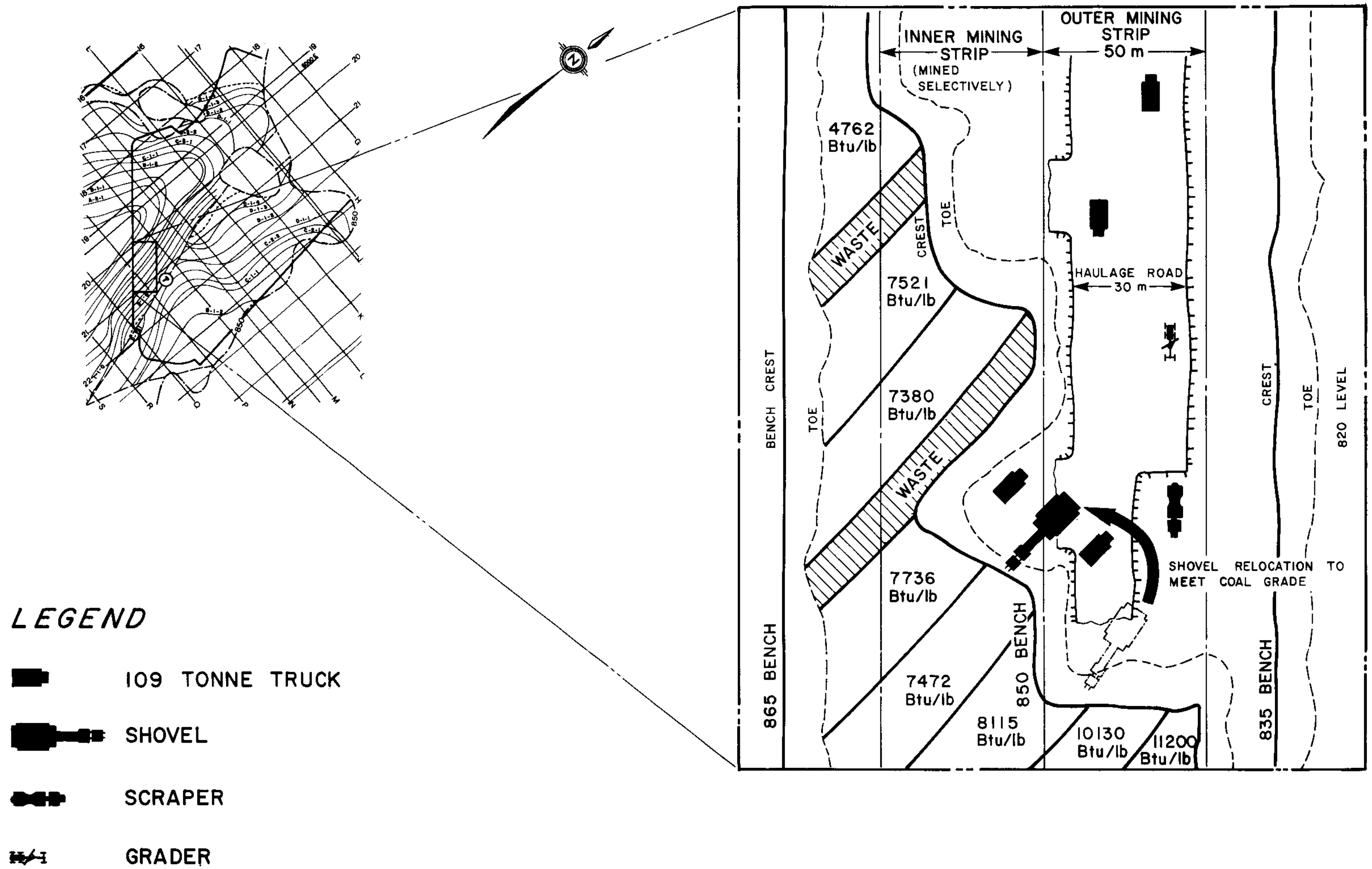


FIGURE 4-4

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

TYPICAL MINING OPERATION
850 m BENCH YEAR 5

A continuous program of fill-in drilling will be required up to six months in advance of mining. The concept is for close-spaced auger drilling from working benches. Spacing will vary with experience and the particular localized requirement, but can be expected to range from 20 to 30 metres. Occasionally it may be necessary to extend holes more than one bench depth, or 15 metres where advance data are required. Such a program could be expected to provide data on the basis of 500 to 1000 cubic metres of coal per metre of drill hole compared to 23 000 cubic metres (35 000 tonnes) at present. Sample data would be obtained from electro-logging with confirming assays from drill cuttings required.

It is recommended that sample records be displayed on large scale bench plans. These would modify both the original wide-spaced drill results and the computer-interpolated 25 x 25 metre block files presently available. Mapping of mining faces would also be carried out and the information collected would be used to continuously revise present coal inventories and short-term production plans.

The generating station may call on the mine for low sulphur coal from time to time when climatic conditions dictate reduced sulphur dioxide emission. There are two likely supply modes: either direct supply from the pit, bypassing the blending system, or supply from a special stockpile at the blending area previously stocked by the mine. Whichever mode is chosen, the low sulphur coal supply rate from the mine must be sufficient to feed the generating station continuously in case its own 48-hour stockpile of low sulphur coal has been depleted. Based on information from B.C. Hydro it is expected that the requirement to burn low sulphur coal is only about 3% of the total time or 11 days per year.

To supply either mode calls for selective mining of low sulphur coal. The only certain source is in D-Zone which would still require careful sampling to avoid sending coal with high local concentrations of sulphur to the generating station.

454 QUALITY VARIATION - R.O.M. COAL

A simulation study was carried out in order to forecast the variations in quality of R.O.M. coal that could be expected at the blending facility. The coal quality data used were derived from existing bore hole assays, but they do not represent the in situ coal quality that would be encountered during mining operations; they reflect only the anticipated quality variations. The results of the study were used as preliminary design data for the coal blending and stockpile facilities.

In the simulation carried out, three shovels mining 50-metre wide strips were operated over a period of seven days in different types of coal found in the mine. The shovels were not directed to mine selective grade blocks of coal. Two of these were placed in the highly variable coals found on the west flank of the mine and the third was located in the better quality D-Zone coal. The simulation allowed for production interruptions due to operating problems such as shovel breakdowns, particularly for the shovel in the better quality coal. Each shovel mined separate blocks of coal, 500 bank cubic metres in size per hour. Btu grades were assigned to each block from existing bore hole assays.

Compilation of the quality variations in R.O.M. coal production is shown graphically in Figure 4-5. The first three graphs show the individual Btu variation per mining shovel over one week of production; the fourth represents the variation for combined production from the three shovels. The coal quality variation by hour for a given shovel is demonstrated to be as much as ± 5000 Btu, and for the combined shovels, ± 4000 Btu. These study results were used to size the blending piles as outlined in Section 683.

455 GRADE CONTROL SIMULATION STUDY

455.1 Year 3 Phase II Study

A detailed simulation of Year 3 mine production was carried out using available Phase II mine model grade information to determine the grade control attainable in the mine by following the operating practice outlined in Section 452. In the simulation six 11.5 cubic metre shovels were assumed for the operation, two of which worked only in waste materials while the rest mined both coal and waste.

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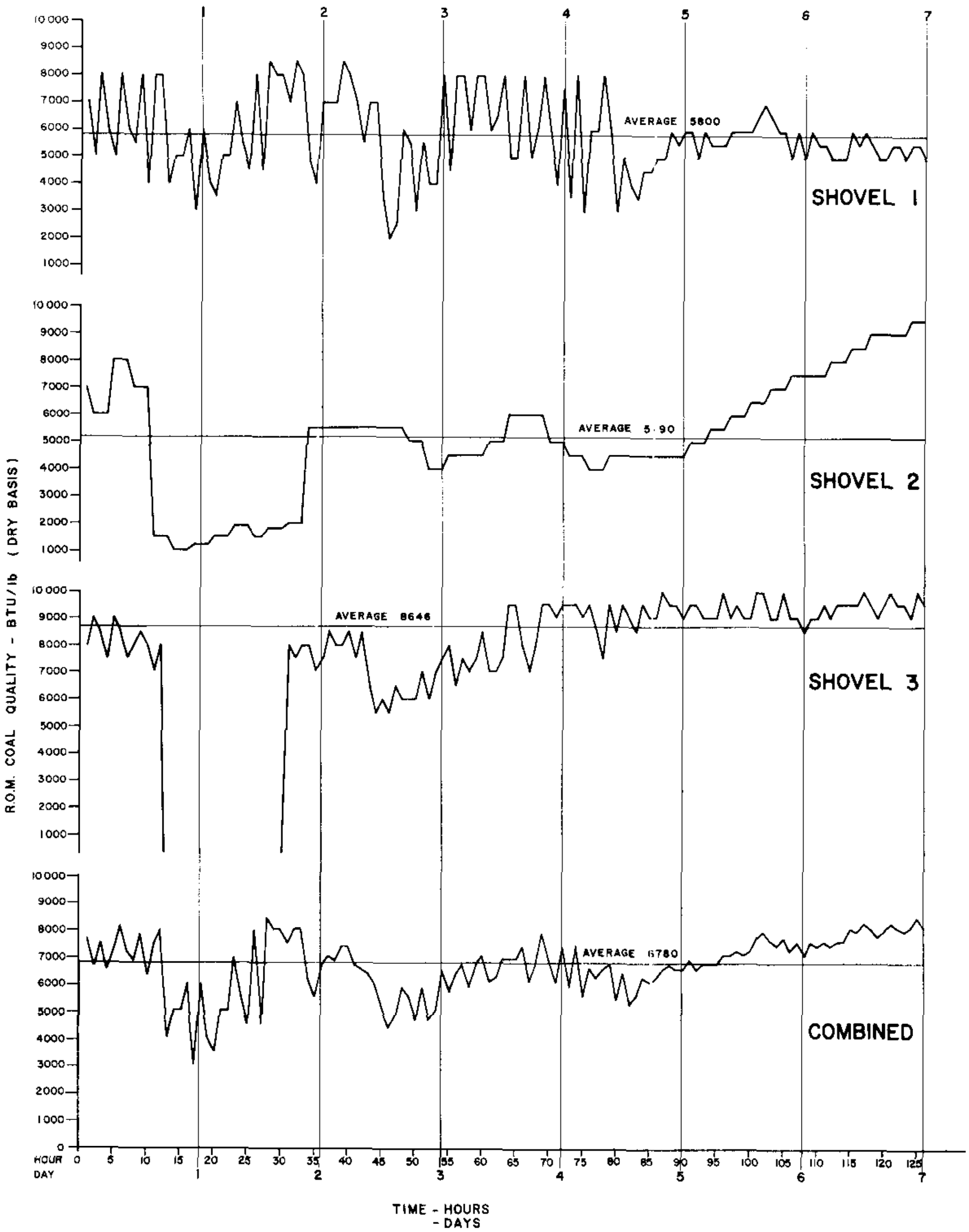

FIGURE 4-5

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

VARIATIONS IN COAL PRODUCTION
VS
TIME

The first part of the study was to estimate the monthly level of grade control that could be achieved over a 12-month mining period. The second was to investigate the type of equipment scheduling required and problems that may occur in trying to achieve a weekly average coal grade close to the monthly average.

(a) Monthly Investigation

The Year 3 coal production outlined on each bench was divided into 50-metre wide strips, irrespective of seam boundaries. Each shovel was assigned 500 production hours per month at 625 bank cubic metres per hour; thus each advanced 0.8-metres per hour on the 50-metre wide strip being mined. The sequencing of the four coal shovels was continued for 12 months and the monthly average Btu grades achieved, including adjustments for dilution, are shown below:

Month	Btu/lb (dry basis)
1	7221
2	7136
3	7294
4	6924
5	7005
6	7076
7	7097
8	7141
9	7179
10	7170
11	7155
12	7218

Year 3 average .... 7135 Btu

(b) Weekly Investigations

To estimate quality variations and equipment scheduling problems that may occur over the short-term, months 2,3,4 and 5 were investigated in greater detail. The same mining approach was adopted and equipment was manoeuvered more carefully to meet a weekly grade average close to the monthly average. The following simulation details are illustrated on Figure 4-6.

- lengths of time that each shovel excavated coal and waste

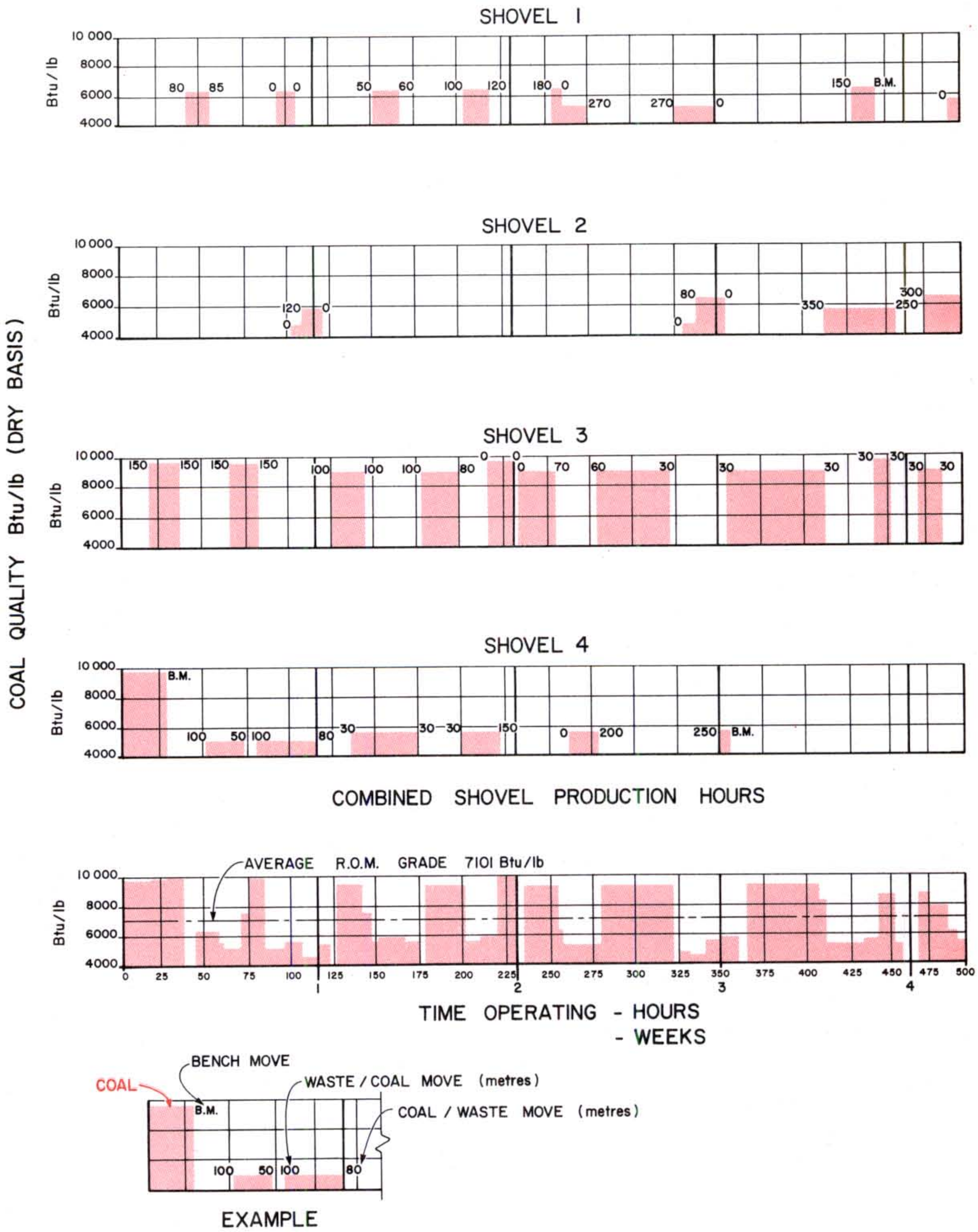

FIGURE 4-6

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

MINING FEASIBILITY REPORT

WEEKLY RUN OF MINE COAL QUALITY VARIATION

- the grade of the coal mined by each shovel during the periods of excavations

- distance of shovel moves in metres and time period when moves occurred

- combined coal production results for all shovels

Scheduling of shovel moves between benches was restricted for this simulation to those occasions when all coal and waste contained in the assigned mining strips on a given bench were completely mined-out. At the level of grade control planning carried out, some problems were encountered in meeting the desired grade when a shovel had mined out the coal, but not the waste, on a particular bench. The longest period for which this problem persisted was two weeks; however, it is felt that with the type of grade control planning recommended in Section 453, improved equipment scheduling to avoid this situation should be possible.

455.2 Year 1 Phase III Study

To support the simulation results of Year 3 production calculated from Phase II mine model information, a new simulation was carried out on Year 1 production using Phase III information. A set of nine bench plans, one of which is shown in Figure 4-7, was prepared based on the Phase III model, calculated volumes, and Btu grades. Mining was sequenced in strips for which monthly grades and volumes were calculated based on a similar equipment schedule to that determined previously. These results are shown in Table 4-8.

Production in Year 1 is considerably lower than that in Year 3 and should require only four shovels. In the simulation 16.8 cubic metre shovels were scheduled to operate in the following areas of the mine: No. 1 shovel, 100% utilized, developed the main haul road on the west side of the mine including the unloading of a portion of the active slide; No. 2 shovel, 83% utilized, mined the low-grade coal and upper bench contour development on the west side; No. 3 shovel, 57% utilized, mined primarily better-grade coal on the 850-metre benches, and carried out some east side stripping on the lower benches; No. 4 shovel, 60% utilized, stripped only on the east side.

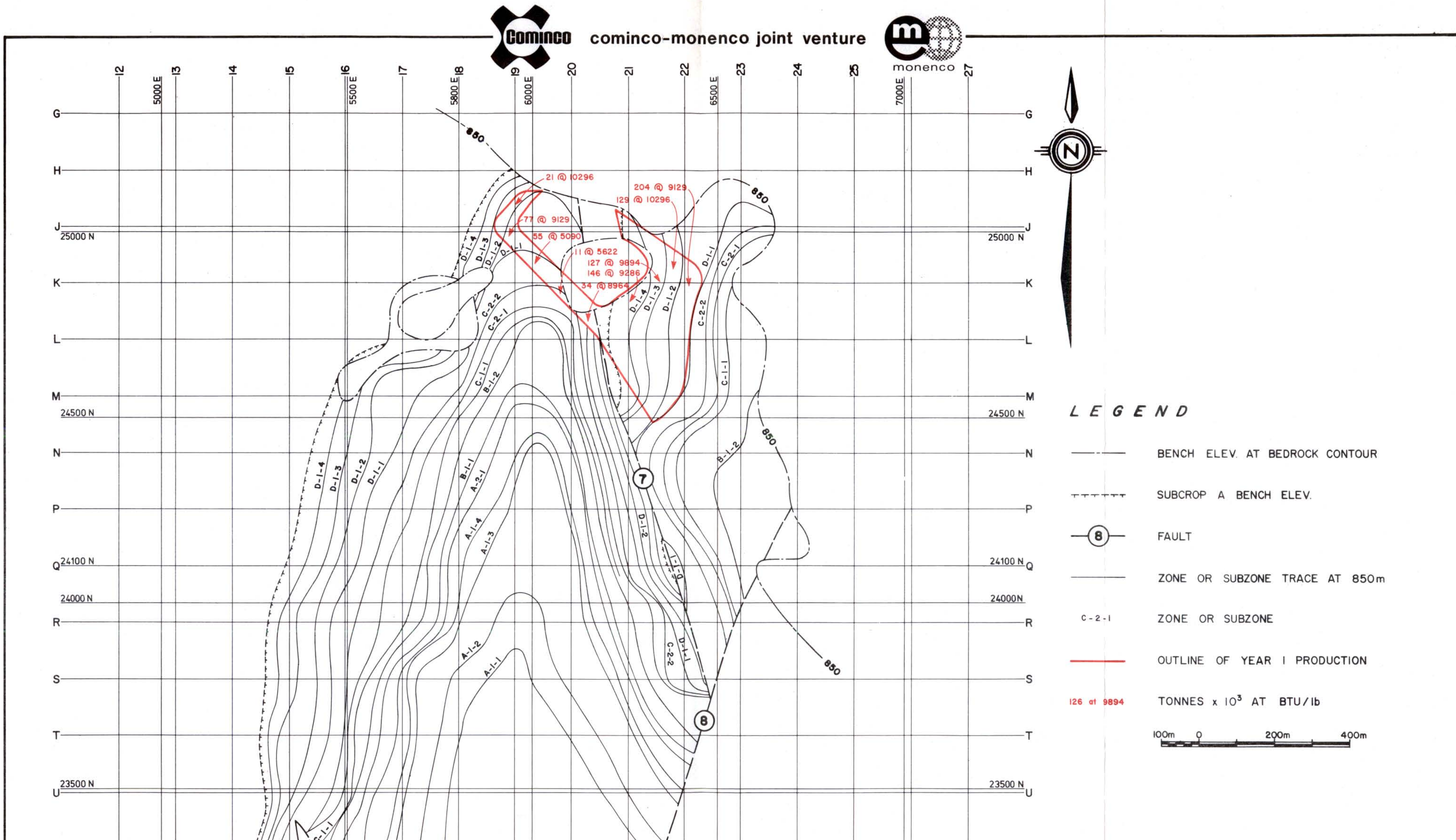

FIGURE 4-7

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

BENCH PLAN
ELEVATION 850m ASL

TABLE 4-8

YEAR 1 PHASE III

Monthly Production During Year 1

Hat Creek Project Mining Feasibility Study 1978

Month	Coal In Situ ($t \times 10^3$)	Calorific Value In Situ (Btu/lb)	Surplus & Shortfall of Grade (Btu/lb)
1	235	7462	-25
2	235	7491	+ 4
3	235	7500	+13
4	235	7480	- 7
5	235	7494	+ 7
6	235	7473	-14
7	235	7489	+ 2
8	235	7480	- 7
9	235	7469	-18
10	235	7460	-27
11	235	7489	+ 2
12	235	7557	+70
TOTAL	2820	7487	

The simulation showed that manoeuvering of shovels to specific grade blocks would probably not be difficult, once sufficient working widths are established on the new benches developed during Year 1.

456 SECURITY OF COAL SUPPLY

Changes in fuel requirements as a result of capacity changes for periods up to six months in duration demand that the coal supply be secure. Adequate quantities of acceptable quality coal should be exposed at any given point of mine production to accommodate possible fuel requirement changes.

The end of Year 5 was chosen for the investigation of security of coal supply. All coal available in the exposed faces were outlined on the working plan and grades were assigned in accordance with available bench plan information. One requirement of the investigation was that the handling of available coal would not require extensive road and ramp preparation.

From the results of the compilation shown on Table 4-9 below, it can be seen that six months of production, averaging 7449 Btu/lb, should be available to the mining shovels at the end of Year 5.

TABLE 4-9

Uncovered Coal Reserves Available in Year 5

Bench	Tonnes x 10^3	Average Btu/lb
940	430	5461
880	1340	5760
865	1085	5976
850	307	8101
835	984	8348
820	1115	7970
790	1340	9790
Total	6651	7449

457 CONCLUSIONS

The data available from the simulation studies were used in the design of the stockpile blending system required to prepare run-of-mine coal for the generating station. The studies also indicated the type of grade-control planning and equipment scheduling desired. The following conclusions have been drawn:

(a) As recommended, the mine operating practices should achieve an acceptable average monthly coal quality within the 7000 to 7800 Btu/lb range which is acceptable to the generating station. The studies carried out support the opinion that the overall operation has the flexibility to selectively mine defined blocks of coal in order to meet short-term grade requirements.

(b) The simulations also showed that, most of the time, the variety and quantities of coal required to achieve the average grade should be within capability of the shovel/truck systems.

(c) The mining practices adopted and reflected in the mine plans support the opinion that adequate quantities of better-grade D-Zone coal will always be available; replenishing the high-grade, low-sulphur stockpile would not appear to be a significant scheduling problem.

(d) Should an increase in coal production be required due to a temporary change (6 months) in the generating station capacity factor, adequate coal should be available in the mine to accommodate the increased requirement.

SECTION FIVE

MINE DEVELOPMENT

5.1 INTRODUCTION

This section of the report discusses alternative mine development concepts such as pit exit, location, depth, and the sequence of mining in light of the geological, geotechnical, and quality considerations previously outlined. An expanded pit to 450-metres ASL is also described in this context. Detailed mine planning methodology is discussed, leading into the development of detailed pit and waste dump plans for the recommended mining system.

Since the bucket wheel systems were considered in a separate study by NAMCO, this section of the report is principally oriented to the shovel/truck/conveyor methods of mine development although where appropriate, references to the bucket wheel alternatives have been made.

5.2 PIT ACCESS

Several approaches for opening and developing the No. 1 Deposit were investigated. This involved two significant alternatives: a north-exiting mine, which was recommended, and a south-exiting mine.

The northern exit offers the following advantages:

(a) At the northern edge of the deposit, significant quantities of better-grade coal are believed to be available close to the surface as the coal either outcrops or subcrops below easily-removed surficials. Opening up the mine in this location offers lower initial strip ratios than would be possible from other locations, and supports the belief that acceptable fuel grades are available for the start-up of operations.

(b) The better-quality coals at the north edge directly overlie footwall rocks that are considered competent in this locality; the angle of plunge of the structures in this area is less than the maximum wall slopes recommended by the geotechnical consultant. In addition, the mine layout is such that the main access conveyorway should not be dislocated by future mining. The particular combination of footwall competence, shallower pit walls, and opportunity for undisturbed access is not available in other locations.

(c) Houth Meadows is the most northern dumping site available in the Hat Creek mining area. It would require the least vertical transport of materials, and economics favour it being utilized to capacity. Since any future mining of the No. 1 Deposit will be a southerly extension of the 35-year pit limits, the more southerly dumping sites should be conserved. The northern mine access accommodates maximum utilization of Houth Meadows.

(d) The northern access is closer to the site selected for the generating station and to an area permanently available for the surface facilities. These facilities include crusher houses, coal blending and stockpile areas, administration buildings, maintenance shops, and water treatment facilities.

Space is available in the event that coal beneficiation is ever considered necessary. The topography in the area selected for these facilities lends itself to relatively low-cost installations.

(e) In the final design, the in-pit mine conveyorway, together with its service road, is located on a structural ridge of footwall waste between No. 7 and No. 8 faults. Since the ridge intrudes into the pit on each bench, sites for truck unloading stations can be selected closest to the centre of gravity during each phase of the pit development. This will result in minimum truck haulage distances.

Disadvantages of the northern exit are:

(a) proximity to the northwest active slide with its potential for failure and subsequent extensive clean-up requirements;

(b) the northwestern truck exit is partially located in materials comprising the inactive slide and higher road maintenance costs are anticipated due to soil creep; and

(c) for a bucket wheel system it would appear that permanent conveyors would have to be located in the slide area.

The risk of the central conveyor operation being halted due to earth movements of any kind is considered to be low, but provision has been made for several truck and service road entries; this accommodates in part any pit wall movements near the northwest slide. In addition, haulage road widths were increased from a pit standard of 30 metres to 60 metres in the slide area to provide room for clean-up should sloughs occur. Pit standards include:

- haul roads - 30-metres wide - maximum ramp gradient 8%
- service roads - 20-metres wide - maximum gradient 10%
- safety berms in the contact zone between surficials and bedrock - 30-metres wide - maximum gradient 10% (60-metres wide below slide surficials)

5.3 VERTICAL MINE DEVELOPMENT

The significant factors leading to the recommendation of the proposed mining strategy are outlined in Section 2.3 - "Distribution of Coal by Grade". In brief, the mine should: open on better-than-average grade coal at the north end of the deposit, achieve a low initial strip ratio, and deepen rapidly to maintain adequate, exposed reserves of D-Zone coal. Figure 5-1 provides a cross-sectional illustration of the mine development. The amount of lower grade coal required for blending with D-Zone coal, and the economics of uniform overall materials movement tended to control the rate of lateral mine development. Other strategies investigated, including a shallow (rapidly-widening) approach, were not able to maintain uniform coal grades over significant periods of mine production, nor provide adequate quantities of acceptable fuel early in the mine life.

Aspects of mine development which require special attention in the layout of the periodic mine plans are discussed below.

531 MINING OF THE SLIDE AREA AND SPECIAL SLOPE STABILITY CONSIDERATIONS

The principal slope stability problem is considered to be the potential for ground movement in the active slide close to the northwest part of the pit. There is also concern that failure could occur in the materials underlying both the active and inactive portions of the slide area during adjacent pit excavating operations. Movements are expected to take the form of soil creep rather than rapid failure. The main problems therefore will involve periodic removal of sloughs, comprising primarily bentonitic clays, and repair of constantly deteriorating roads. The rate of failure is not expected to be sufficiently rapid that main access ways should require replacement during the life of the mine. The proposed mine plan defers until Year 6 any mining in the active slide materials, which will permit on site experience to be gained in handling of these materials.

To achieve this deferment, pit walls are extended to the south of the active slide area and a semi-permanent, north exiting, main haulage road to serve the west flank of the pit would be established. After Year 6, this road will be relocated progressively further into the slide until at approximately Year 15, it will have reached the ultimate mine wall position. All roads in the slide area will be constructed 60 metres wide to accommodate clean-up. During

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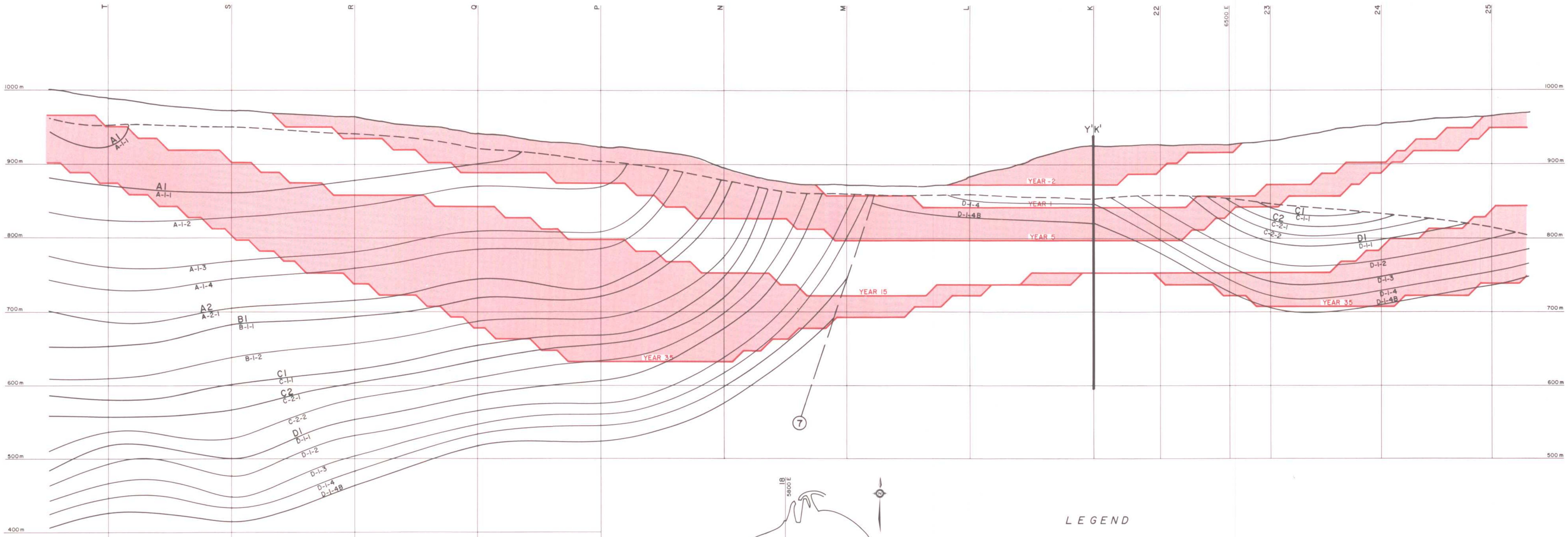

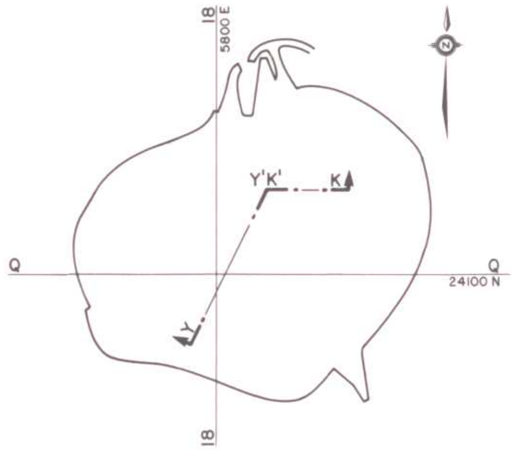

KEY PLAN
35 YEAR PIT

LEGEND

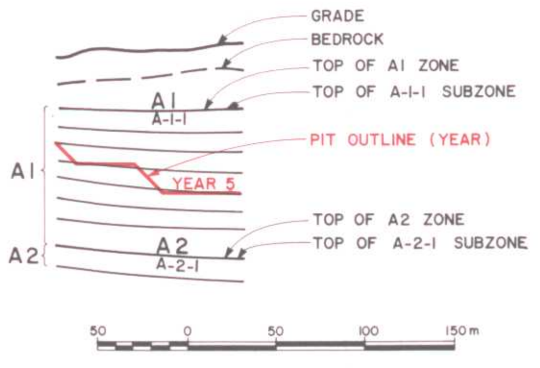

FIGURE 5-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

MINE DEVELOPMENT
CROSS SECTION

Year 1 benches 910 and 925 are excavated into the active slide for monitoring purposes as shown on Figure 5-3.

Small lakes on the west slope of the valley are believed to contribute to the instability of the slide through the hydraulic lifting effect of groundwater seepage. Development plans call for the drainage of some of these lakes prior to the commencement of mining. It is expected to take several years for the effects of drainage to become apparent in slowing the movement of the slide.

Throughout the pit development nearly all materials will be removed from the pit via the central conveyor. Truck access to both the conveyor loading stations and to out-of-pit maintenance facilities is essential. Due to the possibility of the northwest roads being out-of-service at any time, two additional major road accessways have been provided on the mine plans. These will contribute to a more flexible mining operation.

532 TRUCK UNLOADING STATIONS

Investigations were carried out to optimize the vertical locations of the proposed truck unloading stations that feed the central conveyor. Initial evaluation of the probable cost-savings indicated that the construction of three unloading stations during the pit development was desirable. Subsequent study located the three unloading stations such that uphill haulage was minimized and the flow of materials to each station remained relatively uniform throughout the project life. Selected elevations were 895, 820, and 730 metres ASL. These unloading stations are required for operations in Years -2, 6, and 21, respectively.

533 SCHEDULING OF CONSTRUCTION MATERIALS

Construction materials must be excavated from the pit for use during the construction of support facility foundations, dump embankments, and roads. The greatest quantities will be required early in the development, but demands will continue throughout the life of the project. Table 5-1 gives an outline of completion dates for various construction and dump development projects. The mining strategy encourages that adequate quantities of these materials are recovered, even though the general approach is to defer stripping as much as possible. Table 5-2 compares the quantities of materials required by year to meet the construction and dump

TABLE 5-1

Estimated
Completion Dates for Construction Projects
Using Mine Waste Materials

Hat Creek Project Mining Feasibility Report 1978

Project	Materials Required	Completion Date End of
Main waste conveyorway	pervious materials	Year -2
Water treatment dam	impervious materials	Year -2
1st conveyor pad	pervious materials & general mine waste	Year -2
Low-grade coal stockpile sub-base	general mine waste	Year -1
2nd conveyor pad	pervious materials & general mine waste	Year -1
Mine perimeter road	pervious materials & general mine waste	Year 1
HOUTH MEADOWS EMBANKMENT HME-1		
Elevation 865 m ASL	pervious materials	Year -2
900 m ASL	pervious materials	Year 2
935 m ASL	pervious materials	Year 6
975 m ASL	pervious materials	Year 10
MEDICINE CREEK EMBANKMENT		
Elevation 1020 m ASL	pervious materials	Year 16
1065 m ASL	pervious materials	Year 17
1100 m ASL	pervious materials	Year 22
HOUTH MEADOWS EMBANKMENT HME-2	pervious materials	Year 23
HOUTH MEADOWS EMBANKMENT HME-3	pervious materials	Year 31

TABLE 5-2

Comparison of Construction Grade Material
Excavated from the Pit and the Materials Required

Hat Creek Project Mining Feasibility Report 1978

Year	Pervious Construction Materials Required	Pervious Construction Material Excavated
-3	2.05 x 10^6 BCM	2.05 x 10^6 BCM
-2	4.35	4.61
-1	2.98	5.30
1	6.51	8.30
2	4.73	8.30
3	0.60	8.30
4	6.28	8.30
5	6.92	8.30
6	2.30	7.85
7	0.30	7.85
8	0.30	7.85
9	4.30	7.85
10	4.80	6.24
11-15	3.30	23.70
16-21	37.60	44.88
22-26	7.35	23.58
27-35	6.40	-
Total	101.07	183.26

development projects with the quantities of materials actually excavated each year.

While the decision to utilize two waste dump areas was based largely on operating considerations a strong influence on this decision was the fact that adequate quantities of pervious construction grade materials were available for embankment construction from the 35-year pit, and that shortages of these materials would likely occur in any future expansion beyond Year 35. The Medicine Creek embankment is planned to be constructed at its full design width which would allow for utilization of the ultimate dump capacity should mining continue past Year 35. Approximately 26 million cubic metres of pervious, construction grade materials would be placed in the construction of the Medicine Creek embankment. These construction materials might otherwise have been wasted within the dump area if not utilized in this manner.

534 DYNAMIC BENCH ARRANGEMENTS

Following the initial years of operation, 12 or more mining benches will be considered to be active. The working area for any one shovel will be four benches on one side of the pit comprising one active bench and three dormant benches. The slope of the pit walls of these benches is defined as the dynamic slope. Within the open pit, the layout and operation of the mine is generally based on a dynamic slope limitation of 30°; where the risk of slumping is greater, the limitation is reduced to 20°.

The slope represented by the series of four-bench groups from pit floor to crest is termed the working slope. In this case the working slopes are restricted to the geotechnical guidelines for final pit walls. Under this guideline, the bench plans were laid out for operating flexibility in selectively mining blocks of coal to produce an average fuel grade. This method of bench development is illustrated in Figure 3-1 which provides an example of dynamic slopes for a typical bench operation.

5.4 EXPANDED PIT

Development of mine plans for the 35-year pit, including the location of surface facilities, waste dumps, and drainage facilities, must take into consideration any potential expansion of the project that would mine out a significant portion of the remaining No. 1 Deposit. In conjunction with geotechnical recommendations, a mine plan for an expanded pit to the 450-metre ASL floor level was developed. An assessment of the economics of the expanded pit and of its inter-relationship with the adjacent No. 2 Deposit has not been made in this study; however, in practice, this type of evaluation would normally be completed at various stages of the mine development.

The 450-metre floor level determined for the expanded pit was not based on a specific geological bench mark; rather it was considered to display the practical maximum pit dimensions.

The northernmost coal above 650-metres ASL is mined out at the completion of the 35-year project, and future expansion is southward down the plunge of the coal stratigraphy. It can be seen from the site layout plan (Figure 5-9), that the location of the service facilities and waste dumps for the 35-year pit are well beyond the rim of the expanded pit. This would in turn almost double the No. 1 Deposit coal supply as presently forecast for the 35-year project.

It was recognized that most of the surficial materials suitable for constructing retaining embankments at the waste dumps may be mined-out during the 35-year development; as a result, special attention was paid to retaining embankment design. The strategy adopted was to place as much of the suitable granular construction material as possible into the embankment and to prevent mixing or wasting of these materials. The guidelines for embankment construction were based on the ultimate design requirements of an expanded pit, rather than the limitation of 35-year dump capacities. At the end of 35 years, the embankments should be left with broad tops, capable of accommodating dump expansions to the maximum capacity with relatively little incremental, vertical construction. It is recommended that the utilization of construction material be based on long, optimal utilization term, rather than short-term optimization.

Statistics for an expanded pit are provided on Table 5-3.

TABLE 5-3

Summary of Production Statistics
for the Expanded Pit

Hat Creek Project Mining Feasibility Report 1978

	CMJV 35-Year Pit	Increment to 450 m ASL	Expanded Pit
PRODUCTION MATERIALS IN SITU (BCM x 10^6)			
Coal (>4000 Btu/lb)	234.32	205.20	439.52
Low-Grade Coal (3000 to 4000 Btu/lb)	8.96	38.27	47.23
Wastes above Bedrock	161.15	642.32	803.47
Wastes below Bedrock	281.85	215.04	496.89
COAL IN SITU ABOVE 4000 Btu/lb			
Tonnage (x 10^6)	344.19	322.19	666.38
Quality			
Calorific Value (Btu/lb)	7515	6406	6978
Ash Content (%)	35.1	42.5	38.7
Sulphur Content (%)	0.49	0.55	0.52

Strip Ratio

CMJV 35-year pit .... $\frac{8.96 + 161.15 + 281.85}{344.19} = 1.3$ to 1

Increment to 450 m ASL .... $\frac{38.27 + 642.32 + 215.04}{322.19} = 2.7$ to 1

Total Expanded Pit to 450 m ASL........... $\frac{47.23 + 803.47 + 496.89}{666.38} = 2.0$ to 1

Relationship to No. 1 Deposit Proven & Probable Coal Reserves

Expanded Pit to 450 m ASL .... $\frac{666.38}{716.50} \times 100 = 93\%$

5.5 MINE PLANNING PROCEDURES

551 COMPUTER APPLICATION

As described in Section 1, mine planning was carried out using a combination of computer and manual methods.

The twin objectives in using computer techniques in this feasibility study were: first, to build a geological model that describes in mathematical terms both the stratigraphy and the coal reserves; and second, to apply this stored geological data in mine planning. Section 551.1 following describes the three phases in development of geological models used for mine planning, and makes reference to the final element of the basic "MEPS" system called MEPS Mine.

551.1 MEPS Mine

MEPS Mine consists of four routines. The first defines costs and organizes the cut-offs to be subsequently used in pit evaluation. The second provides a mathematical means of modelling a pit surface. The third routine carries out the actual analysis or evaluation of the materials contained between two modelled surfaces such as the topographical surface and the worked-out position of any stage of a series of nested pits. The fourth routine provided graphic displays used either directly in the planning process, or as back-up data in determining whether the objectives were being met.

The following details the use of MEPS Mine for the three phases of model building.

Phase I - 4-Zone Model
The principal use of this model was to provide the initial planning tool for analyzing the production materials of staged pits. When superseded by more refined models, Phase I continued to be used for modifying and refining the basic MEPS routines.

Phase II - 12-Sub-Zone Model
This was the principal computerized planning tool used in the study, with over 200 evaluations. Based on 12 stratigraphic horizons, a set of ore reserves was generated and quality trending within each horizon was accommodated. The model provided localized

breakdowns of these overall deposit reserves onto mining benches for use in planning of mine excavations. It handled the repetitive process of optimizing coal resource utilization, and provided statistics for detailed production scheduling and preliminary cost estimating. Phase II was eventually superseded by a more refined model Phase III, used in preparation of the final report.

Phase III - 14-Sub-Zone Model

The evaluation of the final set of shovel/truck/conveyor nested pits was done by the MEPS Mine, using Phase III geological modelling. The significant changes in this model are described in Volume II.

The evaluation of several pits generated by Phase III were compared to those evaluated in Phase II. It was concluded that differences in the early stages of pit development were sufficient to warrant using Phase III for all final evaluations of production statistics and cost estimates.

In order to compare the project pit to the major portion of the resource in No. 1 Deposit, a large pit was described which extended over an area greater than that defined in the computer model. It was necessary to evaluate this overflow manually. The portion of the pit within the grid limits was evaluated by MEPS Mine.

551.2 Production Scheduling Routines

Production scheduling statistics were generated by a combination of manual and computer techniques. Manually designed, incremental nested pits were evaluated by the computer and modified repetitively in the direction which produced the most desirable quality at the lowest strip ratio. Final production statistics were achieved by graphing the incremental statistics such that material volumes mined represented each year of the project life.

552 BENCH PLANS

Two sets of bench plans were prepared, a main set based on information from the Phase II mine model, and a second set showing the distribution of seven categories of surficial materials. Each bench plan represents a horizontal plane through the mid-point of one 15-metre lift. These extended in a vertical series from the surface to the floor of the 35-year pit. Manual mine planning calculations for grade and volumes were carried out using these plans and then confirmed by the Phase II computer mine model.

553 PHASE II MINE PLANS

The mine plans prepared from the Phase II model information were designed to meet the specified* plant feed requirements at that stage of the feasibility study. Plans were prepared for Years -1, 1, 2, 3, 4, 5, 9, 12, 20, 26, and 35. Cumulative coal tonnages from the mine plan production figures were graphed against cumulative volumes of low-grade coal, seam and non-seam waste, and surficial materials. A production schedule was prepared from the graphed information. This schedule supports the possibility of achieving consistently average coal quality as well as a uniform annual material movement over the life of the mine. The previous production schedule from Phase II, which is not shown, was then superseded with a similar schedule applying the data from the Phase III model.

554 PHASE III RE-DESIGN CONSIDERATIONS

When the Phase III 14-sub-seam mine model became available late in the feasibility study, a sample calculation was carried out based on the Years 1 to 5 and Year 9 mine plans to determine what effects the application of Phase III data would have on the Phase II production figures. The primary Phase III data alterations were brought about by the incorporation of 1977 bore hole data.

From the comparison, it was indicated that significant variations in grade and volumes of material moved should not occur over long-term periods of 5 years or more. However, over the short-term yearly increments, localized re-distribution of coal grades and volumes resulted in significant differences in production figures.

To accommodate the changes, modifications were made to the Phase II pit plans, mainly in areas of the deposit changed by the 1977 bore hole data. These modifications were incorporated into a new set of computer-calculated production figures to assist in achieving the standards of grade consistency and materials movement achieved in Phase II.

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* memo from B.C. Hydro dated 3 April 1978

5.6 FINAL MINE PLANS AND PRODUCTION SCHEDULE

Based on the re-calculated production figures, mine plans were prepared for Years -2, -1, 1, 2, 3, 4, 5, 10, 15, 21, 26, and 35. Of these plans, Years -1, 5, 15, and 35 are included here as Figures 5-2, 5-3, 5-4, and 5-5, respectively. Shown on the mine plans are the extent of pit excavation at the given year and locations of in-pit haul roads, the pit perimeter road, and service roads running out of the pit.

The final production statistics based on the Phase III geological model are shown as incremental developments from pre-production to the final 35-year worked-out position in Table 5-4. The nominal, incremental periods shown on the table were then used to calculate the annual production statistics listed in Table 5-5.

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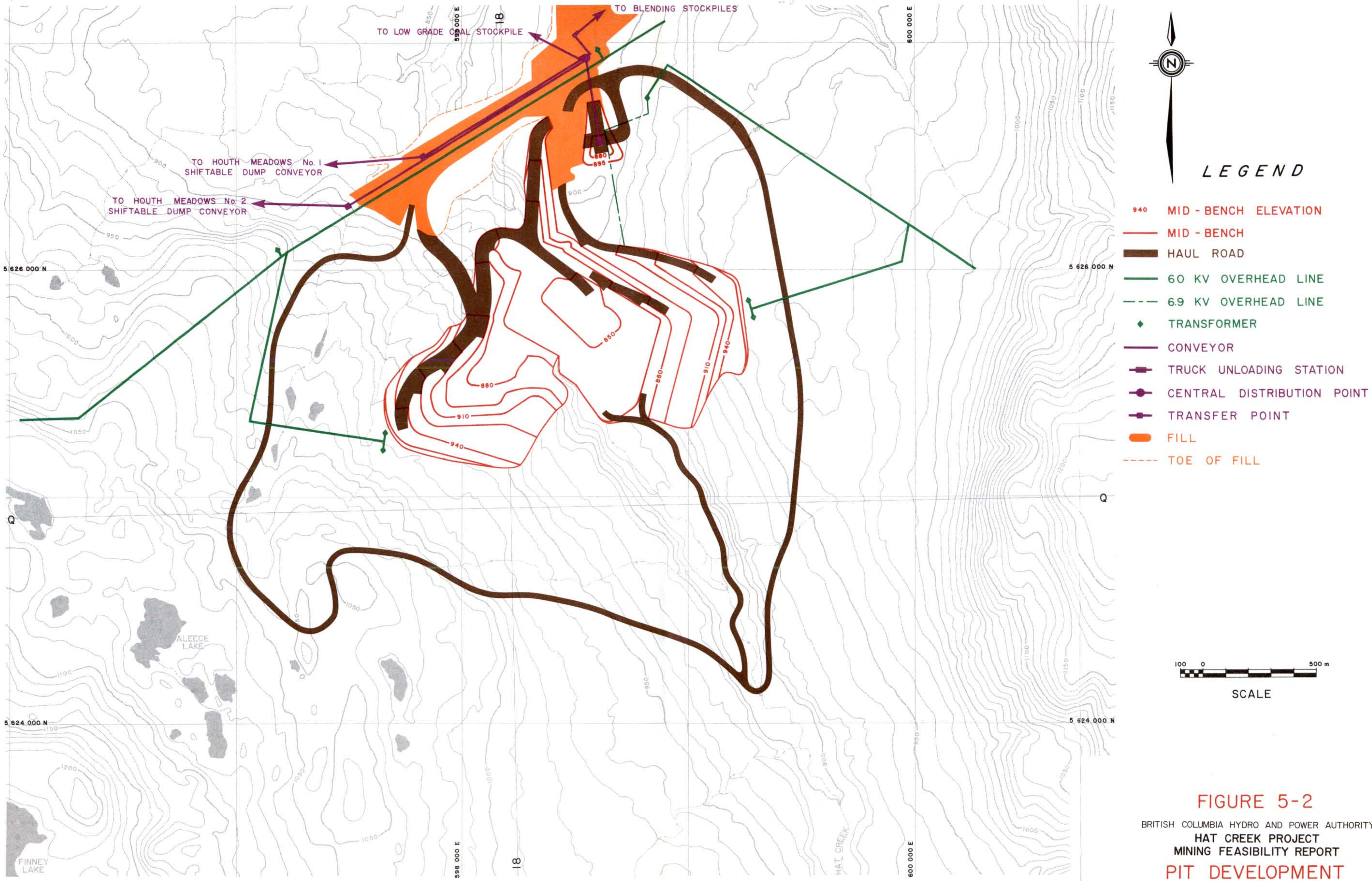

FIGURE 5-2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

PIT DEVELOPMENT
YEAR -1

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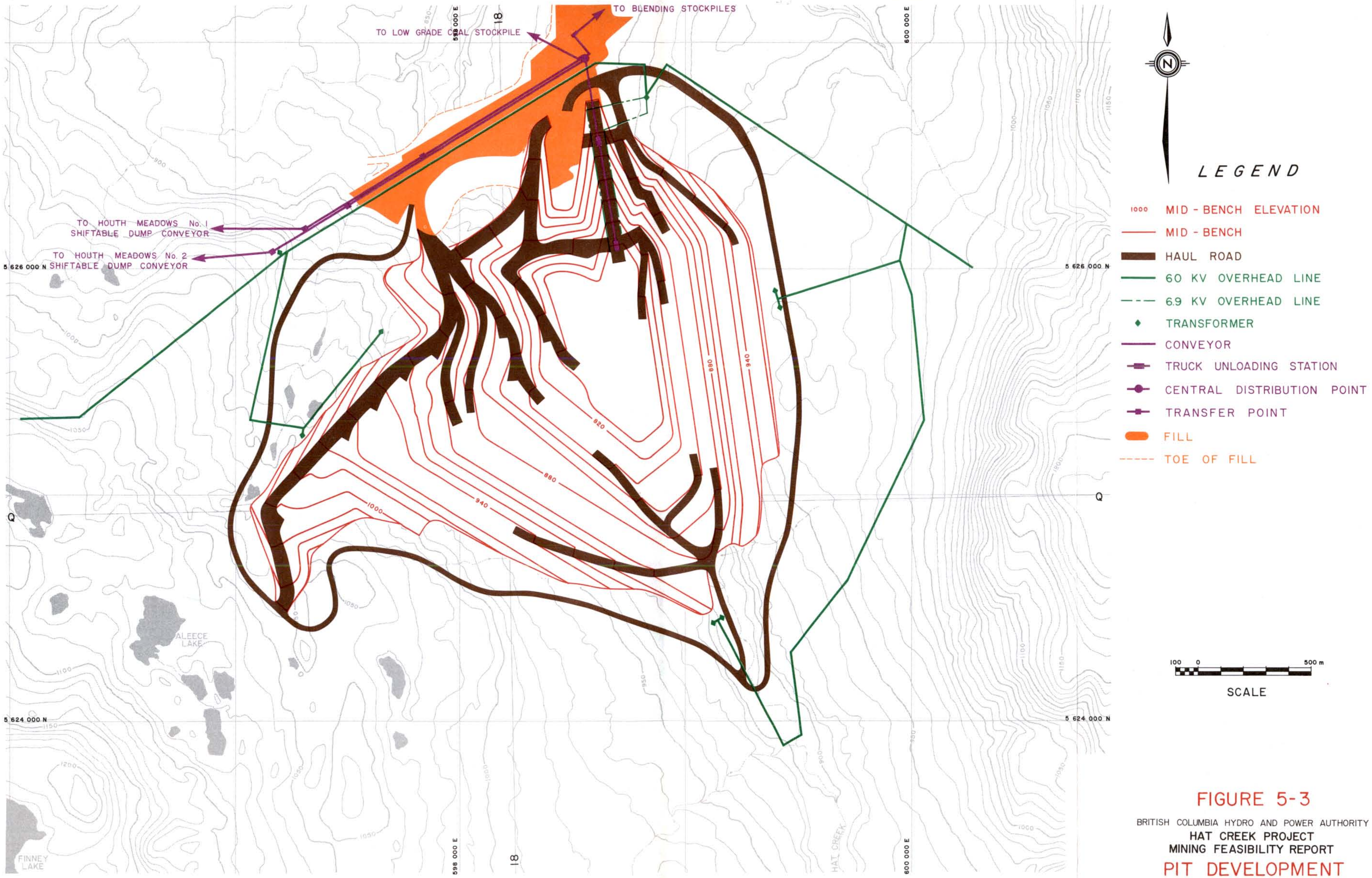

FIGURE 5-3

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

PIT DEVELOPMENT
YEAR 5

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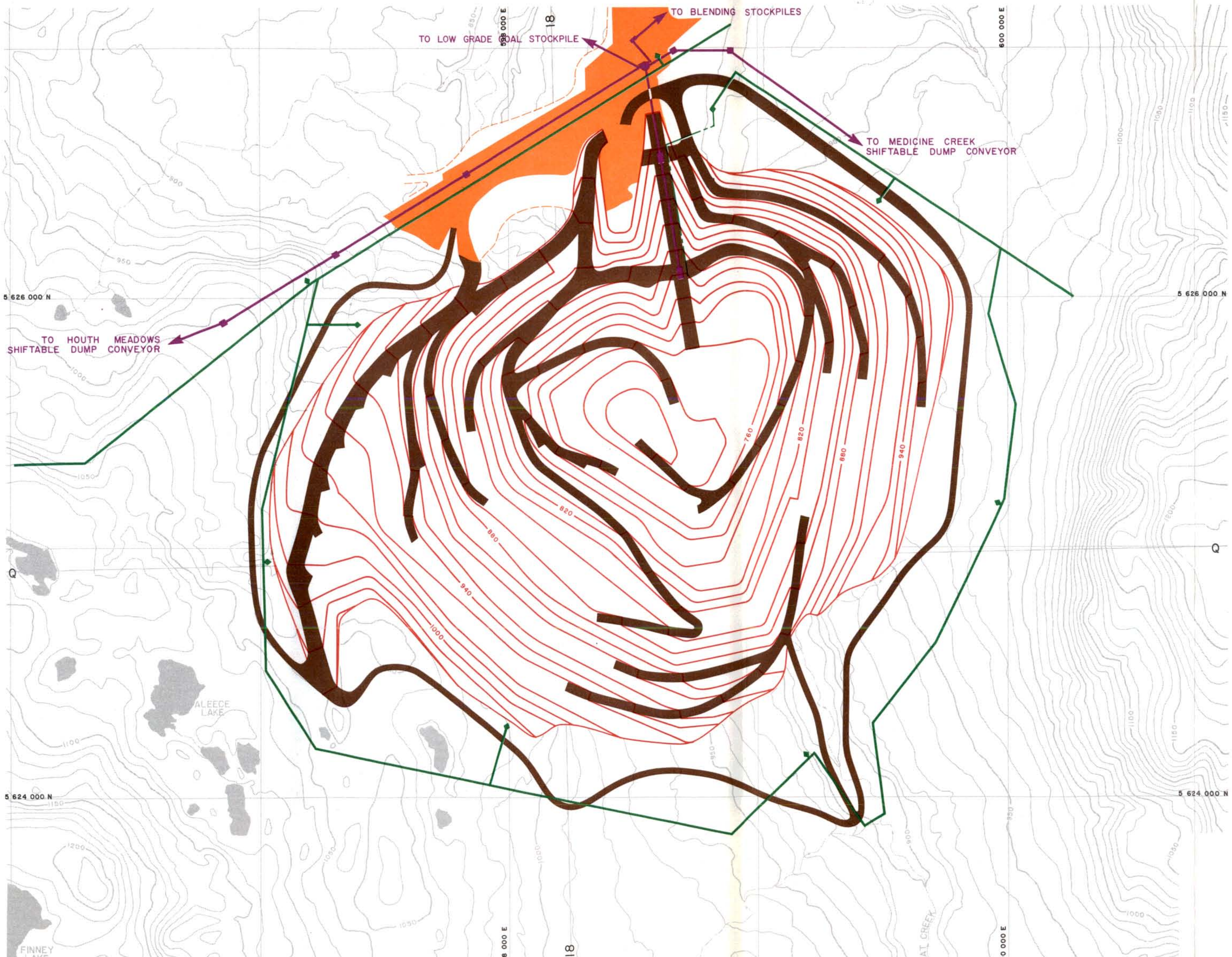
monenco



LEGEND

1000 MID - BENCH ELEVATION
MID - BENCH
HAUL ROAD
60 KV OVERHEAD LINE
6.9 KV OVERHEAD LINE
TRANSFORMER
CONVEYOR
TRUCK UNLOADING STATION
CENTRAL DISTRIBUTION POINT
TRANSFER POINT
FILL
TOE OF FILL

100 0 500 m
SCALE

FIGURE 5-4

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

PIT DEVELOPMENT
YEAR 15

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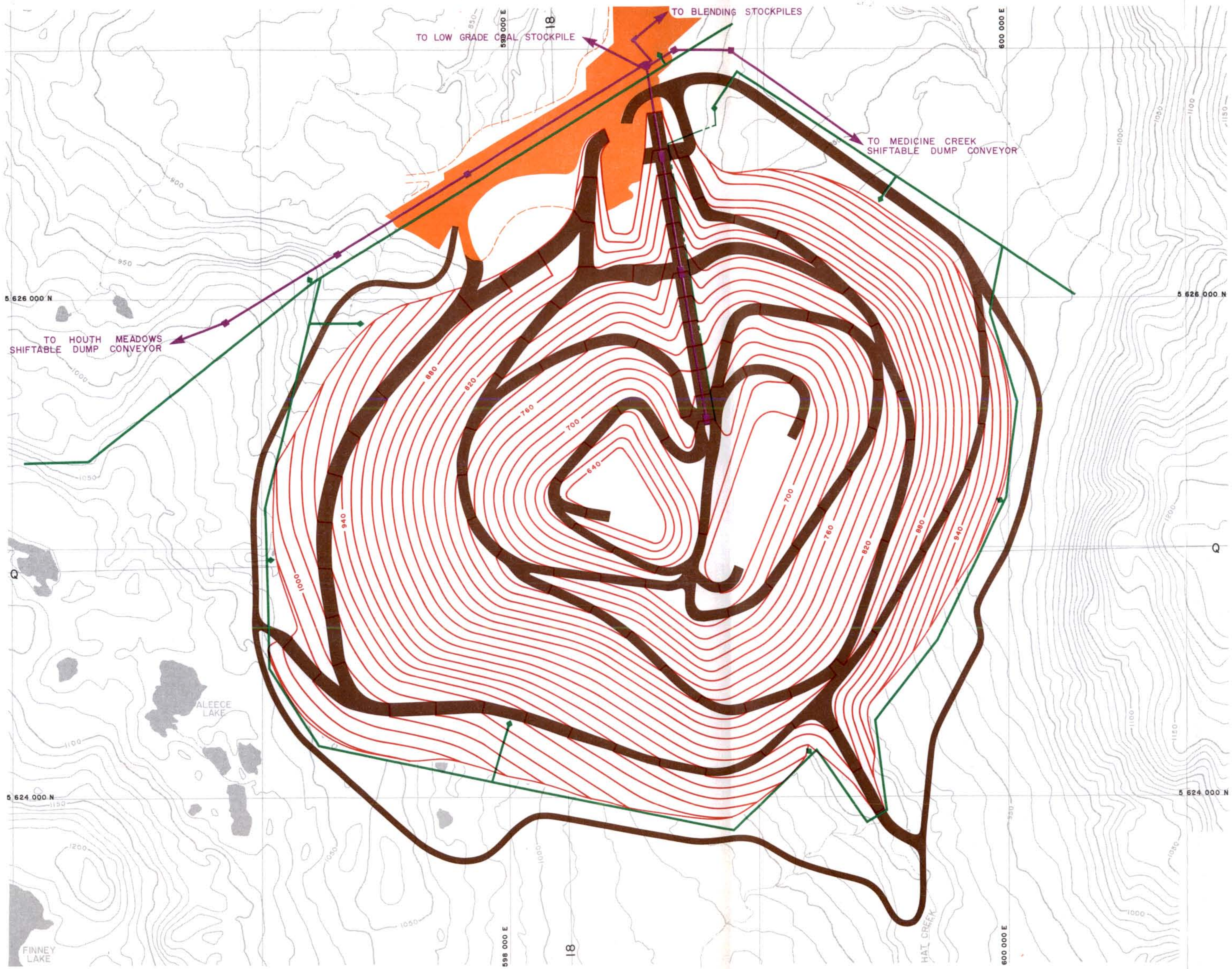


LEGEND

1000	MID - BENCH ELEVATION
	MID - BENCH
	HAUL ROAD
	60 KV OVERHEAD LINE
	6.9 KV OVERHEAD LINE
	TRANSFORMER
	CONVEYOR
	TRUCK UNLOADING STATION
	CENTRAL DISTRIBUTION POINT
	TRANSFER POINT
	FILL
	TOE OF FILL

100 0 500 m

SCALE

FIGURE 5-5

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

PIT DEVELOPMENT YEAR 35

TABLE 5-4

Estimated Coal Quantity and Quality
for Pit Development Periods
Hat Creek Project Mining Feasibility Report 1978

COAL ZONE	COAL QUANTITY (calorific value > 4000 Btu/lb) tonnes x 10^6	% of zone	COAL QUALITY Calorific Value (Btu/lb)	Ash Content (%)	Sulphur Content (%)
TO END OF PRE-PRODUCTION					
A-1	0.19	18.3	5217	51.2	0.68
B-1	0.32	30.8	7285	36.3	0.68
C-1	-	-	-	-	-
C-2	0.08	7.7	4429	55.9	0.34
D-1	0.45	43.2	9125	24.7	0.37
Total	1.04	100.0			
Average			7484	35.2	0.52
END OF PRE-PRODUCTION TO YEAR 0.9					
A-1	0.73	25.5	5329	50.4	0.66
B-1	1.19	41.6	7389	35.6	0.71
C-1	0.01	0.3	4423	55.5	0.46
C-2	0.13	4.5	5299	49.4	0.55
D-1	0.80	28.1	9303	23.5	0.33
Total	2.86	100.0			
Average			7298	36.5	0.58
YEAR 0.9 TO 2.1					
A-1	0.92	16.1	5228	51.1	0.68
B-1	1.97	34.6	7775	32.8	0.61
C-1	0.32	5.6	5890	42.3	0.45
C-2	1.08	18.9	6199	42.8	0.44
D-1	1.41	24.8	9282	23.7	0.32
Total	5.70	100.0			
Average			7333	36.3	0.51
YEAR 2.1 TO 3.0					
A-1	1.58	19.0	5253	50.9	0.67
B-1	2.75	33.1	7735	33.1	0.60
C-1	0.34	4.1	5882	42.4	0.44
C-2	1.38	16.6	6426	41.1	0.41
D-1	2.26	27.2	9300	23.6	0.31
Total	8.31	100.0			
Average			7398	35.8	0.50
YEAR 3.0 TO 4.1					
A-1	1.37	12.0	5292	50.6	0.68
B-1	2.55	22.3	7519	34.6	0.65
C-1	0.57	5.0	5164	48.8	0.50
C-2	3.38	29.7	6275	42.2	0.42
D-1	3.53	31.0	9304	23.5	0.30
Total	11.40	100.0			
Average			7316	36.4	0.47
YEAR 4.1 TO 5.7					
A-1	5.53	30.8	5482	49.3	0.74
B-1	3.02	16.8	7002	38.4	0.70
C-1	0.31	1.7	5278	47.8	0.48
C-2	1.87	10.4	6118	43.4	0.43
D-1	7.25	40.3	9230	24.0	0.33
Total	17.98	100.0			
Average			7312	36.4	0.53

(continued)

TABLE 5-4 (Continued)

COAL ZONE	COAL QUANTITY (calorific value > 4000 Btu/lb) tonnes x 10^6	% of zone	COAL QUALITY Calorific Value (Btu/lb)	Ash Content (%)	Sulphur Content (%)
CUMULATIVE -					
PRE-PRODUCTION TO YEAR 5.7					
A-1	10.32	21.8	5384	50.0	0.71
B-1	11.79	25.0	7455	35.1	0.66
C-1	1.53	3.2	5485	45.9	0.55
C-2	7.94	16.8	6218	42.6	0.43
D-1	15.71	33.2	9263	23.8	0.32
Total	47.29	100.0			
Average			7332	36.3	0.51
YEAR 5.7 TO 9.8					
A-1	10.33	21.9	5418	49.7	0.72
B-1	9.64	20.4	7670	33.5	0.62
C-1	3.22	6.8	6100	40.4	0.46
C-2	9.82	20.8	6662	39.4	0.43
D-1	14.24	30.1	9240	24.0	0.31
Total	47.25	100.0			
Average			7335	36.3	0.50
YEAR 9.8 TO 15.5					
A-1	14.21	22.0	5522	49.0	0.72
B-1	9.13	14.2	6806	39.8	0.67
C-1	1.32	2.1	5099	49.4	0.54
C-2	8.90	13.8	5785	45.8	0.40
D-1	30.91	47.9	8818	26.8	0.31
Total	64.47	100.0			
Average			7312	36.4	0.47
YEAR 15.5 TO 21.1					
A-1	18.73	31.9	5458	49.4	0.70
B-1	9.09	15.5	7519	34.6	0.66
C-1	1.70	2.9	5112	49.3	0.47
C-2	6.73	11.5	6416	41.2	0.43
D-1	22.43	38.2	9049	25.2	0.33
Total	58.68	100.0			
Average			7282	36.6	0.52
YEAR 21.1 TO 26.1					
A-1	11.55	22.5	5522	49.0	0.69
B-1	6.20	12.1	6715	40.4	0.67
C-1	0.69	1.3	4733	52.7	0.49
C-2	5.53	10.8	6221	42.6	0.45
D-1	27.35	53.3	8697	27.6	0.32
Total	51.32	100.0			
Average			7424	35.7	0.46
YEAR 26.1 TO 35					
A-1	13.47	16.8	5507	49.1	0.66
B-1	12.24	15.2	6828	39.6	0.68
C-1	0.86	1.1	4077	58.6	0.53
C-2	13.07	16.2	5217	50.0	0.38
D-1	40.84	50.7	8823	26.7	0.27
Total	80.48	100.0			
Average			7328	36.3	0.42
CUMULATIVE					
PRE-PRODUCTION TO YEAR 35					
A-1	78.61	22.5	5473	49.3	0.70
B-1	58.09	16.6	7188	37.0	0.66
C-1	9.32	2.7	5390	46.8	0.48
C-2	51.99	14.9	6002	44.2	0.41
D-1	151.48	43.3	8918	26.1	0.31
Total	349.49	100.0			
Average			7327	36.3	0.48

ANNUAL PRODUCTION STATISTICS

MATERIALS MINED QUANTITIES IN MILLIONS (10^6)	PRE-PRODUCTION YEARS -4	-3	-2	-1	PRODUCTION YEARS 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COAL DELIVERED TO GENERATING STATION (tonnes)				1·03	3·08	5·43	8·20	10·66	11·30	11·32	11·36	11·36	11·36	11·36	11·40	11·40	11·40	11·40	11·40	10·92	10·68
COAL																					
Fuel above cut-off (bank cubic metres)				0·70	2·07	3·64	5·50	7·14	7·57	7·59	7·62	7·62	7·62	7·62	7·65	7·65	7·65	7·65	7·65	7·31	7·15
Low grade coal (bank cubic metres)				0·01	0·12	0·14	0·13	0·45	0·65	0·49	0·15	0·15	0·15	0·20	0·42	0·42	0·41	0·41	0·41	0·25	0·09
Sub-total coal (bank cubic metres)				0·71	2·19	3·78	5·63	7·59	8·22	8·08	7·77	7·77	7·77	7·82	8·07	8·07	8·06	8·06	8·06	7·56	7·24
WASTE																					
Waste above bedrock (bank cubic metres)		2·05	6·55	11·45	12·59	12·93	10·88	10·02	9·10	9·60	10·57	10·57	10·57	9·74	7·97	7·97	7·97	7·97	7·97	9·14	10·32
Bedrock waste (bank cubic metres)				0·30	0·70	0·63	0·94	1·41	2·76	2·66	2·40	2·40	2·40	2·77	4·27	4·27	4·27	4·27	4·27	5·52	6·72
Sub-total waste (bank cubic metres)		2·05	6·55	11·75	13·29	13·56	11·82	11·43	11·86	12·26	12·97	12·97	12·97	12·51	12·24	12·24	12·24	12·24	12·24	14·66	17·04
TOTAL MATERIALS MINED (bank cubic metres)		2·05	6·55	12·46	15·48	17·34	17·45	19·02	20·08	20·34	20·74	20·74	20·74	20·33	20·31	20·31	20·30	20·30	20·30	22·22	24·28
STRIP RATIO (bank cubic metres waste plus low grade coal per tonne of coal delivered)				11·4	4·3	2·5	1·4	1·1	1·0	1·1	1·1	1·1	1·1	1·1	1·1	1·1	1·1	1·1	1·1	1·3	1·6

MATERIALS MINED QUANTITIES IN MILLIONS (10^6)	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	TOTAL
COAL DELIVERED TO GENERATING STATION (tonnes)	10·68	10·68	10·68	10·68	10·41	10·41	10·41	10·41	8·89	9·02	9·02	9·02	9·02	9·02	9·02	9·02	9·02	9·02	349·49
COAL																			
Fuel above cut-off (bank cubic metres)	7·15	7·15	7·15	7·15	6·97	6·97	6·97	6·97	5·97	6·06	6·06	6·05	6·05	6·05	6·05	6·05	6·05	6·05	234·32
Low grade coal (bank cubic metres)	0·09	0·09	0·09	0·09	0·20	0·22	0·22	0·22	0·22	0·27	0·27	0·27	0·27	0·27	0·28	0·28	0·28	0·28	8·96
Sub-total coal (bank cubic metres)	7·24	7·24	7·24	7·24	7·17	7·19	7·19	7·19	6·19	6·33	6·33	6·32	6·32	6·32	6·33	6·33	6·33	6·33	243·28
WASTE																			
Waste above bedrock (bank cubic metres)	10·32	10·32	10·31	10·31	9·98	8·88	8·88	8·88	7·98	1·50	1·07	1·07	1·07	1·07	1·07	1·07	1·07	1·07	281·85
Bedrock waste (bank cubic metres)	6·72	6·71	6·71	6·71	7·55	7·87	7·87	7·87	7·87	5·06	4·66	4·67	4·67	4·66	4·65	4·65	4·65	4·64	161·15
Sub-total waste (bank cubic metres)	17·04	17·03	17·02	17·02	17·53	16·75	16·75	16·75	15·85	6·56	5·73	5·74	5·74	5·73	5·72	5·72	5·72	5·71	443·00
TOTAL MATERIALS MINED (bank cubic metres)	24·28	24·27	24·26	24·26	24·70	23·94	23·94	23·94	22·04	12·89	12·06	12·06	12·06	12·05	12·05	12·05	12·05	12·04	686·28
STRIP RATIO (bank cubic metres waste plus low grade coal per tonne of coal delivered)	1·6	1·6	1·6	1·6	1·7	1·6	1·6	1·6	1·8	0·7	0·2	0·2	0·2	0·2	0·2	0·2	0·2	0·2	1·3

FUEL QUALITY	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
COAL DELIVERED TO GENERATING STATION																					
Btu / lb (dry basis)				7484	7301	7333	7397	7319	7312	7312	7335	7335	7335	7331	7312	7312	7312	7312	7312	7284	7250
MJ / kg (dry basis)				17·4	17·0	17·1	17·2	17·0	17·0	17·0	17·1	17·1	17·1	17·1	17·0	17·0	17·0	17·0	17·0	16·9	16·9
Ash content (%) (dry basis)				35·2	36·5	36·3	35·8	36·4	36·4	36·4	36·3	36·3	36·3	36·3	36·4	36·4	36·4	36·4	36·4	36·6	36·9
Sulphur content (%) (dry basis)				0·53	0·58	0·51	0·50	0·47	0·53	0·52	0·50	0·50	0·50	0·49	0·47	0·47	0·47	0·47	0·47	0·49	0·51
Sulphur (lb / million btu's)				0·70	0·79	0·69	0·67	0·64	0·72	0·71	0·68	0·68	0·68	0·67	0·64	0·64	0·64	0·64	0·64	0·67	0·71
HEAT UNITS DELIVERED																					
Btu x 10^{13}				1·3	3·7	6·6	10·0	12·9	13·7	13·7	13·8	13·8	13·8	13·8	13·8	13·8	13·9	13·8	13·8	13·2	12·8
MJ x 10^{10}				1·3	3·9	6·9	10·6	13·6	14·4	14·4	14·5	14·5	14·5	14·5	14·5	14·5	14·5	14·5	14·5	13·9	13·5
(based on 25% delivered moisture and coal cut-off at 4000 btu / lb or 9·3 MJ / kg)																					

FUEL QUALITY	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	AVERAGE
COAL DELIVERED TO GENERATING STATION																			
Btu / lb (dry basis)	7250	7250	7250	7250	7417	7424	7424	7424	7424	7341	7328	7328	7328	7328	7328	7328	7328	7328	7327
MJ / kg (dry basis)	16·9	16·9	16·9	16·9	17·3	17·3	17·3	17·3	17·3	17·1	17·0	17·0	17·0	17·0	17·0	17·0	17·0	17·0	17·0
Ash content (%) (dry basis)	36·9	36·9	36·9	36·9	35·7	35·7	35·7	35·7	35·7	36·2	36·3	36·3	36·3	36·3	36·3	36·3	36·3	36·3	36·3
Sulphur content (%) (dry basis)	0·51	0·51	0·51	0·51	0·46	0·46	0·46	0·46	0·46	0·43	0·42	0·42	0·42	0·42	0·42	0·42	0·42	0·42	0·48
Sulphur (lb / million btu's)	0·71	0·71	0·71	0·71	0·62	0·62	0·62	0·62	0·62	0·58	0·57	0·57	0·57	0·57	0·57	0·57	0·57	0·57	0·65
HEAT UNITS DELIVERED																			TOTAL
Btu x 10^{13}	12·8	12·8	12·8	12·8	12·8	12·8	12·8	12·6	10·9	10·9	10·9	10·9	10·9	10·9	10·9	10·9	10·9	10·9	423·43
MJ x 10^{10}	13·5	13·5	13·5	13·5	13·5	13·5	13·5	13·5	11·5	11·5	11·5	11·5	11·5	11·5	11·5	11·5	11·5	11·5	446·72
(based on 25% delivered moisture and coal cut-off at 4000 btu / lb or 9·3 MJ / kg)																			

5.7 MATERIALS MOVEMENT PLANNING

571 INTRODUCTION

This section of the report investigates the delivery of non-fuel materials from the mine to stockpiles, waste dumps, and earthworks construction sites. The production statistics on Table 5-5 provide quantities for three classifications of non-fuel products: low-grade coal, wastes above bedrock, and bedrock wastes. Low-grade coal will be stockpiled as discussed in Section 572 below, and is not included in other waste disposal considerations. A further breakdown of the production schedule waste materials is given in Section 573, which identifies specialized materials to be directed to dumps or specific construction sites. The development of the Houth Meadows and Medicine Creek dumps is discussed in some detail in Section 574 with regard to approach, ultimate capacity, and staged planning over the life of the project. Section 585 discusses the delivery of construction-grade materials to selected sites, and the scheduling of all waste materials movement.

572 LOW-GRADE COAL STOCKPILE

It is estimated that 8.96×10^6 bank cubic metres of low-grade coal will be excavated during the 35-year life of the project. To allow for swell factors and machine compaction, an ultimate stockpile capacity of 10.75×10^6 cubic metres has been projected. Two locations for the stockpile were originally considered, one on the mountain between the Medicine Creek dump and the generating station, the other at the toe of the Houth Meadows embankment between the water treatment facility and the conveyor right-of-way, as shown in Figure 5-9, later in this section. The latter site chosen is more accessible, is more protected from wind, is close to an established water treatment facility, and is located in an area which would of necessity, be environmentally disturbed. The surface area of valley floor that would be taken up by the stockpile is 17.2 hectares and the final elevation at Year 35 would be 910-metres ASL, with side slopes of 2.5 to 1.

Stockpile design includes the construction of a rigid foundation of impervious compacted material, sloping towards the water treatment facility. The lowest point on the foundation surface is at 840-metres ASL.

To minimize the possibility of spontaneous combustion provision has been made for crushing the low-grade coal, placing and compacting it in relatively thin layers. To minimize the extent of disturbed areas and to allow for early reclamation, the stockpile is divided into four areas of equal size and the coal would be deposited in only one of these areas at a time, until an elevation of 895 metres ASL has been reached. At this level, the compacted surface would be temporarily reclaimed and deposition of low-grade material would then be directed to the next of the four areas. After all four areas have been built up, a final lift will be deposited over the entire stockpile to bring its final elevation up to 910 metres ASL. Permanent reclamation at this final level would then be carried out.

573 WASTE MATERIALS BREAKDOWN

The two broad categories of waste materials, wastes above bedrock and bedrock wastes, result in a total quantity of 443 x 10^6 bank cubic metres which require disposal. In order to determine final disposal requirements, the "wastes above bedrock" have been further categorized from the MEPS computer program results and from surficial geology bench plans as follows:

Topsoil - material of suitable quality for plant growth which should be stockpiled for future use in reclamation

Pervious Materials - are sands and gravels identified in this report as glaciofluvial, colluvial, and alluvial deposits, which should be used for several construction purposes

Baked Zone Materials - should be used in road construction

Hardpan and Consolidated Tills - will require blasting or dozer ripping during mining operations and should be disposed of in waste dumps

Impervious Wastes - this category comprises the balance of all "wastes above bedrock" not included above. A certain portion of these materials will be used for buffer zones and other specific construction requirements, but the majority contains slide debris and should be disposed of in the waste dumps.

No sub-divisions have been made of the "bedrock wastes" and it has been assumed that all such materials will be dumped as general mine waste in areas behind the retaining embankments at the dumps.

The quantities of material contained in each of the above categories, as excavated from the pit, including bedrock wastes, are estimated and summarized by year in Table 5-6.

574 DUMP DEVELOPMENT

574.1 General

The production schedule indicates that 443 x 10^6 bank cubic metres of waste will be directed to dumps over the life of the mine. Two main dumping areas have been identified and estimated to have adequate capacity and access for the life of the mine. Houth Meadows has a final projected capacity of 449 x 10^6 bank cubic metres, while the Medicine Creek dump is projected to hold 495 x 10^6 bank cubic metres. Both figures are based on the material swell factors used throughout this study. The layouts of the dumps are shown on plans of the overall mine and its facilities at Years -1, 5, 15 and 35, Figures 5-6, 5-7, 5-8 and 5-9, respectively.

The approach to dump development was threefold:

(a) Although either of the areas may be large enough to contain all of the mine waste over 35 years, it was considered that due to the uncertainty of the swell factors and stability characteristics being used, both dumps should be developed during the 35-year life of the mine. However, it was decided to defer development of the Medicine Creek dump, which is the least economic of the two, to approximately Year 15, thus permitting accurate swell factors to be determined through actual mining experience. This deferral would also result in the costs of developing the second dump having little effect on the overall cost estimates for the feasibility study.

TABLE 5-6

Estimated Volumes of Waste Materials
(in BCM x 10^6)
Hat Creek Project Mining Feasibility Report 1978

	WASTES ABOVE BEDROCK						BEDROCK WASTES	TOTAL
Year	Top Soil	Pervious Materials	Baked Zone	Hardpan & Consolidated Tills	Impervious Materials	Sub Total	Impervious Materials	COMBINED WASTES
-3	0.26	1.79	-	-	-	2.05	-	2.05
-2	0.04	4.61	-	-	1.90	6.55	-	6.55
-1	0.04	5.30	0.58	-	5.53	11.45	0.30	11.75
1	0.04	8.30	1.21	-	3.04	12.59	0.70	13.29
2	0.04	8.30	0.47	-	4.12	12.93	0.63	13.56
3	0.04	8.30	0.55	-	1.99	10.88	0.94	11.82
4	0.04	8.30	0.87	-	0.81	10.02	1.41	11.43
5	0.04	8.30	0.58	-	0.18	9.10	2.76	11.86
6	0.04	7.85	0.33	0.74	0.64	9.60	2.66	12.26
7	0.04	7.85	0.33	0.74	1.61	10.57	2.40	12.97
8	0.04	7.85	0.33	0.74	1.61	10.57	2.40	12.97
9	0.04	7.85	0.33	0.74	1.61	10.57	2.40	12.97
10	0.04	6.24	0.24	0.74	2.48	9.74	2.77	12.51
11	0.04	4.74	0.24	0.74	2.21	7.97	4.27	12.24
12	0.04	4.74	0.24	0.74	2.21	7.97	4.27	12.24
13	0.04	4.74	0.24	0.74	2.21	7.97	4.27	12.24
14	0.04	4.74	0.24	0.74	2.21	7.97	4.27	12.24
15	0.04	4.74	0.24	0.74	2.21	7.97	4.27	12.24
16	0.04	7.48	0.24	0.74	0.64	9.14	5.52	14.66
17	0.04	7.48	0.24	0.74	1.82	10.32	6.72	17.04
18	0.04	7.48	0.24	0.74	1.82	10.32	6.72	17.04
19	0.04	7.48	0.24	0.74	1.82	10.32	6.71	17.03
20	0.04	7.48	0.24	0.74	1.81	10.31	6.71	17.02
21	0.03	7.48	0.13	0.74	1.91	10.31	6.71	17.02
22	0.03	4.72	0.11	0.73	4.39	9.98	7.55	17.53
23	0.03	4.72	0.11	0.73	3.29	8.88	7.87	16.75
24	0.03	4.72	0.11	0.73	3.29	8.88	7.87	16.75
25	0.03	4.71	0.12	0.73	3.29	8.88	7.87	16.75
26	0.03	4.71	0.11	0.73	2.40	7.98	7.87	15.85
27	0.03	-	0.11	0.73	0.63	1.50	5.06	6.56
28	0.03	-	0.11	0.73	0.20	1.07	4.66	5.73
29	0.03	-	0.12	0.73	0.19	1.07	4.67	5.74
30	0.03	-	0.12	0.73	0.19	1.07	4.67	5.74
31	-	-	-	0.73	0.34	1.07	4.66	5.73
32	-	-	-	0.73	0.34	1.07	4.65	5.72
33	-	-	-	0.73	0.34	1.07	4.65	5.72
34	-	-	-	0.70	0.37	1.07	4.65	5.72
35	-	-	-	0.70	0.37	1.07	4.64	5.71
TOTALS	1.44	183.00	9.37	22.00	66.04	281.85	161.15	443.00

cominco-monenco joint venture

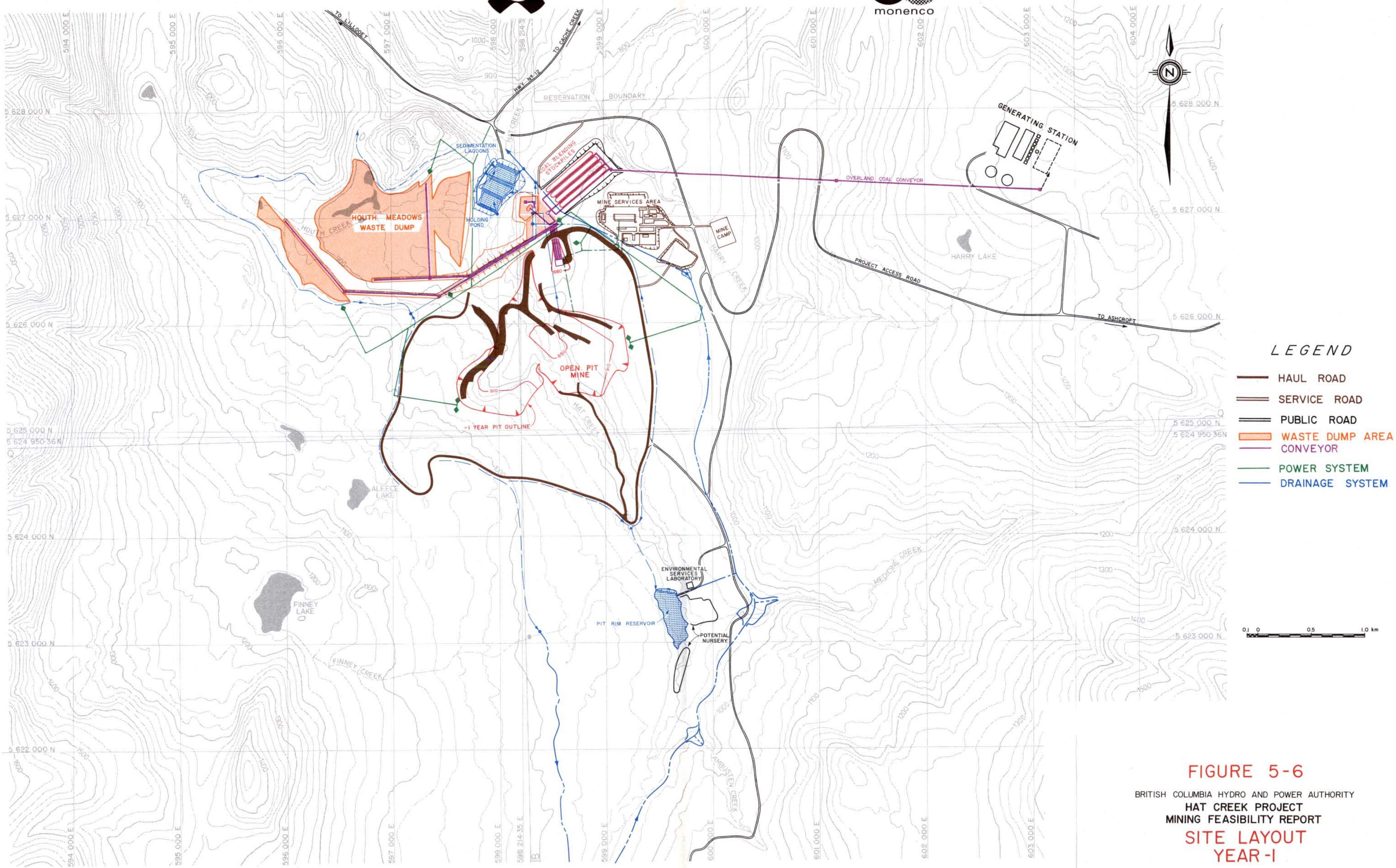


FIGURE 5-6

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

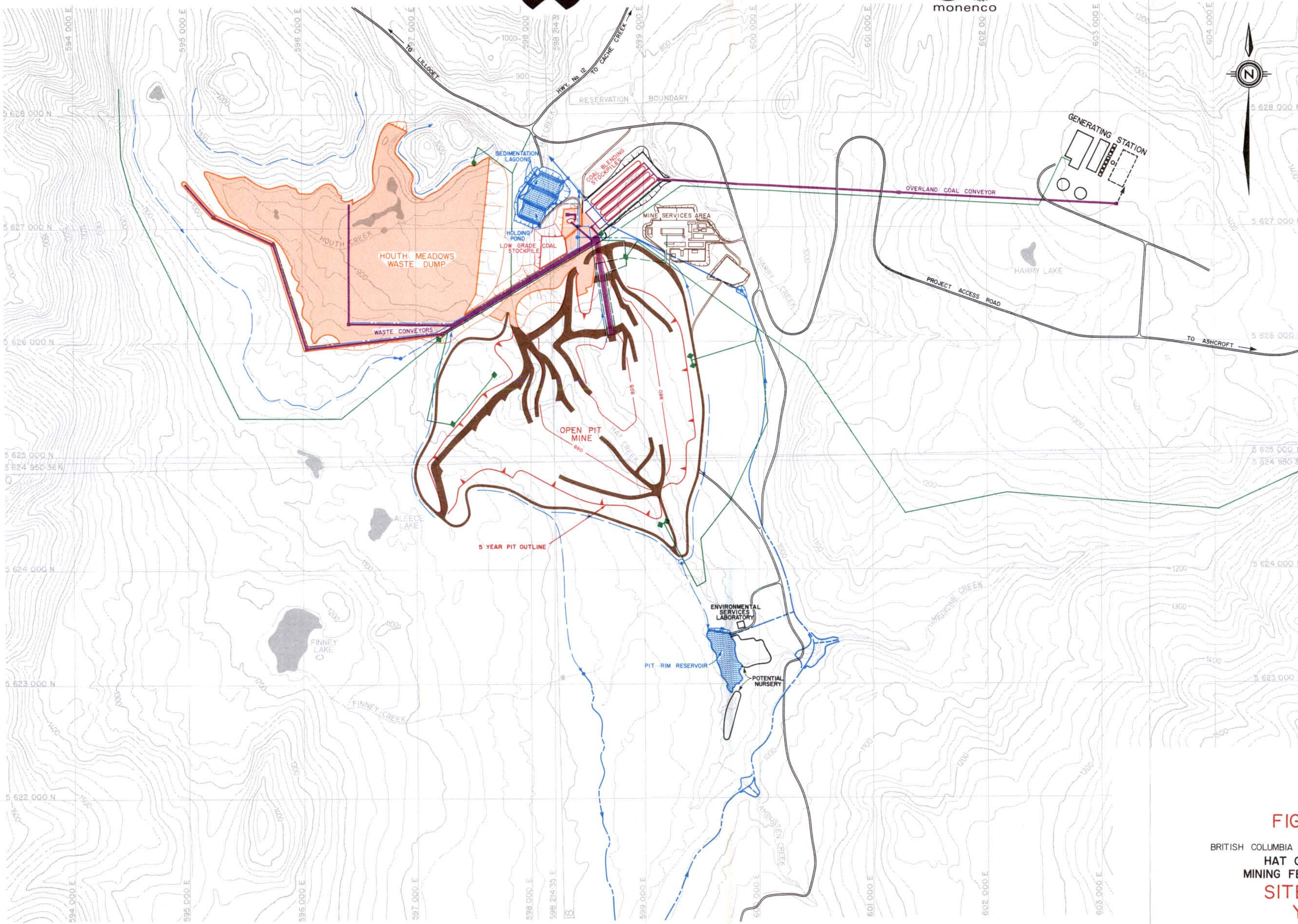
HAT CREEK PROJECT

MINING FEASIBILITY REPORT

SITE LAYOUT

YEAR -1

cominco-monenco joint venture



LEGEND

- HAUL ROAD
- SERVICE ROAD
- PUBLIC ROAD
- WASTE DUMP AREA
- CONVEYOR
- POWER SYSTEM
- DRAINAGE SYSTEM

0.1 0 0.5 1.0 km

FIGURE 5-7

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT
SITE LAYOUT
YEAR 5

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FIGURE 5-8

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

SITE LAYOUT
YEAR 15

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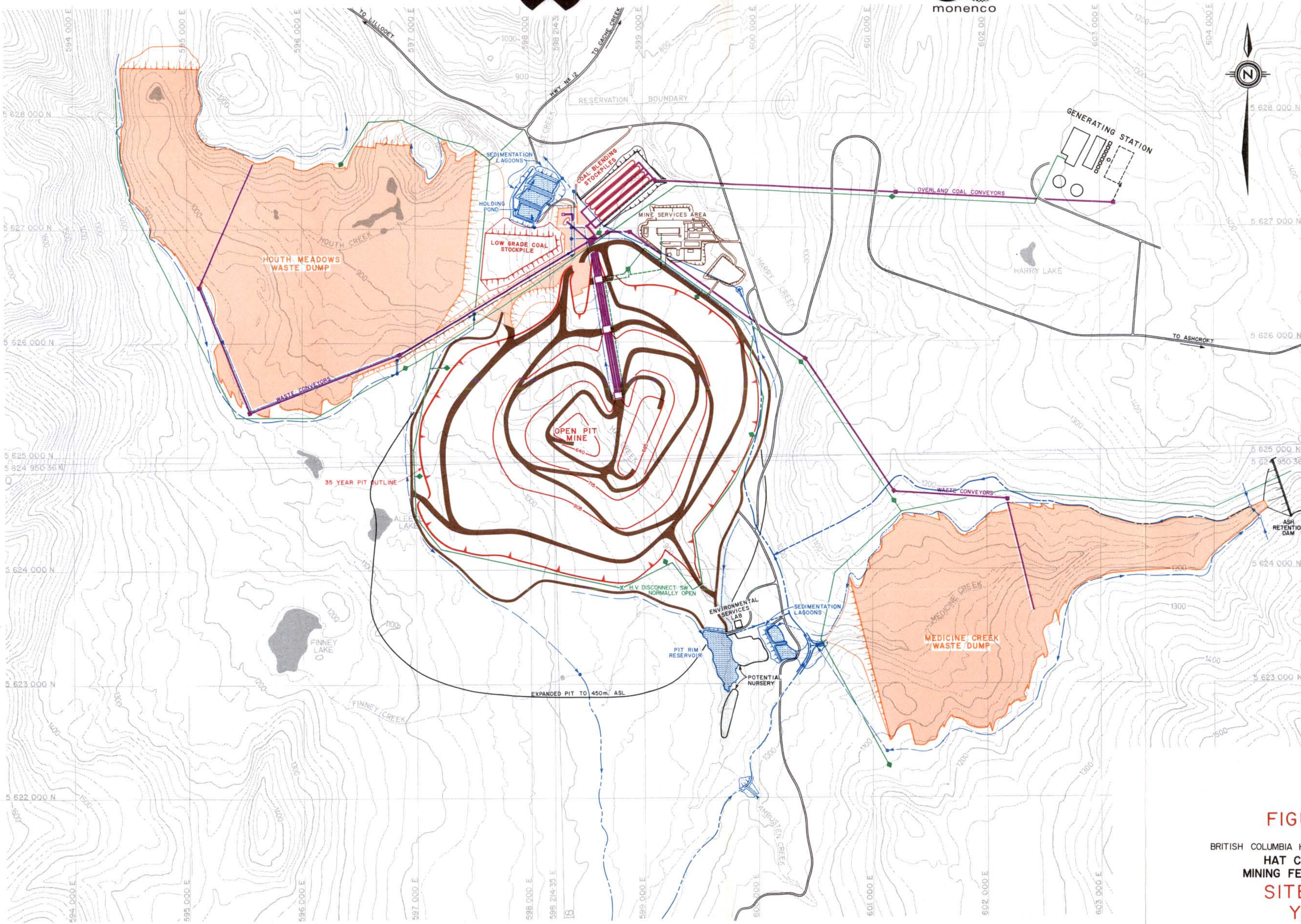


LEGEND

- HAUL ROAD
- SERVICE ROAD
- PUBLIC ROAD
- WASTE DUMP AREA
- CONVEYOR
- POWER SYSTEM
- DRAINAGE SYSTEM

FIGURE 5-9

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT
SITE LAYOUT
YEAR 35

(b) Because of the northern conveyor access, the development of Houth Meadows dump is preferred. Also, since mining beyond Year 35 would probably be carried out south of the current project area, it would be advantageous to reserve more southerly dumping sites including some capacity in the Medicine Creek dump for these future operations.

(c) A review of the availability of pervious materials beyond Year 35 indicates possible shortages of suitable construction materials. It would therefore be beneficial to future operations if the materials currently available were utilized for embankment construction during the initial 35 years of the life of the No. 1 Deposit. Thus both embankments would be constructed with a view to eventual complete utilization. These portions constructed during the 35-year project will be the lower increment of a final design that is considered large enough to retain spoil equal to the maximum capacity of each valley.

A conveyor system is used to transport wastes from the mine and deposit them in 35-metre benches within the dumps. This system is described in detail in Section 6.

Following are the recommended lift heights and other limitations for depositing weak mine waste materials by spreaders based on the geotechnical criteria given:

(a) Based on the 1.0 safety factor, the spreader will deposit a 20-metre high lift below itself. The spreader will operate at a minimum edge safety distance of 4 m or one-fifth of the lift height.

(b) In the anticipated normal mode of operation, the shiftable belt conveyor will be moved periodically toward the crest of a newly-constructed 20-metre high waste life. This would occur approximately one or two months after the lift was deposited thus allowing some time for the materials to stabilize.

Under these conditions the recommended minimum distance between the crest and the conveyor is equal to one-half of the lift height or 10 metres. Ongoing slope monitoring will be required to detect abnormal differential settling before the conveyor is moved.

(c) An additional 15-metre lift will be deposited by the spreader on the operating bench of the conveyor, behind the conveyor itself. The total bench height is therefore 35 metres for each pass of the shiftable conveyor.

Figure 5-10 shows the development of the dumps for Years -1, 5, 10, 15, 16 and 35, based on layout plans as discussed below in Sections 574.2 and 574.3. Areas of the dumps which are at their final level for relcamation are indicated. The positioning of conveyors at key points of dump development is shown, indicating the various equipment and site preparations required.

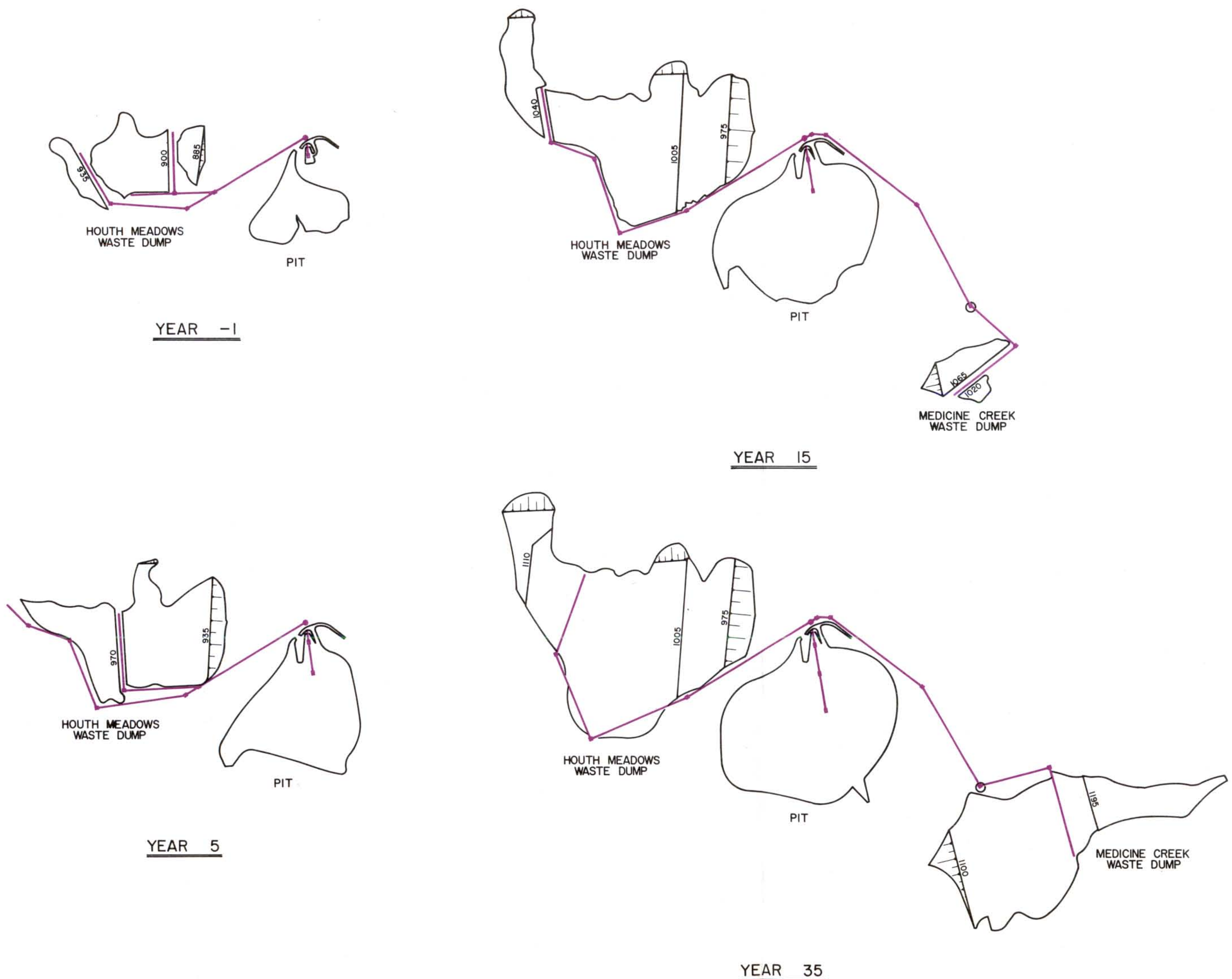
The cost estimates for the dumps were based on the development scheme described above. A more specific discussion of each dump is presented in the following sections.

574.2 Houth Meadows Waste Dump

By Year 35, the quantity of materials placed on the Houth Meadows waste dump, including embankments, is estimated at 291.56 x 10^6 bank cubic metres or 65% of its maximum capacity of 449 x 10^6 bank cubic metres. A set of plans was prepared to show the incremental dump layout for Years-1, 1, 3, 5, 10, 15, 25, and 35. In addition, a detailed layout of the dump at Year 35 as shown on Figure 5-11 indicates the final contoured slopes for reclamation, configurations of the retaining embankments, a cross-section of the build-up by lifts and the ultimate dump boundary.

In planning of the final dump configuration, incremental development over the 35-year period was studied in detail to recommend the conveyor layout required for the lifts totalling 35 metres, and to determine the annual movement of materials necessary to ensure maximum utilization of construction materials. Foundation conditions and dump design features incorporated various geotechnical and environmental considerations, including removal and stockpiling of suitable topsoil for use in future reclamation.

Three retaining embankments, as shown on Figure 5-10 and referred to as HME-1, HME-2, and HME-3, should be constructed at Houth Meadows. They require 35.1 x 10^6, 3.5 x 10^6, and 4.0 x 10^6 respectively for a total of 42.6 x 10^6 bank cubic metres of material by the end of Year 35. This volume represents about 67%



LEGEND

- CONVEYOR
- CENTRAL DISTRIBUTION POINT
- TRUCK UNLOADING STATION
- TRANSFER POINT
- 1005 ELEVATION

FIGURE 5-10

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

WASTE DUMP DEVELOPMENT

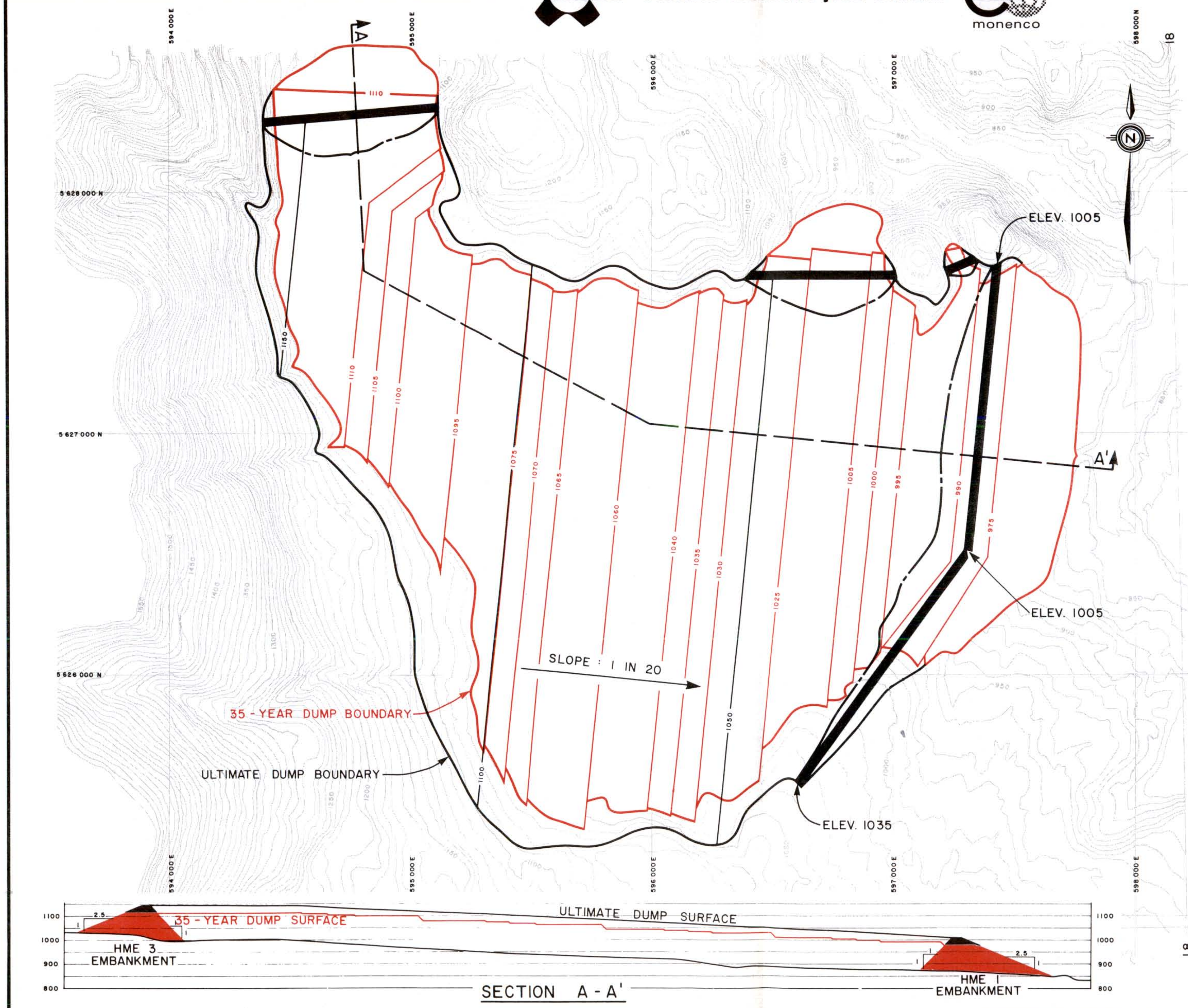


FIGURE 5-11

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

HOUTH MEADOWS WASTE DUMP

of the maximum design capacity of these embankments. The embankments should be founded on stable materials as mapped by the geotechnical consultant. As shown on Figure 5-11, the centre-line of HME-1 changes direction near its mid-point to avoid an identified area of soft surficial materials.

Leachate from the toe of HME-1 should collect and drain into a water treatment pond. No special drain beds were incorporated into the design of the embankment as the materials used for its construction are considered to have a permeability of 10^{-4} cm/sec or better; the estimated amount of leachate flow ranges from 300 to 1500 cubic metres per day. Embankments HME-2 and HME-3 are designed to be impermeable and are therefore assumed to be leachate-free.

574.3 Medicine Creek Waste Dump

The final projected capacity of the Medicine Creek dump is 495×10^6 bank cubic metres at elevation 1280 metres ASL, of which 28% or 139.55×10^6 bank cubic metres should be utilized by the end of Year 35. A set of layout plans showing incremental development of the dump at Years 16, 17, 25 and 35, although not included within this report, were prepared. A detailed layout for the Medicine Creek waste dump showing information similar to that described earlier for Houth Meadows was prepared and is shown on Figure 5-12.

The same geotechnical considerations that applied to Houth Meadows also pertain to Medicine Creek. Site conditions are generally similar, with the exception that no areas of soft surficials or near-surface springs have been identified.

Only one embankment should be constructed at Medicine Creek, its 35-year volume reaching approximately 26.23×10^6 bank cubic metres or 43% of its maximum design capacity.

575 WASTE MATERIALS DELIVERY SCHEDULE

Table 5-7 estimates the annual breakdown over the life of the project of the volume of waste materials removed from the mine and the mode of transport used to deliver each type of material to its final destination. Outside the mine approximately 93% of waste materials are transported exclusively by conveyor. Table 5-8 outlines the total volumes of waste materials transported to and placed in the Houth Meadows and Medicine Creek waste dumps.

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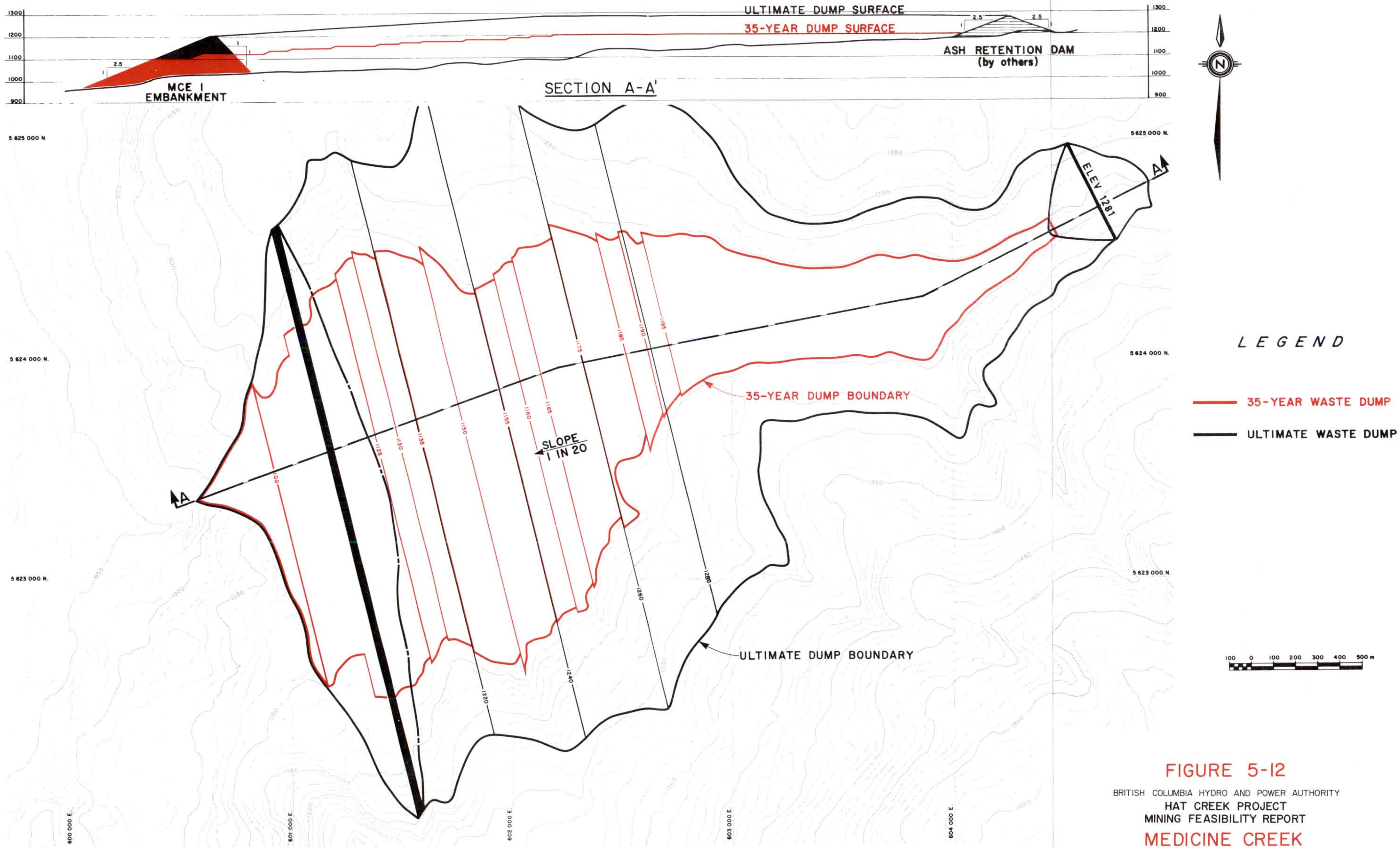

FIGURE 5-12

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

MEDICINE CREEK
WASTE DUMP

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TRANSPORTATION AND DESTINATION SCHEDULE
FOR WASTE MATERIALS

TABLE 5-7

DESCRIPTION	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																																			
	-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	TOTAL
PERVIOUS MATERIALS FOR CONSTRUCTION																																										
1. TRANSPORTED BY TRUCK																																										
MAIN CONVEYOR R/W & HAT CREEK VALLEY FILL				1.79	1.95	0.26		1.50	0.50	0.50	0.40																															6.90
ROAD CONST., SITE PREP., SOIL REMOVAL				0.26	0.80	0.07	0.33	0.23	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	4.99
CONVEYOR R/W CONSTRUCTION (MH)					0.10																																					0.10
EMBANKMENT CONSTRUCTION (HME-1)					1.50																																					1.50
TOTAL TRUCKED				2.05	4.35	0.33	0.33	1.73	0.60	0.60	0.50	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	13.49
2. TRANSPORTED BY CONVEYOR																																										
EMBANKMENT CONSTRUCTION (HME-1)						2.35	5.66	3.00		5.53	6.27	2.00			4.00	4.50																										33.31
EMBANKMENT CONSTRUCTION (HME-2)																													1.50													1.50
EMBANKMENT CONSTRUCTION (HME-3)																	0.30	0.30												1.70							1.00					3.30
EMBANKMENT CONSTRUCTION (MCE)																						4.50	5.10	6.50	6.00	2.00	6.00	1.15														31.25
TOTAL CONVEYED						2.35	5.66	3.00		5.53	6.27	2.00			4.00	4.50	0.30	0.30				4.50	5.10	6.50	6.00	2.00	6.00	1.15	1.50	1.70							1.00					69.36
3. TRANSPORTED BY CONVEYOR AND TRUCK																																										
CONVEYOR R/W CONSTRUCTION (HM)						0.20	0.20			0.15	0.15	0.10	0.10	0.10	0.10	0.10							0.10	0.10	0.10	0.10	0.10	0.10														1.80
EMBANKMENT CONSTRUCTION (HME-1)							0.09								0.10	0.10																										0.29
EMBANKMENT CONSTRUCTION (HME-2)						0.10	0.20						0.10							0.50	0.80								0.30													2.00
EMBANKMENT CONSTRUCTION (HME-3)																	0.20	0.20												0.30												0.70
GENERAL DUMP (HM)							0.03					0.10		0.10			0.10	0.10	0.30								0.40	0.20				0.20	0.20	0.20	0.20	0.20	0.30	0.30	0.30	0.20	0.20	3.63
GENERAL DUMP (MC)																						1.40	1.00					0.20	0.20	0.20	0.20	0.10	0.10	0.10	0.10	0.20						3.80
CONVEYOR R/W CONSTRUCTION (MC)																								0.50	0.50	0.50					0.30	0.20	0.20	0.20	0.20	0.10	0.20	0.20	0.20	0.30	0.30	3.90
EMBANKMENT CONSTRUCTION (MC)																						1.10	1.00																			2.10
TOTAL TRUCKED & CONVEYED						0.30	0.52			0.15	0.15	0.20	0.20	0.20	0.20	0.20	0.30	0.30	0.30	0.50	0.80	2.50	2.10	0.60	0.60	0.60	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	0.50	18.22
GENERAL MINE WASTE																																										
4. TRANSPORTED BY CONVEYOR																																										
HOUTH MEADOWS					2.20	8.77	6.78	8.83	11.22	5.15	4.94	9.95	12.67	12.67	8.67	7.71	11.54	11.54	11.84	11.64	11.34	3.56	4.34	3.84	3.33	3.72	6.12	6.28	3.65	3.75	8.45	7.45	4.16	3.33	3.34	3.24	2.53	3.72	3.72	3.72	3.71	243.43
MEDICINE CREEK																						4.00	5.40	6.00	7.00	10.60	4.30	9.50	11.00	10.70	7.70	7.80	1.80	1.80	1.80	1.90	1.60	1.40	1.40	1.40	1.40	98.50
TOTAL WASTE MATERIAL MOVED				2.05	6.55	11.75	13.29	13.56	11.82	11.43	11.86	12.26	12.97	12.97	12.97	12.51	12.24	12.24	12.24	12.24	12.24	14.66	17.04	17.04	17.03	17.02	17.02	17.53	16.75	16.75	16.75	15.85	6.56	5.73	5.74	5.74	5.73	5.72	5.72	5.72	5.71	443.00
COMPARISON OF REQUIRED AND AVAILABLE PERVIOUS MATL																																										
TOTAL PERVIOUS MATERIALS REQUIRED				2.05	4.35	2.98	6.51	4.73	0.60	6.28	6.92	2.30	0.30	0.30	4.30	4.80	0.70	0.70	0.40	0.60	0.90	7.10	7.30	7.20	6.70	2.70	6.60	1.75	2.10	2.30	0.60	0.60	0.60	0.60	0.60	0.60	1.60	0.60	0.60	0.60	0.60	101.07
TOTAL PERVIOUS MATERIALS AVAILABLE				2.05	4.61	5.30	8.30	8.30	8.30	8.30	8.30	7.85	7.85	7.85	7.85	6.24	4.74	4.74	4.74	4.74	4.74	7.48	7.48	7.48	7.48	7.48	7.48	4.72	4.72	4.72	4.71	4.71										183.00

TABLE 5-8

Estimated Waste Volumes
(including embankments)

(BCM x 10^6)

Hat Creek Project Mining Feasibility Report 1978

Pit Production Period		To Houth Meadows Dump	To Medicine Creek Dump	Civil Construction Requirements	Total Waste
Year	-3			2.05	2.05
	-2	3.80		2.75	6.55
	-1	11.42		0.33	11.75
	1	12.96		0.33	13.29
	2	11.83		1.73	13.56
	3	11.22		0.60	11.82
	4	10.83		0.60	11.43
	5	11.36		0.50	11.86
Years	6-10	63.18		0.50	63.68
	11-15	60.70		0.50	61.20
	16-20	19.19	63.10	0.50	82.79
	21-25	32.85	51.45	0.50	84.80
	26-35	42.22	25.00	1.00	68.22
Total		291.56	139.55	11.89	443.00

SECTION 6

RECOMMENDED MINING SYSTEM

6.1 INTRODUCTION

This section of the report describes in detail the recommended mining system: a combination of shovel, truck, and conveyor. Figure 6-1 illustrates the major components of the materials handling system; the overall system layout is shown on Figure 5-9 of Section 5.

All mine materials except surface soils and limited quantities of construction material are excavated by electric shovels with a 16.8 cubic metre bucket capacity. These load rear-dump, diesel-electric trucks of two size categories: 109-tonne for coal and 136-tonne for waste. After delivery to a truck unloading station for sizing by a primary crusher, materials (comprising coal, low-grade coal, construction grade waste, or general mine waste) will be transported via one of three in-pit conveyors northwards to a central distribution point.

Table 6-1 below outlines the destination of the mine materials and identifies the conveyors used beyond the central distribution point.

TABLE 6-1

Destination of Mine Materials

Mine Material	Destination	Conveyor
Coal	Generating Station	Coal transfer and overland conveyors
Low-grade coal	Low-grade stockpile	Low-grade transfer conveyor
Construction grade waste	Houth Meadows and Medicine Creek waste dumps	No. 1 or No. 2 waste conveyor
General mine waste	Houth Meadows and Medicine Creek waste dumps	No. 1 or No. 2 waste conveyor

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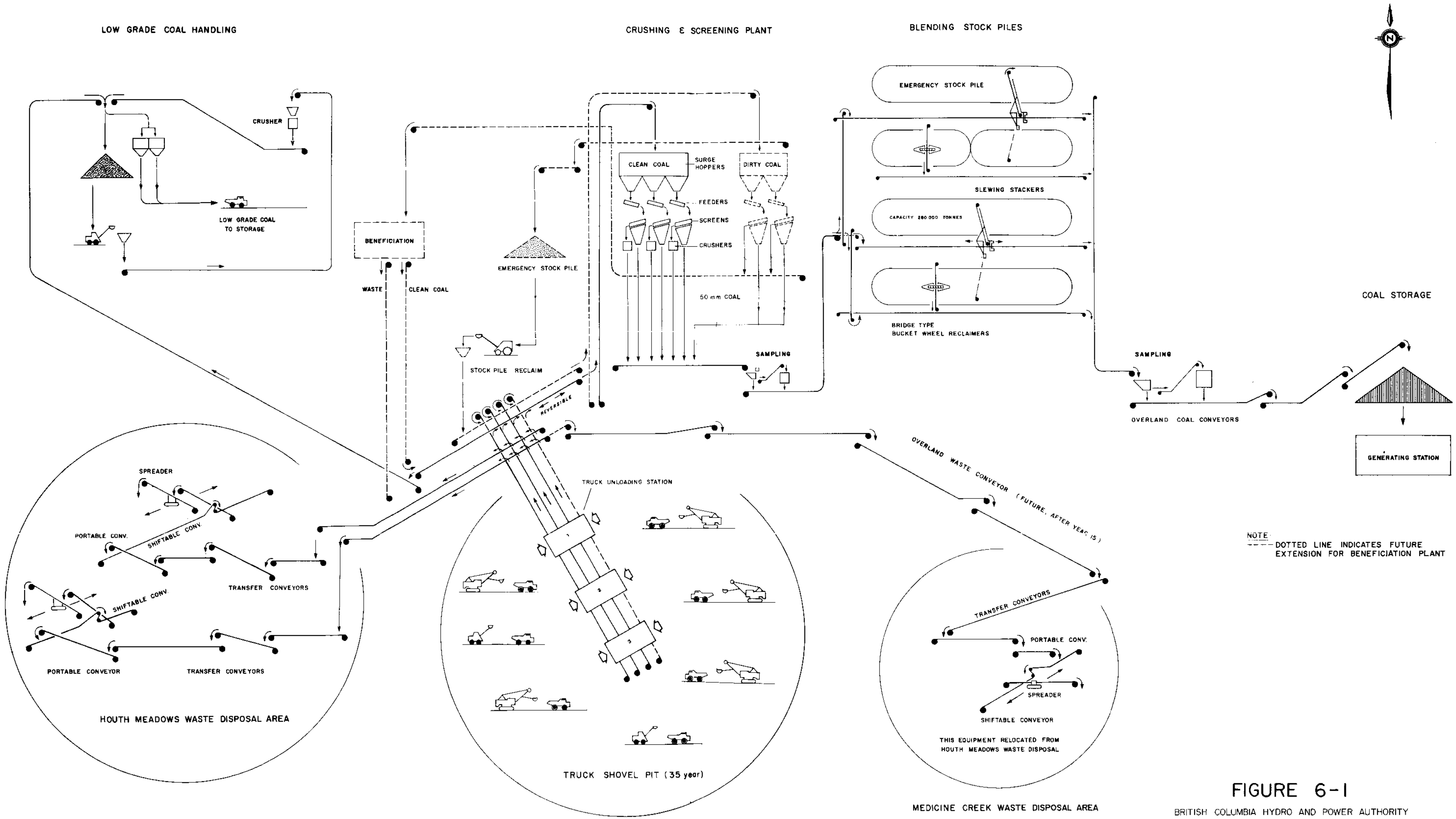


FIGURE 6-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

SYSTEM FLOW SHEET

Coal will be directed by a transfer conveyor to a crushing plant, where it is reduced in size from 300 mm to 50 mm, and placed in a stockpile for blending purposes. Delivery to the generating station would be completed with an overland coal conveyor. Low-grade coal will be conveyed to a small surge-pile after which it will be crushed, sized, and transported to a permanent stockpile via 32-tonne trucks. Waste materials will be conveyed directly to the required dump area from the central distribution point. Up to Year 15, it is proposed that all waste be directed to the Houth Meadows dump using two conveyors. By relocating one conveyor to the Medicine Creek disposal area, in Year 14, both waste dump areas would be utilized thereafter.

The principal units of equipment for the shovel/truck/conveyor system are outlined on Table 6-2. The list is applicable to Year 17, one of several peak production years. Additional mine fleet requirements have been included to accommodate periodic increases in generating station-coal production requirements. A capacity factor for the generating station of 85% for periods of time up to five months was used as the basis for fleet requirements.

TABLE 6-2

Summary of Mining Equipment
End of Year 17

Hat Creek Project Mining Feasibility Report 1978

Item	Number
Shovels 16.8 m^3 bucket capacity	7
Trucks 109-tonne	9
136-tonne	18
32-tonne	10
Scrapers 24 LCM	6
Graders	6
Dozers track	17
wheeled	2
Front-end loaders 11.5 m^3	2
5.4 m^3	3
1.5 m^3	3
Drills - Auger, Rotary, Rotary Percussion	3
Blasting Truck	1
Compactors	4
Gradall	1
Backhoe 1 m^3	1
Water Wagon	3
Mobile crusher	1
Mobile cranes 5 to 90 tonne	6
Mobile service vehicles	21
Light vehicles	130
Truck unloading stations	2
Crawler mounted waste spreaders	2
Rail mounted stackers	2
Bridge type bucketwheel reclaimers	2
	Length
Mine conveyors	2490 m
Coal transfer conveyors in preparation area	3290 m
Overland coal conveyors to generating plant	4000 m
Low-grade coal transfer conveyors	355 m
Waste conveyors	15 500 m

6.2 REASONS FOR RECOMMENDATION

The reasons for recommending the shovel/truck/conveyor system as the most suitable for the development of the Hat Creek deposit are summarized as follows:

(a) Of all systems studied it is the most capable of providing a reliable quantity and average quality of fuel from the No. 1 Deposit to the proposed generating station over a 35-year period of time.

(b) It offers the most favourable economics.

(c) The system utilizes techniques and equipment that are the most proven, dependable, and widely used by the mining industry in British Columbia.

(d) In accordance with mine development recommended, it is best adapted to utilization of the total coal resource, particularly with regard to future development beyond Year 35.

(e) The shovel/truck is the most adaptable of all excavating combinations to changes in mine planning.

(f) This system requires the least amount of staff and operator training to bring the mine into production.

(g) From the studies carried out to date the measures necessary for the protection of the environment and the reclamation of disturbed areas can be satisfactorily accommodated for this system.

6.3 SHOVELS

The principle excavator recommended for the mine operation is the 16.8 cubic metre electric shovel. A maximum of seven shovels will be required to meet production demands. This includes the possible increased coal requirements to meet up to an 85% capacity factor for the generating station.

The 11.5 cubic metre capacity shovel is the size now commonly found in mining operations throughout British Columbia. However, the 16.8 cubic metre capacity shovel was selected for the following reasons:

(a) The increased excavating power of the larger shovel is expected to reduce, from 100% to 45%, the quantity of coal that must be blasted.

(b) The average hourly excavating productivity in all materials is expected to be 240 bank cubic metres per hour greater with the larger shovel, resulting in a 10% improvement in the direct costs of excavation.

(c) The larger shovel is a better match with the 109-tonne and 136-tonne capacity trucks. Fewer dipper loads are required to load the truck with the larger shovel which increases the truck performance by reducing the fixed time for loading.

(d) The larger bucket capacity would not affect the opportunities for coal quality improvement by selective mining of partings.

Calculations to derive productivities, fleet sizes and operating costs were based on the following criteria.

Nominal bucket capacity	-	16.8 cubic metres
Average fill factor	-	0.8
90° swing cycle	-	36 seconds
Cycles per 50 minute hour	-	83
Scheduled hours	-	8760/year
Mechanical availability	-	85%
Use of availability	-	80%
Effective utilization	-	68%

Material	Swell Factor	Specific Gravity
Waste above bedrock		
- Granular surficials	20%	2.2
- Cohesive surficials	30%	2.2
Bedrock waste	30%	2.0
Coal (as fuel)	35%	1.55

An example of the shovel productivity calculations for coal is shown as follows:

Bucket capacity per cycle

$$= \frac{\text{Nominal bucket capacity times average fill factor}}{1 + \text{swell factor}}$$

$$= \frac{16.8 \times 0.8}{1.35}$$

= 9.96 bank cubic metres

Shovel capacity per 50 minute hour

= bucket capacity per cycle times cycles per 50 minute hour

= 9.96 x 83

= 827 bank cubic metres

Similarly, shovel productivities were calculated for waste above bedrock and bedrock waste materials and are summarized as follows:

Material	Bank Cubic Metres/Operating Hour
Waste above bedrock	
- Granular surficials	929
- Cohesive surficials	858
Bedrock waste	858
Coal	827

From these productivities and the quantities of each type of material excavated over the life of the mine, a weighted average hourly productivity of 865 bank cubic metres per hour was developed. The annual average shovel productivity output was calculated as the product of:

Scheduled hours 8760 x utilization 68% x hourly productivity 865

and equals 5.19 million bank cubic metres.

The annual shovel operating hours and fleet requirements are derived by dividing the scheduled quantities of materials to be mined by the estimated hourly productivities. Shown below is an example calculation of shovel hours and fleet size required in Year 3:

Material	Volume $\times 10^6$ BCM	Shovel Productivities BCM/hour	Shovel Operating Hours Required
Coal	5.63	827	6808
Waste above bedrock			
- Granular surficials	8.30	929	8934
- Cohesive surficials	2.58	858	3007
Bedrock waste	0.94	858	1096
Total:	17.45		19845

The shovel fleet size to meet the 60% generating station capacity factor specified is:

$$= \frac{\text{Shovel operating hours required in Year 3}}{\text{shovel operating hours per annum}}$$

$$= \frac{19,845}{8,760 \times .68}$$

$$= 3.33 \quad \text{(4 shovels)}$$

By directive from B.C. Hydro during each of the 35 years of thermal plant operation, the mine shovel fleet has to be large enough to cope with coal demands for a generating station capacity factor of 85% over a 5 month period. The scheduled capacity factors range from 55% to 70% over a 12 month period.

Year 3 has a scheduled capacity factor of 60%; therefore the increase in productive capacity required to meet the 85% capacity factor is

$$\frac{0.85}{0.60}$$

$$= 1.42$$

The 42% increase in productivity is applied to the calculated shovel operating hours to determine the maximum number of shovels that may be required.

The total shovel operating hours required to meet the 85% capacity factor are:

19,845 hours x 1.42 = 28,180 hours

The number of shovels required in year 3 are:

$$\frac{28,180}{8760 \times .68} = 4.73 \quad \text{(5 shovels)}$$

Table 6-3 summarizes the shovel hours and number of shovels required over the life of the mine to meet both the scheduled capacity factors as well as the 85% capacity factor.

Shovels for the scheduled capacity factors are replaced at the end of each 20 years or 120,000 operating hours. The additional shovels that may be required to meet the 85% capacity factor are never replaced.

Shovels load all the materials except topsoil and minor amounts of construction materials which are handled by a combination of scrapers, front-end loaders, and small trucks. However, to meet mine production schedules during pre-production Year -3 an estimated 2.05 million bank cubic metres of materials would be excavated under separate contract prior to shovel commissioning.

TABLE 6-3

Shovel Operating Hours and Fleet Sizes Required
To Meet Scheduled and 85% Generating Plant Capacity Factor

Hat Creek Project Mining Feasibility Report 1978

	SCHEDULED			85% GENERATING CAPACITY FACTOR			
Year	Generating Plant Capacity Factor as %	Shovel Operating Hours	Shovels Required	% Increase to 85% Capacity Factor	Shovel Operating Hours	Shovels Required	Actual Fleet Size
Pre-Prod.	-	21 305	3.0	-	-	-	3
1	69	17 398	2.9	23	21 400	3.6	4
2	60	19 635	3.3	42	27 881	4.7	5
3	60	19 845	3.3	42	28 180	4.7	5
4	61	21 760	3.7	39	30 246	5.1	6
5	65	23 023	3.9	31	30 160	5.1	6
6	70	23 360	3.9	21	28 266	4.8	6
7	70	23 812	4.0	21	28 813	4.8	6
8	70	23 812	4.0	21	28 813	4.8	6
9	70	23 812	4.0	21	28 813	4.8	6
10	70	23 480	3.9	21	28 410	4.8	6
11	70	23 602	4.0	21	28 558	4.8	6
12	70	23 602	4.0	21	28 558	4.8	6
13	70	23 590	4.0	21	28 544	4.8	6
14	70	23 590	4.0	21	28 544	4.8	6
15	70	23 590	4.0	21	28 544	4.8	6
16	65	25 563	4.3	31	33 488	5.6	6
17	65	27 949	4.7	31	36 613	6.2	7
18	65	27 949	4.7	31	36 613	6.2	7
19	65	27,938	4.7	31	36 599	6.2	7
20	65	27 927	4.7	31	36 584	6.2	7
21	65	27 927	4.7	31	36 584	6.2	7
22	65	28 682	4.8	31	37 573	6.3	7
23	65	27 797	4.7	31	36 414	6.2	7
24	65	27 797	4.7	31	36 414	6.2	7
25	65	27 797	4.7	31	36 414	6.2	7
26	55	25 539	4.3	55	39 586	6.7	7
27	55	15 299	2.6	55	23 714	4.0	7
28	55	14 320	2.4	55	22 196	3.7	7
29	55	14 332	2.4	55	22 215	3.7	7
30	55	14 332	2.4	55	22 215	3.7	7
31	55	14 320	2.4	55	22 196	3.7	7
32	55	14 321	2.4	55	22 198	3.7	7
33	55	14 321	2.4	55	22 198	3.7	7
34	55	14 321	2.4	55	22 198	3.7	7
35	55	14 309	2.4	55	22 179	3.7	7

In order to match short-term coal quality requirements, shovels allocated to dig coal can be expected to move up to 200 metres along a mining bench, three to eight times per week. Productivity estimates allow for these periodic moves together with the associated work to re-establish temporarily dormant working benches. In addition to providing excavating capacity to the truck fleet during shovel moves, the fleet has provided for the availability of two 11.5 cubic metre front-end loaders.

During Year -2, which is the break-in period for the mine operation, shovel productivities were reduced to 75% of that normally scheduled. In addition to this only eight months of the first operating year were scheduled and each shovel was phased in as a production unit as follows:

April	1/2 month at one shift per day 1/2 month at two shifts per day
May to December	7 months at three shifts per day

6.4 TRUCKS

Three sizes of rear dump haulage trucks are recommended for use at Hat Creek. They are 109-tonne capacity for coal haul, 136-tonne capacity for waste haul, and 32-tonne capacity for support tasks. The first two truck sizes are referred to as principal haulers and are utilized to haul the majority of coal and waste from the excavation. The third truck size is referred to as the auxiliary hauler and is utilized to perform those tasks not suited to the use of large trucks.

In recommending the principal haulage trucks, the following factors were taken into consideration:

(a) There would be a significant benefit to the mining operation in having the recommended trucks sized with boxes to permit loading by either electric shovel or front-end loader.

(b) The unit weight differential of coal and waste materials was such that an economic saving of 15% in capital and unit operating costs could be achieved by using two different sized trucks for principal haulers.

(c) The most commonly used and proven truck sizes in operating mines in British Columbia range in capacity from 91 to 155 tonnes. Shovel efficiencies encouraged recommending truck capacities within this range and as close to the maximum size as practical.

The auxiliary hauler was recommended for its:

(a) ability to travel over soft, unprepared ground conditions

(b) ability to operate in restricted working conditions

(c) size, which is suitable for intermittent and small tasks associated with low-grade coal and waste dump stockpiles, road and ramp construction

(d) proven capabilities in British Columbia mining and construction operations.

The three rear dump truck sizes recommended have the following features.

Principal Haulers

Coal	Waste
109-tonne capacity	136-tonne capacity
1200 H.P. diesel/electric wheel drive	1600 H.P. diesel/electric wheel drive
coal box - 109-tonne capacity	rock box - 136-tonne capacity
fleet size - pre-production 4 Year 12 (peak) 9	fleet size - pre-production 10 Year 16 (peak) 18

The principal haulers chosen permit loading by 11.5 cubic metre front-end loader and are shown on Figure 6-2.

Auxiliary Haulers

Construction Materials, Low-Grade Coal, Transport on Dumps Etc.

32-tonne capacity

415 H.P. diesel/mechanical drive

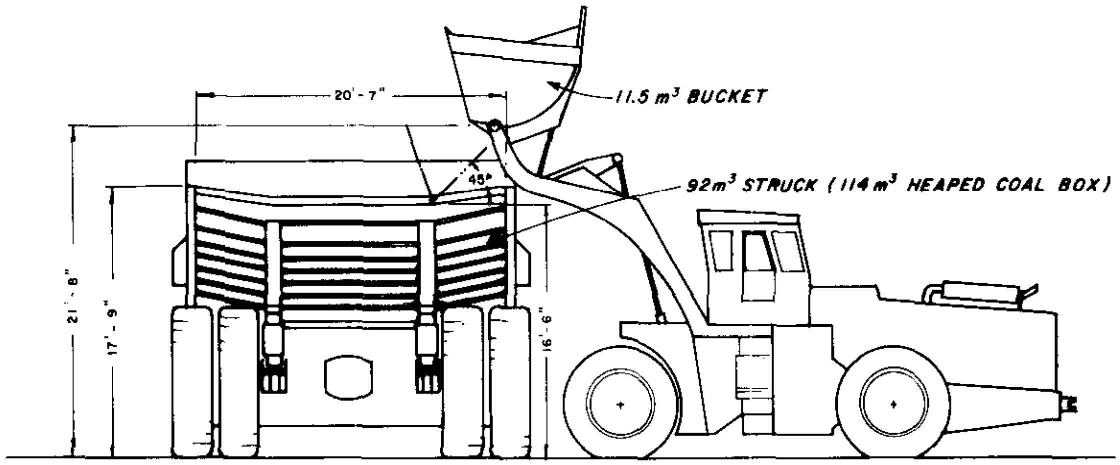
rock box - 32-tonne capacity

fleet size - pre-production 7
Year 5 (peak) 10

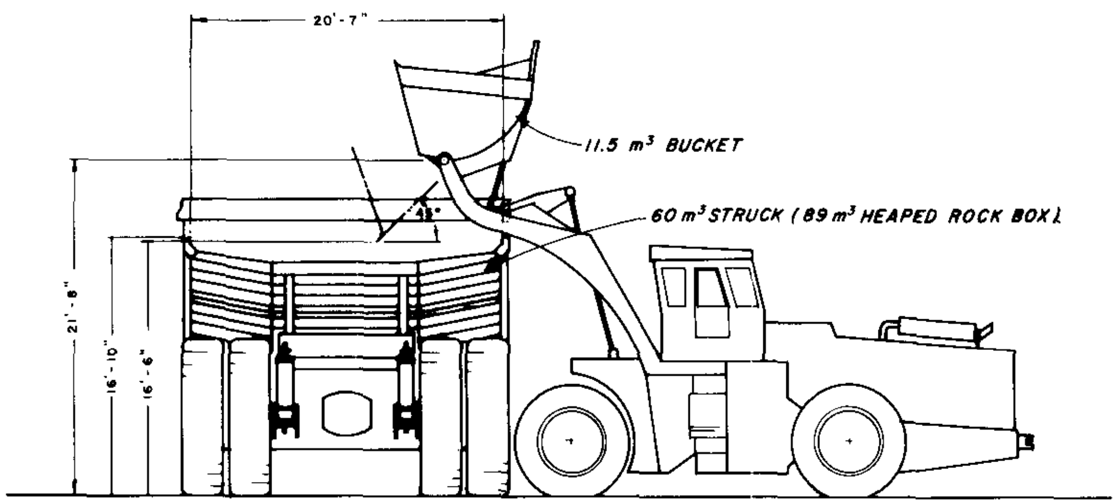
In developing the truck fleet size the mine planning concept was to group several sequential benches to best locate the fixed truck unloading station installations. Three average haul profiles were then evaluated for each unloading station: loaded up, loaded down, and loaded on a level haul.

A commercial computerized program was used to simulate truck haulage on the resulting profiles to determine productivity values for both coal and waste. Manual adjustments were made to convert computerized results from 60 minute hours to the more common 50 minute hour. Truck box capacities were slightly downrated to the next lowest non-fractional shovel bucket load as follows:

cominco-monenco joint venture



11.5 m³ F.E. LOADER LOADING 109 t TRUCK



11.5 m³ F.E. LOADER LOADING 136 t TRUCK

FIGURE 6-2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

11·5 m³ FRONT END LOADER
TRUCK LOADING CHARACTERISTICS

Coal Truck	Waste Truck
Capacity 109-tonnes	Capacity 136-tonnes
Actual load 104-tonnes	Actual load 128-tonnes
Shovel bucket loads - 7	Shovel bucket loads - 6

Effective utilization is estimated at 57% of 8760 scheduled hours or 5000 hours/year for each unit and includes the correction to 50 minute hours. A sample calculation of a truck haul cycle and productivity is as follows:

Year 3 - Coal Haulage Upward to 895 m Unloading Station

Average Haul Profile (loaded)	Computerized Haul Times in Minutes	
	Loaded	Empty
1. 500 metres flat	1.07	0.87
2. 380 metres ± 8%	1.79	0.71
3. 220 metres flat	0.51	0.35
4. 380 metres ± 8%	1.80	0.71
5. 250 metres flat	0.65	0.45
Total 1730 metres	5.82	3.09

Fixed time at shovel and unloading station = 5.25 minutes.

Total cycle time = 5.82 + 3.09 + 5.25 = 14.16 minutes.

Cycles per 60 minute hour = 4.24

Coal hauled per unit = 5000 hours x 104 tonnes x 4.24 trips/hr
= 2.205×10^6 tonnes per annum.

Coal required to be hauled on average profile = 6.928 x 106 tonnes.

Coal haulage units required = 6.928/2.205 = 3.14 (4 trucks).

Similarly, haul cycle calculations were made for the loaded downhill and load level hauls to the 895 m unloading station. It was determined that 4 trucks were also required on these hauls.

A 42% increase in productivity (refer Section 6.3) is applied to both coal and waste truck operating hours to determine the maximum number of trucks that may be required.

For the 109-tonne coal trucks the fleet size needed to meet an 85% generating capacity factor is:

$4 \times 1.42 = 5.68$

i.e., 6 trucks

Similar calculations were carried out to determine the 136-tonne truck fleet sizes throughout mine life. Tables 6-4 and 6-5 show the operating hours and fleet sizes necessary to meet the scheduled and 85% generating capacity factors throughout the life of the mine. For both the 109 and 136-tonne trucks the direct mining costs were estimated from the operating hours required to meet the actual generating capacity factor. Trucks for the scheduled capacity factors are replaced every 50,000 hours; additional trucks that may be required to meet the 85% capacity factor are never replaced.

All materials except for topsoil and gravels required for the construction of conveyor causeways and in-pit roads would be trucked to the truck unloading stations as described in the section following. Hat Creek soil conditions warrant special attention to the construction and maintenance of first class haul roads to ensure full operating efficiency of the truck fleet. Details of the program recommended are covered in Section 6.9.

The fleet of 32-tonne auxiliary haulers is required to perform the following essential tasks in support of mining operations

- bench preparation
- topsoil removal and replacement
- conveyor causeway construction
- road building activities in-pit and other areas
- rehandle of waste materials on sections of the dumps considered inaccessible to conveyors

It has not been practical to calculate haul cycles for this type of work due to difficulties in defining the scope in sufficient detail. The fleet size has been determined on the basis of judgement as to the number of truck hours required each year to perform the various tasks outlined.

TABLE 6-4

109 Tonne Truck Operating Hours And Fleet Sizes Required
To Meet Scheduled and 85% Generating Plant Capacity Factor

Hat Creek Project Mining Feasibility Report 1978

	SCHEDULED			85% GENERATING CAPACITY FACTOR			
Year	Generating Plant Capacity Factor as %	Truck Operating Hours	Trucks Required	% Increase To 85% Capacity Factor	Truck Operating Hours	Trucks Required	Actual Fleet Size
Pre-Prod.	-	2809	0.6	-	-	-	4
1	69	6630	1.3	23	8155	1.6	4
2	60	12 248	2.4	42	17 392	3.5	5
3	60	19 215	3.8	42	27 285	5.5	6
4	61	22 137	4.4	39	30 770	6.2	7
5	65	28 430	5.7	31	37 243	7.5	8
6	70	28 829	5.8	21	34 883	7.0	8
7	70	27 723	5.5	21	33 545	6.7	8
8	70	27 723	5.5	21	33 545	6.7	8
9	70	27 723	5.5	21	33 545	6.7	8
10	70	27 902	5.6	21	33 761	6.8	8
11	70	30 340	6.1	21	36 711	7.3	8
12	70	30 340	6.1	21	36 711	7.3	9
13	70	30 340	6.1	21	36 711	7.3	9
14	70	30 340	6.1	21	36 711	7.3	9
15	70	30 340	6.1	21	36 711	7.3	9
16	65	28 262	5.7	31	37 023	7.4	9
17	65	27 067	5.4	31	35 458	7.1	9
18	65	27 067	5.4	31	35 458	7.1	9
19	65	27 067	5.4	31	35 458	7.1	9
20	65	27 067	5.4	31	35 458	7.1	8
21	65	21 933	4.4	31	28 732	5.8	8
22	65	21 721	4.3	31	28 455	5.7	7
23	65	21 782	4.4	31	28 534	5.7	7
24	65	21 782	4.4	31	28 534	5.7	7
25	65	21 782	4.4	31	28 534	5.7	7
26	55	18 397	3.7	55	28 515	5.7	7
27	55	18 817	3.8	55	29 166	5.8	7
28	55	18 817	3.8	55	29 166	5.8	7
29	55	18 787	3.8	55	29 120	5.8	7
30	55	18 787	3.8	55	29 120	5.8	6
31	55	18 787	3.8	55	29 120	5.8	6
32	55	18 817	3.8	55	29 166	5.8	6
33	55	18 817	3.8	55	29 166	5.8	6
34	55	18 817	3.8	55	29 166	5.8	6
35	55	18 817	3.8	55	29 166	5.8	6

TABLE 6-5

136 Tonne Truck Operating Hours And Fleet Sizes Required
To Meet Scheduled and 85% Generating Plant Capacity Factor

Hat Creek Project Mining Feasibility Report 1978

	SCHEDULED			85% GENERATING CAPACITY FACTOR			
Year	Generating Plant Capacity Factor as %	Truck Operating Hours	Trucks Required	% Increase To 85% Capacity Factor	Truck Operating Hours	Trucks Required	Actual Fleet Size
Pre-Prod.	-	71 071	9.7	-	-	-	10
1	69	43 528	8.7	23	53 539	10.7	13
2	60	45 310	9.1	42	64 340	12.9	15
3	60	40 710	8.1	42	57 808	11.6	15
4	61	36 168	7.2	39	50 274	10.1	15
5	65	41 630	8.3	31	54 535	10.9	15
6	70	47 725	9.6	21	57 747	11.6	15
7	70	47 725	9.6	21	57 747	11.6	15
8	70	47 725	9.6	21	57 747	11.6	15
9	70	47 725	9.6	21	57 747	11.6	15
10	70	47 725	9.6	21	57 747	11.6	15
11	70	44 965	9.0	21	54 408	10.9	15
12	70	44 965	9.0	21	54 408	10.9	15
13	70	44 965	9.0	21	54 408	10.9	15
14	70	44 965	9.0	21	54 408	10.9	15
15	70	44 965	9.0	21	54 408	10.9	15
16	65	61 295	12.3	31	80 297	16.1	18
17	65	61 295	12.3	31	80 297	16.1	18
18	65	61 295	12.3	31	80 297	16.1	18
19	65	61 295	12.3	31	80 297	16.1	18
20	65	61 295	12.3	31	80 297	16.1	18
21	65	58 650	11.7	31	76 832	15.4	18
22	65	58 650	11.7	31	76 832	15.4	18
23	65	58 650	11.7	31	76 832	15.4	18
24	65	58 650	11.7	31	76 832	15.4	18
25	65	58 650	11.7	31	76 832	15.4	18
26	55	17 078	3.4	55	26 471	5.3	15
27	55	17 078	3.4	55	26 471	5.3	15
28	55	17 078	3.4	55	26 471	5.3	15
29	55	17 078	3.4	55	26 471	5.3	9
30	55	17 078	3.4	55	26 471	5.3	9
31	55	17 078	3.4	55	26 471	5.3	9
32	55	17 078	3.4	55	26 471	5.3	9
33	55	17 078	3.4	55	26 471	5.3	9
34	55	17 078	3.4	55	26 471	5.3	9
35	55	17 078	3.4	55	26 471	5.3	9

6.5 TRUCK UNLOADING STATIONS

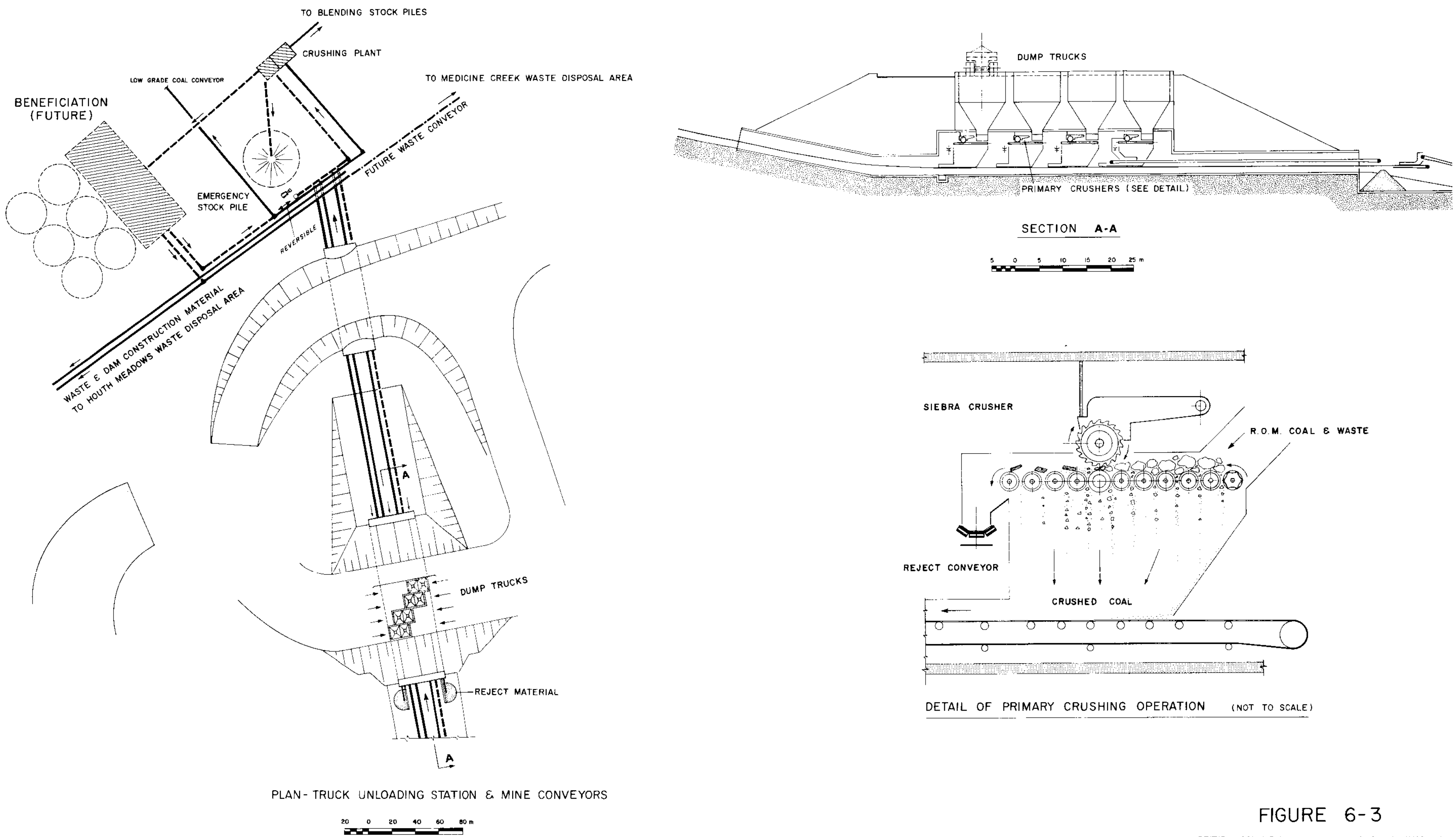
In order to transfer materials from in-pit truck haulers to conveyor belt transport, three truck unloading stations have been provided. Location and time of installation of the three stations would be: at bench elevation 895 metres ASL during pre-production, at bench elevation 820 metres ASL in Year 5, and at bench elevation 730 metres ASL in Year 20. Horizontal distance between each station is 450 to 600 metres. Construction of the three unloading stations is on excavations into footwall rocks, and vertical location has been selected in consideration of the center of gravity of the haulage tasks associated with downward versus upward trucking.

As illustrated on Figure 6-3, a truck unloading station is comprised of: two compartment hopper, coarse grizzly screens, primary crusher, and feeder. Each hopper/crusher element is designed to feed a single conveyor line. Although each unloading station is presently planned on the basis of three conveyor lines, a fourth element may be added to the upslope end of each station should an additional conveyor line be required. Hopper size has been matched to the capacity of three 136-tonne trucks to permit rapid dumping.

The hopper design incorporates two compartments and two outlet openings decreasing the overall hopper height and permitting the installation of reciprocating feeders of smaller capacity than would be required if only one outlet opening were provided. This improves the reliability of the system.

Several types of feeders were reviewed for installation below the hoppers. The choice at this stage of the feasibility study is a reciprocating feeder with hydraulic drive. Its advantages include sturdy construction, positive feeding action, and ease of feed rate adjustments. This unit feeds, in turn, the rotating screen that forms part of the Siebra crusher as illustrated. At this stage of the study the recommended screen opening is 30 cm.

Reject material such as petrified wood, chunks of siderite, etc., would be trucked out of the pit periodically.



NOTE
---- DOTTED LINE INDICATES FUTURE EXTENSION FOR BENEFICIATION PLANT

FIGURE 6-3

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

TRUCK UNLOADING STATION
& DISTRIBUTION AREA

6.6 CONVEYORS

661 GENERAL

Conveyors are used extensively in the development of the mine, as they provide a suitable and economical means of transport for the large quantities of material to be moved between the pit, the generating station, and the waste dump areas at Houth Meadows and Medicine Creek.

Conveyors form an integral part of the mine system and may be classified in three categories:

- permanent conveyors
- semi-permanent conveyors
- shiftable conveyors

Permanent conveyors are installed at the start of mining and remain in operation during the entire 35-year period. These include the mine conveyors, the coal transfer conveyors in the preparation area, and the overland conveyor systems for both coal and waste.

The semi-permanent conveyors are located at the edge of the dump areas and serve to feed the shiftable conveyors on the dumps. The semi-permanent conveyors are kept in place for a number of years at a given elevation and re-located to a higher elevation immediately following completion of each 35-metre lift on the dump.

The shiftable conveyors, located on the waste dumps, feed waste materials to the spreaders which in turn deposit the material in layers of up to 35 metres in thickness. The shiftable conveyor would be moved periodically, parallel to its own axis, over a distance of about 40 metres. The frequency of shifting will depend on the rate at which material is deposited and the length of the conveyor in use at the time. On the average, the conveyor would be shifted once every six to eight weeks, although short conveyors would require more frequent shifting.

The total length of all conveyors in use at any time would vary; however, the maximum length in use during the 35-year period is estimated at 27 435 metres.

The capacity of all conveyors, with the exception of the overland coal conveyor and those performing reclaim functions only, is determined by the hourly production of the 16.8-cubic metre shovels and 11.5-cubic metre front-end loaders. The maximum loading per conveyor, based on the peak output of two shovels, is approximately 3000 loose cubic metres per hour. Consequently, the design capacities used are:

coal conveyors
before blending .... 3200 tonnes per hour

waste conveyors .... 5000 tonnes per hour

These capacities have led to the preliminary recommendation of 1200-millimetre belt width for the main conveyors and 35^{o} troughing idlers and a speed of 5 metres per second. A surcharge angle of 20^{o} was used in the calculation of the conveyor capacity. As a result, the capacity of the three conveyor lines in the mine is substantially greater than the average hourly mine output. Applying a conveyor line utilization factor of 85%, equivalent to 7400 hours per year, the average load on the conveyors during the peak waste production years is approximately 47% of design capacity.

The criteria issued by the Conveyor Equipment Manufacturers' Association for belt load, friction factors, and power calculations have been used for preliminary selection. The maximum slope adopted for the conveyors is 14^{o}, although, where possible, a smaller angle should be used, particularly at the conveyor transfer and loading points. Most of the conveyors will require steel cord belting due to their great length and high belt tensions, although the shorter conveyors, particularly in the coal preparation area and on the stackers, reclaimers, and spreaders, could probably be equipped with fabric belting. All conveyors should be designed to start fully loaded under the most severe conditions.

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

(a) sequential starting and stopping of all pieces of equipment forming one line of transportation;

(b) chute plugging switches which detect the blockage of a chute and stop the system;

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

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

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

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

(g) holdbacks to prevent an inclined conveyor from running backwards under load; and

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

When the conveyor system is fully extended, the total installed motor horsepower will be about 63,900 H.P. A typical operating load is estimated at approximately 38,000 H.P. but will vary greatly depending on the productivity of the mine. The average hourly operating electrical load calculated for the peak production years is 16 000 kW.

Figures 6-4 and 6-5 illustrate typical conveyor layout plans for Years 1 to 15 and Years 16 to 35, respectively, of the mine development. The conveyor layout for the coal preparation area is shown on Figure 6-6.

662 MINE CONVEYORS

The mine conveyors shown in Figure 6-7 and listed in Table 6-6 are located on the inclined ramp on the north side of the pit.

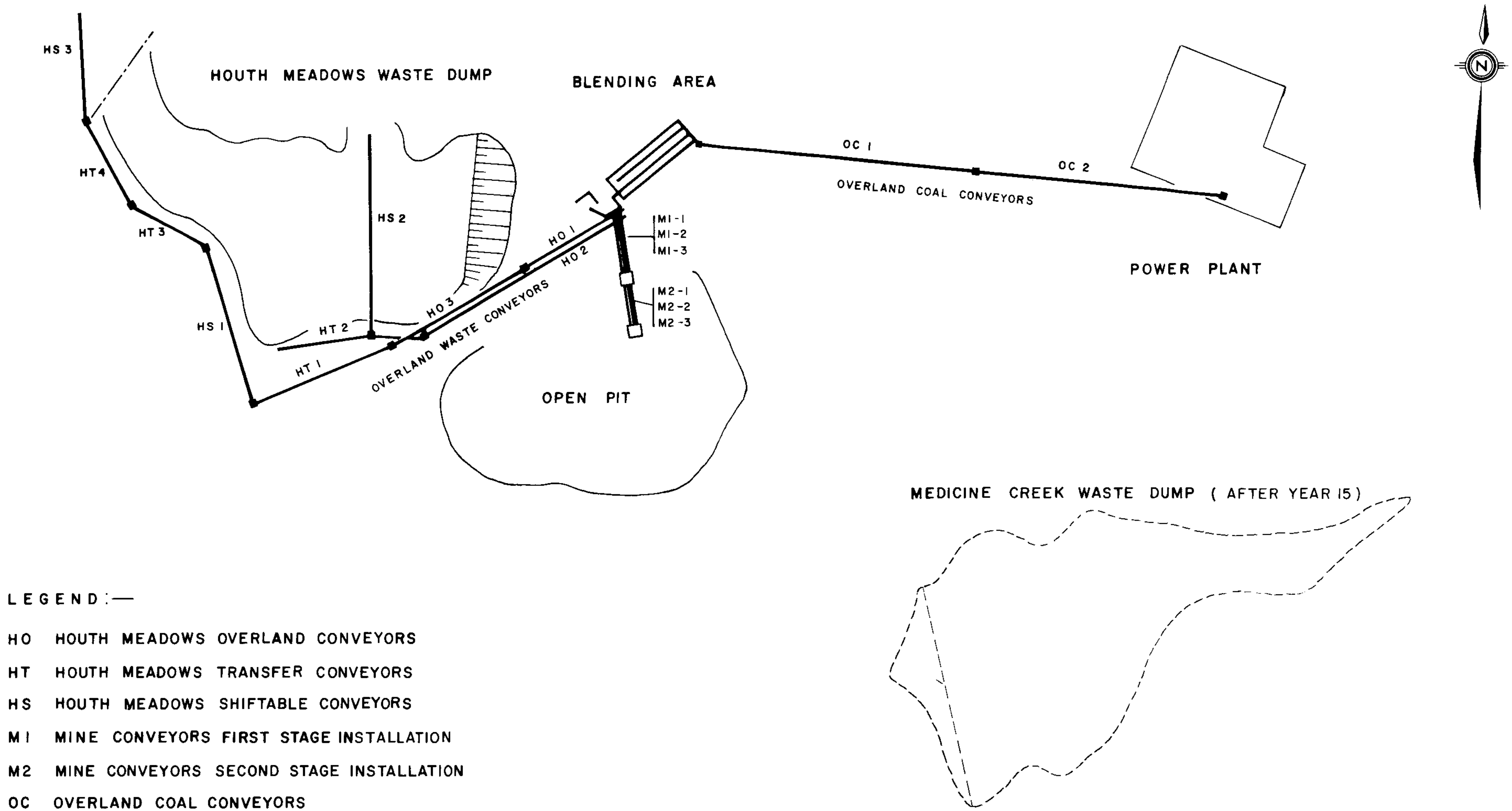
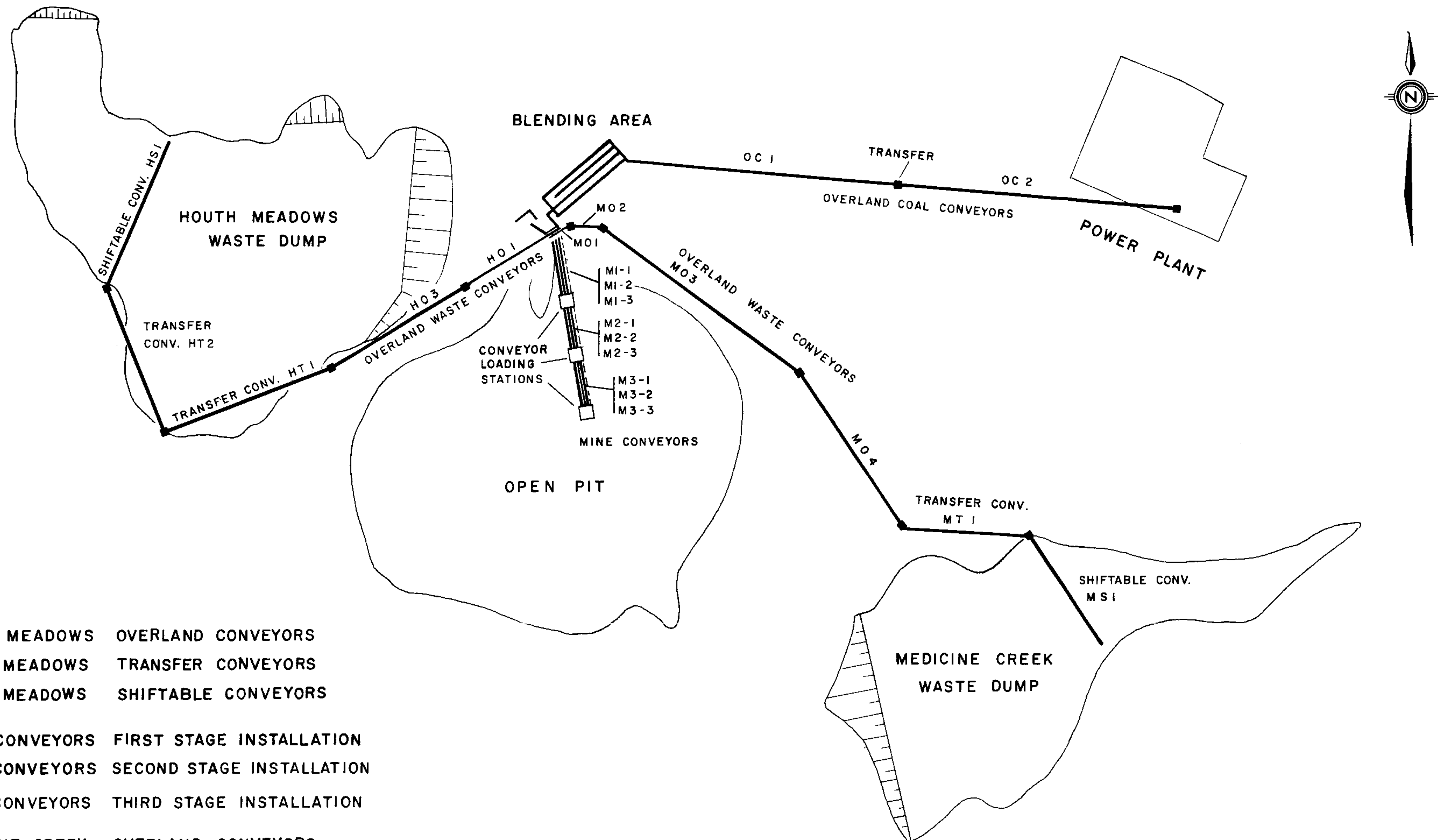


FIGURE 6-4

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

TYPICAL CONVEYOR LAYOUT
DURING YEARS 1 TO 15



LEGEND:—

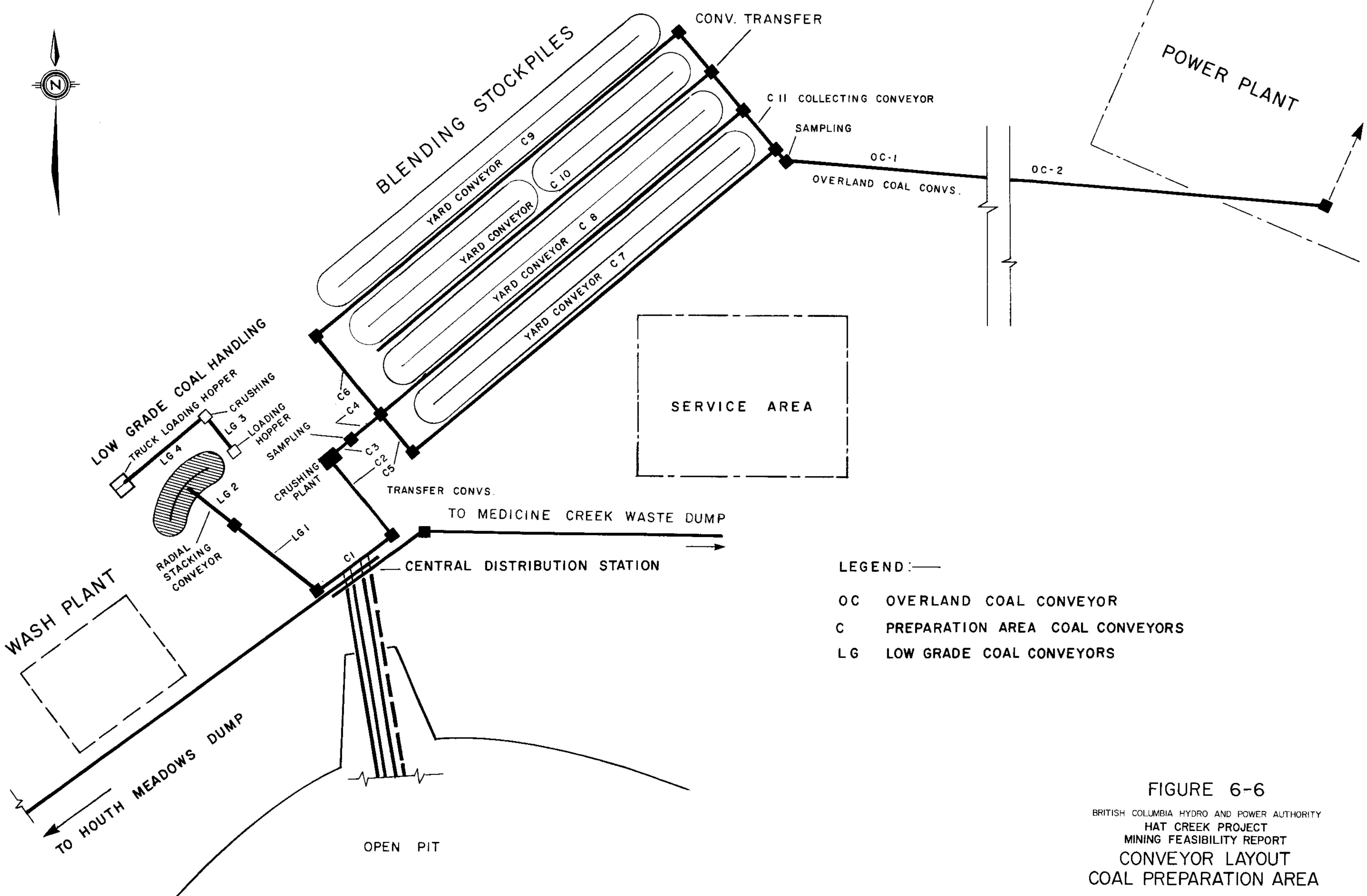
HO	HOUTH MEADOWS	OVERLAND CONVEYORS
HT	HOUTH MEADOWS	TRANSFER CONVEYORS
HS	HOUTH MEADOWS	SHIFTABLE CONVEYORS
M1	MINE CONVEYORS	FIRST STAGE INSTALLATION
M2	MINE CONVEYORS	SECOND STAGE INSTALLATION
M3	MINE CONVEYORS	THIRD STAGE INSTALLATION
MO	MEDICINE CREEK	OVERLAND CONVEYORS
MT	MEDICINE CREEK	TRANSFER CONVEYORS
MS	MEDICINE CREEK	SHIFTABLE CONVEYORS
OC	OVERLAND COAL CONVEYORS	

FIGURE 6-5

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

TYPICAL CONVEYOR LAYOUT
DURING YEARS 16 TO 35

cominco-monenco joint venture



LEGEND:—

OC	OVERLAND COAL CONVEYOR
C	PREPARATION AREA COAL CONVEYORS
LG	LOW GRADE COAL CONVEYORS

FIGURE 6-6

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

CONVEYOR LAYOUT
COAL PREPARATION AREA

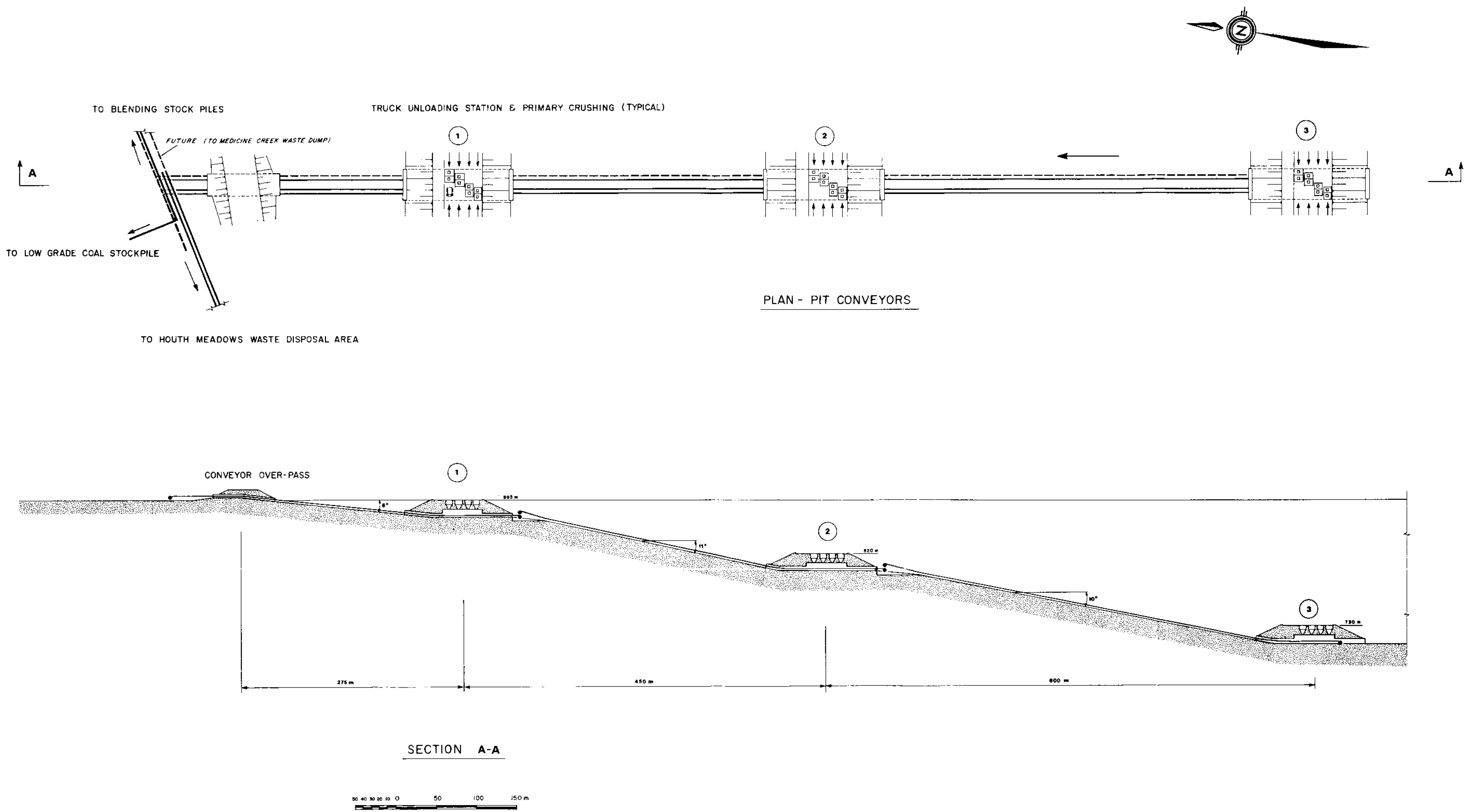


FIGURE 6-7
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT
MINE CONVEYORS

TABLE 6-6

Mine Conveyors

Hat Creek Project Mining Feasibility Report 1978

Conveyor	Year of Installation	Length (metres)	Lift (metres)	Width (mm)	Capacity (tonnes/hr)	Nominal Speed (metres/s)	Calculated Horsepower	Installed Horsepower
M1-1 - Mine	-2	360	40	1200	5000	5	1085	1100
M1-2 - Mine	-2	360	40	1200	5000	5	1085	1100
M1-3 - Mine	-2	360	40	1200	5000	5	1085	1100
SUB TOTAL		1080					3255	3300
M2-1 - Mine	5	470	75	1200	5000	5	1874	2000
M2-2 - Mine	5	470	75	1200	5000	5	1874	2000
M2-3 - Mine	5	470	75	1200	5000	5	1874	2000
SUB TOTAL		1410					5622	6000
M3-1 - Mine	20	600	90	1200	5000	5	2378	2400
M3-2 - Mine	20	600	90	1200	5000	5	2378	2400
M3-3 - Mine	20	600	90	1200	5000	5	2378	2400
SUB TOTAL		1800					7134	7200
TOTAL		4290					16011	16500

The slopes of conveyors between the lowest unloading station, mid-ramp station, pit rim station, and the central distribution system are 10^{o}, 11^{o}, and 6^{o}, respectively.

These three conveyor lines would receive the four types of mine material from the truck unloading station. Coal would be transported to the crushing plant and blending area, while general mine wastes, construction grade materials, and low-grade coal would be conveyed directly to their designated areas. The transport of these four materials on three conveyors must be scheduled during mine production planning; however, this is not anticipated to present any serious problems as the amount of low-grade coal to be transported is relatively small. Should a coal wash plant ever be required, it would be necessary to increase the number of mine conveyors to four to permit the simultaneous transport of dirty and clean coal. For this reason, space has been provided for an additional conveyor as indicated on the drawings.

Service roads should be located alongside the conveyors to facilitate inspection, maintenance, and clean-up of oversized materials rejected from the crushing plant. The truck unloading stations should be so designed that the service road would pass under them, permitting a continuous road system from the top to the bottom of the pit.

663 CENTRAL DISTRIBUTION SYSTEM

The distribution of the four types of materials to the waste disposal areas, embankments, coal preparation area, and low-grade stockpile by conveyors would take place at the head end of the mine conveyor ramp. From the central distribution system, it is recommended that two waste conveyors and one combined coal and low-grade coal conveyor be established. The latter should be reversible thereby permitting the transport of low-grade coal westward, and of the fuel-grade coal eastward.

Complete change-over flexibility has been incorporated in the distribution system, giving each mine conveyor access to all destinations. This would be achieved by installing movable head pulleys which permit the discharge point of these conveyors to travel to each of the transfer or overland conveyors.

664 COAL TRANSFER CONVEYORS IN PREPARATION AREA

The coal conveying system from the central distribution point to the crushing plant/blending area consists of a number of belt conveyors of varying lengths.

As listed on Table 6-7 and shown on Figure 6-6 these include:

C1 - reversible transfer conveyor at the distribution point

C2 - plant conveyor between the distribution point and the crushing plant

C3 - conveyor between crushing plant and sampling plant

C4 - conveyor between sampling plant and distribution point at the stockpile area

In the blending area, conveyors C5 and C6 carry the coal to the yard conveyors located between the stockpiles. A total of four yard conveyors are to be installed, three of these, C7, C8, and C9, to deliver coal to the piles and to receive the coal during the reclaim operation, and the fourth, C10, to serve only as a reclaim conveyor.

The yard conveyors deliver the coal to a conveyor C11, which transfers the blended product to the overland coal conveyor and second sampling house.

665 OVERLAND COAL CONVEYORS TO THE GENERATING STATION

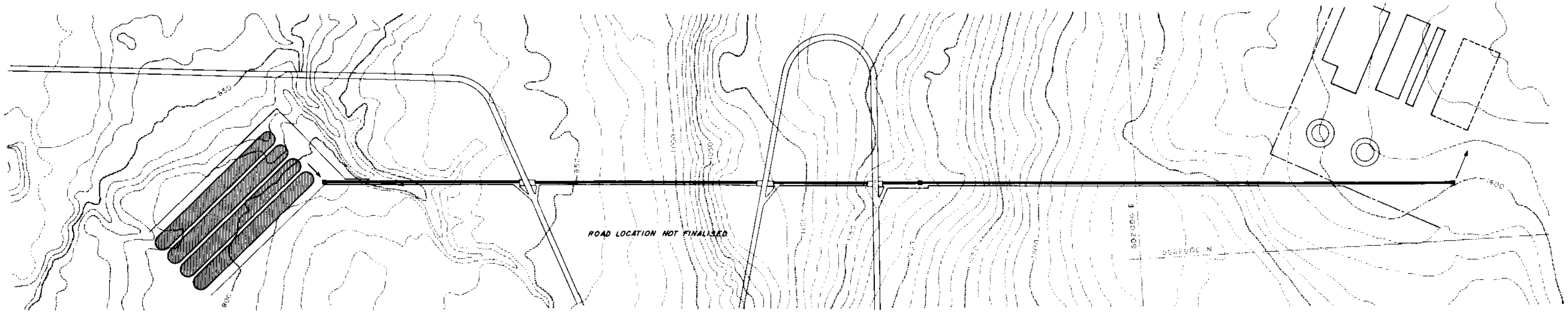
Overland coal conveyors OC1 and OC2, comprising one conveyor line in two flights, are shown on Figure 6-8 and listed in Table 6-8. These conveyors should have a capacity of 2500 tonnes per hour and run at a speed of about 4.2 metres per second. This capacity has been recommended to ensure compatibility with the feed and stockpile requirements of the generating station. As the average consumption at the generating station is 1150 tonnes per hour, based on 65%

TABLE 6-7

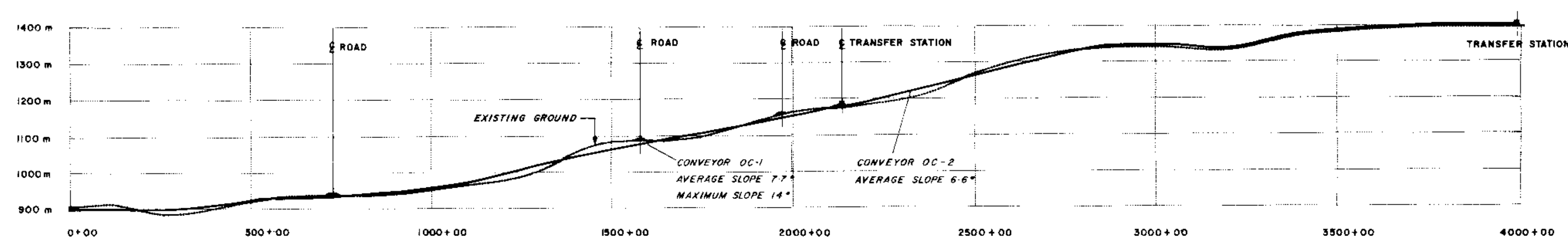
Coal Transfer Conveyors in Preparation Area

Hat Creek Project Mining Feasibility Report 1978

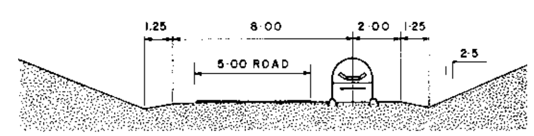
Conveyor	Year of Installation	Length (metres)	Lift (metres)	Width (mm)	Capacity (tonnes/hr)	Nominal Speed (metres/s)	Calculated Horsepower	Installed Horsepower
C1 - Transfer to Crushing Plant (reversible)	-2	60	6	1200	3200	5	117	125
C2 - Transfer to Crushing Plant	-2	115	24	1200	3200	5	309	350
C3 - Transfer from Coal Crushing Plant	-2	40	6	1200	3200	5	103	125
C4 - Transfer to Blending Area	-2	70	6	1200	3200	5	123	125
C5 - Transfer to Blending Area	-2	70	6	1200	3200	5	123	125
C7 - Yard	-2	670	10	1200	3200	5	568	600
C8 - Yard	-2	670	10	1200	3200	5	568	600
C11 - Collecting	-2	220	6	1200	3200	4	175	200
SUB TOTAL		1915					2086	2250
C6 - Transfer to Blending Area	-1	135	6	1200	3200	5	166	200
C9 - Yard	-1	670	10	1200	3200	5	568	600
C10 - Yard	-1	570	6	1200	2500	4	451	500
SUB TOTAL		1375					1185	1300
TOTAL		3290					3271	3550



PLAN - OVERLAND COAL CONVEYOR



PROFILE - CONVEYOR ℄



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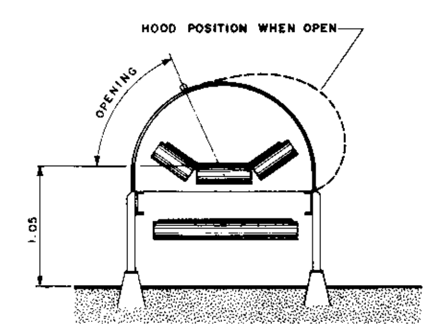
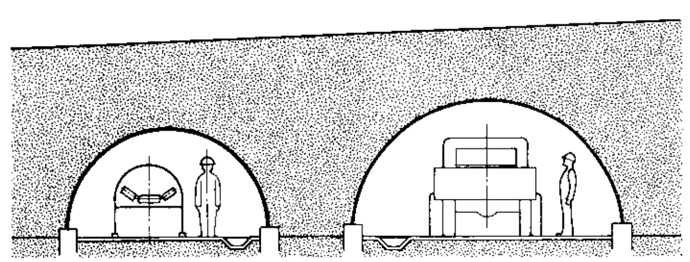
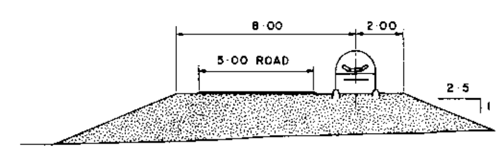


TYPICAL SECTION - COAL CONVEYOR



SECTION - HIGHWAY OVERPASS



TYPICAL SECTION - FILL

FIGURE 6-8

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

OVERLAND COAL CONVEYORS

TABLE 6-8

Overland Coal Conveyors to Generating Station

Hat Creek Project Mining Feasibility Report 1978

Conveyor	Year of Installation	Length (metres)	Lift (metres)	Width (mm)	Capacity (tonnes/hr)	Nominal Speed (metres/s)	Calculated Horsepower	Installed Horsepower
OC1 - Overland	-2	2100	270	1200	2500	4	3952	4000
OC2 - Overland	-2	1900	250	1200	2500	4	3554	4000
TOTAL		4000					7506	8000

capacity factory, the recommended delivery rate of 2500 tonnes per hour contributes to greater flexibility for the bunker filling mode of operation.

The great difference in elevation and long distance between the mine and the generating station require that the conveyor line be constructed in two flights with a transfer point midway. The average slope on these conveyors is 7.7^{o} and 6.6^{o}, respectively, while the maximum slope in any length is 14^{o}.

The conveyor is provided with a hood for protection against the wind and containment of dust. In certain areas where severe snowdrifts may occur, the conveyor should be completely covered in an enclosed gallery consisting of a simple steel structure covered with standard cladding and mounted on concrete foundations. The location of these galleries must be determined by observation of the area over several winters to determine where the most severe drifting occurs. A total of 2000 metres of enclosed galleries has been included in the cost estimate.

666 LOW-GRADE COAL TRANSFER FACILITY

Low-grade coal is transported from the central conveyor distribution point by transfer conveyor to a load-out facility. Although intermittent, delivery will be at a rate of 3000 loose cubic metres per hour. The planned load-out facilities include a radial stacker which deposits the material into a surge pile sized at about one week's average production of 10 000 to 12 000 tonnes.

In order to reduce the potential for spontaneous combustion, low-grade coal will be crushed before deposition and compaction in the stockpile area. A small crushing installation has therefore been included in the cost estimate. This crusher would be fed from the surge pile by means of a 5.4-cubic metre front-end loader and a conveyor. The relatively small quantity of low-grade coal would require the crusher to be in operation for 40 hours (5 shifts) per week at a rate of 300 tonnes per hour.

Annual production statistics for low-grade coal during Years -1, 1, 2, and 3 indicate sufficiently small quantities that the need for a low-grade coal handling facility during those years is doubtful. Therefore, the construction and intial operation of this facility has been deferred to commence in Year 4. During the initial period from Year -1 to Year 3, the low-grade coal will be hauled directly from the open-pit to the low-grade stockpile by 109-tonne trucks.

The material handling equipment in the low-grade coal system comprises:

C1 - transfer conveyor in the central distribution system

L91 - transfer conveyor to the low-grade preparation area

L92 - radial stacker with a lifting height of approximately 18-metres

L93 - transfer conveyor with loading hopper to the crushing plant

L94 - transfer conveyor from the plant to a truck loading hopper

Table 6-9 lists the design details of these conveyors.

667 WASTE CONVEYORS

The conveying system for waste material between the central distribution point and the disposal areas at Houth Meadows and Medicine Creek as shown on Figures 5-9 and 6-1 includes:

- permanent overland conveyors
- semi-permanent conveyors
- shiftable conveyors
- trippers
- spreaders

TABLE 6-9

Low-Grade Coal Transfer Conveyors

Hat Creek Project Mining Feasibility Report 1978

Conveyor	Year of Installation	Length (metres)	Lift (metres)	Width (mm)	Capacity (tonnes/hr)	Nominal Speed (metres/s)	Calculated Horsepower	Installed Horsepower
LG1 - Transfer	4	150	6	1200	5000	5.0	214	250
LG2 - Stacking	4	75	17	1200	5000	5.0	400	400
LG3 - Reclaim	4	30	6	900	300	2.5	20	20
LG4 - Truck Loading	4	100	25	900	300	2.5	60	60
TOTAL		355					694	730

Until Year 15 all waste material will be deposited in the Houth Meadows waste dump area via a double conveyor line with two independent shiftable conveyor lines and spreaders.

The second overland conveyor system and associated shiftable conveyors, trippers, and spreaders are transferred from Houth Meadows to the Medicine Creek dump in Year 15.

The shiftable conveyor, mounted with two shifting rails, is moved laterally by means of a tractor equipped with a side boom. Steel ties are recommended for the conveyor supports.

Grading and bulldozer clean-up of the waste dump surface would be done prior to relocating the conveyor. General clean-up of spillage and material accumulating under the belt should be performed by a small traxcavator.

Tables 6-10 and 6-11 list the conveyors required for the Houth Meadows and Medicine Creek disposal areas.

The main specifications for the spreaders are as follows:

Length of loading boom	40 metres
Length of discharge boom ...	40 metres
Belt width	1400 millimetres
Belt speed	4.2 metres per second
Travelling speed	4.5 metres per minute
Slewing speed at discharge point	29 metres per minute
Discharge height	18 metres
Distance between crawler-mounted track centres	7.5 metres
Track length	7.6 metres (approximately)

TABLE 6-10

Waste Conveyors to Houth Meadows

Hat Creek Project Mining Feasibility Report 1978

Conveyor	Year of Installation	Length (metres)	Lift (metres)	Width (mm)	Capacity (tonnes/hr)	Nominal Speed (metres/s)	Calculated Horsepower	Installed Horsepower
LINE #1								
H01 - Overland	-2	850	5	1200	5000	5	764	800
HT1 - Transfer	-2	1000	10	1200	5000	5	980	1000
HS1 - Shiftable	-2	1000	10	1200	5000	5	980	1200
SUB TOTAL		2850					2724	3000
H03 - Overland	2	600	75	1200	5000	5	1976	2000
HS1 - Shiftable Extension	2	400	-	1200	5000	5	620*	400
HS3 - Shiftable	2	500	10	1200	5000	5	747	800
SUB TOTAL		1500					3334	3200
H03 - Overland Extension	8	550	70	1200	5000	5	1836	2000
HT4 - Transfer	8	700	10	1200	5000	5	647	800
HS1 - Shiftable Extension	8	400	-	1200	5000	5	-	-
HS3 - Shiftable	8	200	-	1200	5000	5	-	-
SUB TOTAL		1850					2483	2800
HT1 - Transfer Extension	19	350	30	1200	5000	5	883	1000
HT5 - Transfer	19	1000	38	1200	5000	5	981	1000
SUB TOTAL		1350					1864	2000
TOTAL LINE #1		7550					10405	11000
LINE #2								
H02 - Overland	-1	1250	30	1200	5000	5	1578	2000
HT2 - Transfer	-1	900	10	1200	5000	5	903	1000
HS2 - Shiftable	-1	1250	10	1200	5000	5	1176	1200
SUB TOTAL		3400					3657	4200
H02 - Overland Extension	4	400	70	1200	5000	5	1719	2000
HT2 - Transfer Extension	4	250	-	1200	5000	5	195*	100
HT3 - Transfer	4	500	10	1200	5000	5	647	800
HS2 - Shiftable	4	350	-	1200	5000	5	273	400
SUB TOTAL		1500					2834	3300
HT3 - Transfer Extension	8	200	-	1200	5000	5	-	-
SUB TOTAL		200						
TOTAL LINE #2		5100					6491	7500

* Note: Spare H.P. capacity available in initial conveyr.

TABLE 6-11

Waste Conveyors to Medicine Creek

Hat Creek Project Mining Feasibility Report 1978

Conveyor	Year of Installation	Length (metres)	Lift (metres)	Width (mm)	Capacity (tonnes/hr)	Nominal Speed (metres/s)	Calculated Horsepower	Installed Horsepower
M01* - Overland	15	150	5	1200	5000	5	218	250
M02* - Overland	15	250	30	1200	5000	5	798	800
M03* - Overland	15	1650	120	1200	5000	5	3894	4000
Extension		250	-	1200	5000	5		
M04 - Overland	15	1400	140	1200	5000	5	3906	4000
MT1 - Connecting	15	1200	148	1200	5000	5	regenerative**	
MT2* - Transfer	15	800	10	1200	5000	5	825	1000
MS1* - Shiftable	15	1600	10	1200	5000	5	1371	1600
TOTALS (new equipment only)		2850					3906	4000

* Note: Relocated from Houth Meadows

** Note: Regenerated power at initial installation is 2400 H.P. due to drop of 148-metres. As the conveyor is relocated the drop diminishes and at the final location the conveyor becomes horizontal, *requiring 1200 H.P.*

668 DUST CONTROL IN THE CONVEYOR SYSTEM

Dust control may be required at various areas of the mine operation, including:

- mine and waste conveyors and disposal areas
- crushing plants and transfer houses
- blending area
- overland coal conveyors and transfer points

668.1 Mine and Waste Conveyors and Disposal Areas

A water main equipped with dry hydrants should be installed along the conveyors so that water is available to the areas around and below the conveyors as required. Freshly mined waste material should be sufficiently moist so that dust will not blow off the conveyors. Therefore, the waste conveyors and dumping equipment should not require dust control equipment or dust covers.

668.2 Crushing Plant and Transfer Houses

Dust collection systems within the crushing plant and transfer houses are provided to minimize the amount of dust in the air and on the floors. A collection system would normally consist of enclosures around the conveyor transfer points and other equipment. The dust laden air would be withdrawn from these enclosures and passed through the dust collectors. Although selection of suitable dust collectors would take place in the design stage of the project, provision has been made for bag filters.

668.3 Blending Area

The degree of dusting anticipated in the blending area has not yet been determined, but provision has been made for a water spray system around the stockpile and blending area. The conveyors along the stockpiles cannot be covered, as an enclosure or hood would interfere with the operation of the reclaimers and stackers.

668.4 <u>Overland Coal Conveyors and Transfer Points</u>

These conveyors are protected by hoods which will reduce the possibility of wind-blown coal dust. The transfer houses will be equipped with dust collection systems as described previously.

6.7 CRUSHING AND SAMPLING

671 PRIMARY SCREENING AND CRUSHING

All coal and waste material will be passed through a track unloading station. Lump sizes in excess of 1-metre will be broken down by a hydraulic breaker while material less than 1-metre would pass through a coarse grizzly screen. Material will be prepared for transportation on the belt conveyor by reduction in a primary crusher to a maximum size of 300 millimetres. This would be performed below the hoppers in the truck unloading stations.

Several types of crushers were considered for installation below the hoppers, but in view of the weak characteristics of the materials a Krupp Siebra crusher, as shown on Figure 6-3, was chosen for this stage. This machine combines a roller screen with a lump breaker and permits the passage of uncrushable or hard lumps which must then be removed from the crusher area by separate means. Further investigation of sizing methods are recommended in the final design stage.

The combination with a roller screen provides the advantage of a shallower excavation and hopper structure since less vertical space is required than other types of installations.

Removal of the uncrushed, oversized lumps is accomplished by means of short cross conveyors, which receive the lumps and transport them to a deposit area outside the building. They are then deposited by front-end loader into 32-tonne trucks and hauled to the waste dump areas.

672 SECONDARY SCREENING AND CRUSHING

The purpose of the screening and crushing plant is to reduce the coal particles to 50 millimetres maximum dimension which is suitable for delivery to the generating station. Coal would be delivered from the pit on the inclined mine conveyors at a rate ranging from an average of 1500 tonnes per hour to a peak of 3200 tonnes per hour. It is the peak rate which determines the capacity of the screening and crushing plant. The coal would be received in the screening and crushing plant by a distributing hopper. Each hopper would direct the flow over three screens where the material is separated into two size classifications:

- oversized material of greater than 50 millimetres
- undersized material of less than 50 millimetres

The oversized material would be crushed to 50 millimetres or less, and all material transported to the blending piles on a single conveyor.

672.1 Equipment Selection

Final selection of the equipment can only take place in Phase II of the study upon performance of the necessary screening and crushing tests by the manufacturers and the economic and technical evaluations of the proposals submitted by the manufacturers.

Selection of the crushers must take into consideration the material handling characteristics resulting from the bentonitic clays which form part of the deposit.

The secondary crushers must therefore incorporate design features which could accommodate the potential problems with bentonite, such as plugging of the crusher inlet and crusher rollers. The crusher must also be able to crush hard materials such as pieces of siderite and petrified wood, etc., which were not rejected in the primary crushing operation.

These design requirements have led to the preliminary selection of an impact crusher with an internal heating coil in the impact plates through which hot oil would be circulated.

The hot surface of the plates should prevent buildup of the sticky material and thus reduce plugging of the crusher. This type of crusher is extensively used in limestone quarries, coal, and lignite mines.

The equipment included in the estimates comprises:

(a) distributing hopper with three outlet openings and control gates;

(b) three vibrating feeders 100 tonnes per hour capacity each;

(c) three vibrating screens - 100 tonnes per hour capacity each; and

(d) auxiliary equipment such as dust control, monorails, control and safety devices, etc.

673 SAMPLING SYSTEMS

Automatic sampling systems should be installed on the conveyor system leading both to and from the blending area.

The first sampling installation is located after the crusher house and takes the primary cut at the head pulley of the transfer conveyor C3. This sampler determines the aggregate quality of the coal in the stockpiles and the need for addition of high-quality coal to the average grade stockpile.

Analysis of the sample would be done in a laboratory and the cumulative result calculated upon completion of the blending pile.

A weight installation and recorder in conjunction with the sampler is required in order to determine the weighted average calorific value of all the coal in the blending pile.

In addition, provision has been made for the installation of a stationary ash monitor. This instrument permits the determination of the ash content within three minutes of the taking of the sample and is capable of continuous recording of the ash content.

Since a direct relation exists between the ash content and the calorific value of the coal, the recorded ash values can easily be converted into calorific values.

A second sampler installation, located at the transfer point from the cross conveyor at the blending area to the overload coal conveyor, will provide a measurement of the quality actually delivered to the power plant and thus serve as a cross check on the blending process.

6.8 STOCKPILE AND BLENDING SYSTEM

681 SYSTEM RECOMMENDED

The general arrangement of the coal blending system is shown on Figure 6-9 and the selected method, layout, and equipment is described as follows.

The run-of-mine coal from the various mining areas would be brought out of the pit and directed by conveyor to one of three blending areas termed as follows:

- average-grade stockpiles
- high-grade stockpiles
- emergency stockpiles

Two qualities of coal could be deposited separately in the above locations, and the emergency stockpile used to receive either average or high-grade depending on the particular needs at any point in time.

The average and high-grade blending areas consist of two stockpiles each, one for the deposition of material and the other for reclamation. The average-grade piles contain about 280 000 tonnes of coal, or one week's production. The high-grade piles are designated to receive only low sulphur high-grade D-Zone coal. Each of these would contain about 135 000 tonnes of high-grade coal, or one-half week's mine production.

When required, the material from both systems is mixed on the overland conveyor in the proportions required to achieve the desired coal quality.

The emergency blending area is available to receive coal should the other blending areas be unable to take additional coal. The capacity of this pile is about 280 000 tonnes of coal.

Two stackers have been provided in the blending area. During normal operation, one stacker would be in full time use at the average-grade blending pile. The second stacker would be in operation from time to time to stockpile high-quality, low-sulphur coal.

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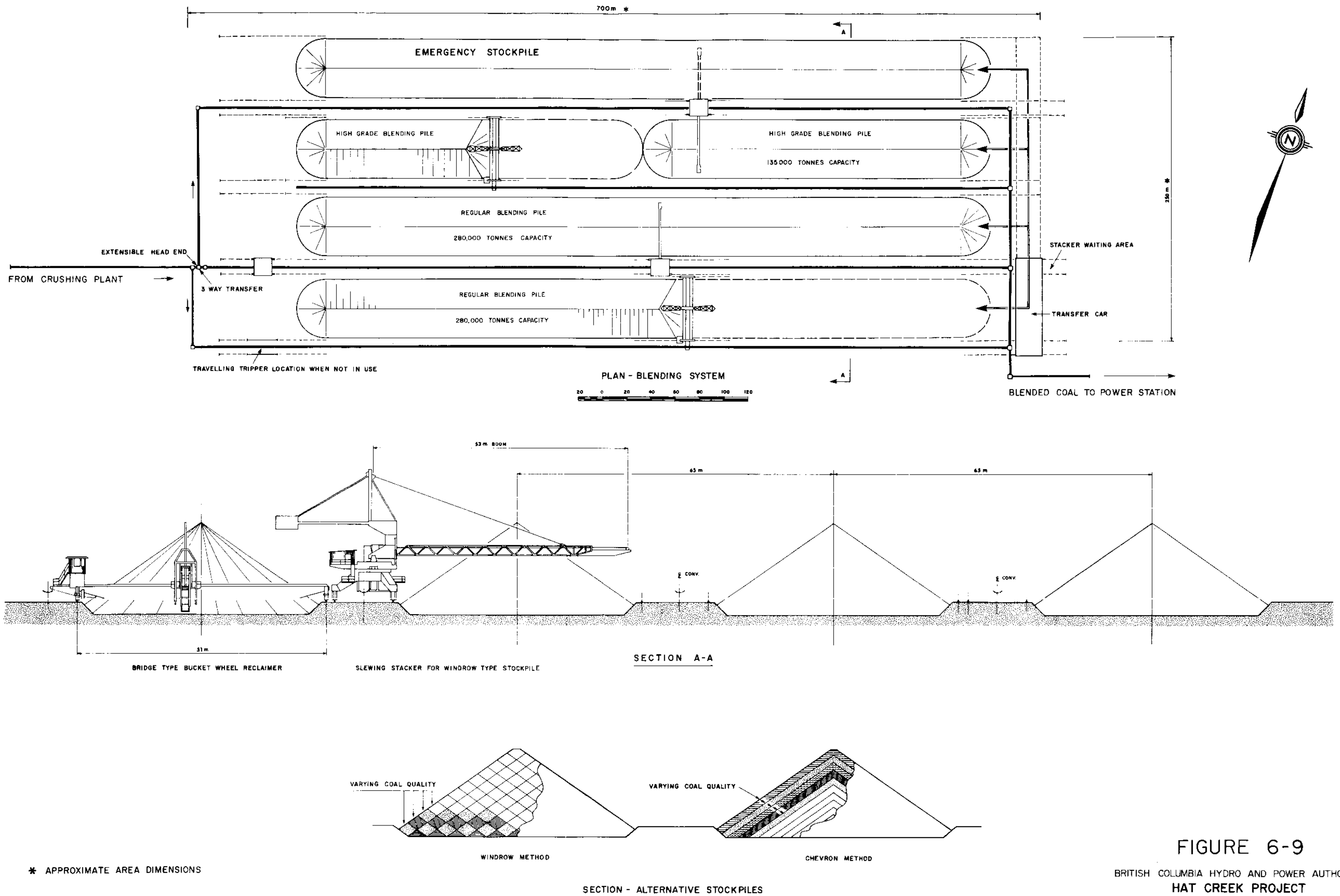

FIGURE 6-9

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
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MINING FEASIBILITY REPORT

GENERAL ARRANGEMENT
COAL BLENDING SYSTEM

Similary, two reclaimers have been recommended for the blending operation, one in the average-grade areas and the other in the high-grade and emergency areas.

682 FACTORS AFFECTING THE RECOMMENDATION

682.1 Location

The blending facility is integrated within the mine operation rather than at the generating station since:

(a) the mine has responsibility for quality control of boiler feed coal; and

(b) it would provide a surge capacity in the flow of coal from the mine in case of breakdown of the overland conveyor.

682.2 Blending Requirements

The mine produces coal that varies in calorific value from about 4000 to over 10,000 Btu/lb (dry basis).

The target average calorific value to be delivered to the thermal plant has been estimated at the mine average production grade of 7327 Btu/lb (dry basis), and ranging from 7000 Btu/lb to 7800 Btu/lb providing reasonable notice is given to the generating station. Further, the recommended hourly variation in fuel calorific value should not exceed approximately $\pm$150 Btu/lb (refer Section 4).

Two particular grade problems have been identified: firstly, production of coal below the lower boundary of the range could occur from time to time due to production scheduling problems as discussed in Section 455. Secondly, the sulphur content of the coal may have to be decreased under certain climatic conditions.

The task of the blending systems is therefore twofold:

(a) prepare the short-term mine feed variations into a mix which closely achieves the target average calorific value of 7327 Btu/lb with a maximum hourly variation of ±150 Btu/lb; and

(b) stockpile sufficient high-grade, low-sulphur coal to allow grade adjustment by means of the simultaneous reclamation of average-grade and high-grade, low-sulphur coal.

682.3 Blending Areas Layout

The blending system is shown on Figure 6-9. The blending area contains four rows of piles which serve the functions described previously.

The configuration of the piles is influenced by the space limitations at the site and the selection of 280 000 tonnes as the size of a blending unit for average-grade fuel. Selection of the configuration of the blending piles must also take into consideration:

(a) the stockpile efficiency which is determined by the ratio of the stockpile length and width; and

(b) size and availability of the required stackers and reclaimers.

Access roads are provided for maintenance of the equipment and clean-up along the conveyors.

682.4 Equipment Selection

Blending equipment selection is influenced primarily by the method in which coal is deposited and reclaimed. Deposition of the material can be done either in windrows or by the chevron method of layering the material.

In the first method a stacker lays a number of rows, side by side, by travelling along the stockpile and after each pass the slewing boom is moved to a position beside the previous row. In the chevron method, the stacker boom deposits the material in the centre of the pile and each subsequent layer is placed on top of the previous one. Both methods are illustrated on Figure 6-9.

Selection of the most suitable method is based on differences in operation and equipment.

The boom length of the stacker utilized in the chevron method can be shorter and therefore requires a lighter machine than the stacker used in the windrow method. The windrow method gives a more accurate result, particularly where the particle size distribution favours segregation of the larger particles at the bottom of the stockpile.

Dusting is normally considered to be more of a problem with the chevron type operation than the windrow method. This is due to the fact that in the chevron method, material is in motion over a much larger area when being deposited and the surfaces of the stockpile are continuously being renewed with a new layer of coal.

For these reasons the preliminary selection of the windrow method of stacking was made and forms the basis for the estimate. Two stackers are recommended for installation in the blending system. Each stacker would be automatically controlled and programmed to form a windrow blending pile. Travelling speed is interlocked with the belt weighing scale so that a reasonably uniform windrow cross-section may be achieved. In case of a breakdown of one stacker, the remaining stacker would be able to provide up to 100% capability, either by relocation of the stacker or by using the emergency stockpile. This would be achieved by means of a rail mounted transfer car which is able to transfer the stacker as well as the reclaimer from one pile to another.

Major specifications of the windrow stacking equipment are:

stacker capacity	3200-tonnes per hour
boom length	53 metres
slewing arc	200° minimum
lifting height	18 metres
travelling speed	variable from 3 to 30-metres per minute

Commonly used reclaiming machines are the bridge mounted bucket wheel reclaimer and the drum reclaimer. Due to size limitations and cost, a preliminary selection of the bridge mounted bucket wheel reclaimer with one bucket wheel was made.

Two of these reclaimers have been recommended for the blending operation, one in the average-grade stockpiles and the other in the high-grade and emergency stockpiles.

The reclaimer in the high-grade stockpiles should be reversible as it would be operating between two stockpiles in the same row. Similarly, in order to permit interchangeability with the other reclaimer and to allow for a greater number of average-grade stockpiles if required, the second reclaimer should be made reversible.

The most important specifications of the reclaimers are:

- track mounted with reversible operation
- bridge span between tracks - 51 metres
- number of bucket wheels - 1 per reclaimer
- capacity - variable from 3200 tonnes per hour
  to 500 tonnes per hour

In order to permit interchangeability of each of the two stackers and each of the two reclaimers, the design of these machines should be identical. During normal operational practice both the stacker and the reclaimer must be transferred from one stockpile to another.

683 BLENDING SIMULATION STUDIES

683.1 Study Procedure and Conclusions

The approach to the quality control within the blending area is directly related to that achieved in the mining operation. For this reason, two types of simulation studies were conducted:

(a) an estimation of the hourly variation in run-of-mine coal production that would normally be delivered to the blending area; and

(b) an estimation of the shortest period of time that the mine can produce an average grade of coal within the calorific ranges acceptable to the generating station. This was carried out in four steps:

- five year average
- yearly average
- monthly average
- weekly average

No significant problems were encountered in achieving the desired averages by normal grade control methods for the 5-year periods through to the monthly averages. Some problems were encountered in trying to achieve the weekly average. However, it was felt that fill-in drilling and longer range, grade control planning would overcome these problems. It was subsequently recommended that the size of stockpiles to be considered for the blending study should equal approximately one to two weeks of plant feed requirement.

The blending study was carried out to determine what blending accuracy is practical and also to give some indication of the size ranges for stockpiles.

Using the results of the first simulation study involving the hourly grade variation over a one week period as discussed in Section 454, two methods of blending were investigated. Deviations from average quality in the blended stockpile was calculated in both cases.

Method I All coal is transported out of the pit on one mine conveyor and immediately stacked out in the normal manner for blending in one stockpile.

Method II All coal is transported out of the pit in one conveyor. A high-grade stockpile would be maintained and replenished by mining high grade D-Zone coal. Coal from this stockpile is reclaimed and mixed with coal from an average grade stockpile when upward adjustments in fuel coal quality are required.

The difference between the two methods is that in Method I the weekly average grade must always be achieved by the mine production schedule whereas in Method II the average grade may be improved by the addition of high-grade coal from the reserve high-quality stockpile.

Another consideration in the selection of the blending method is the requirement to be able to deliver low-sulphur coal to the generating station for short periods of time. As described in Section 685, this material may be obtained either from the high-grade stockpile or alternatively from the mine.

Blending Method II was therefore accepted as the preferred system. This, together with the layout and equipment recommended, overcomes the need to make specific recommendations as to minimum size of the individual stockpiles for the current study. The blending simulation studies indicate that based on the information available, the required quality control may be achieved with stockpiles having a capacity of one week's feed requirement. This is discussed in more detail in Section 683.2.

683.2 Blending Study

Blending simulation studies were performed in order to establish a practical number of layers required in the blending pile to meet the specified maximum deviation of ±150 Btu/lb.

However, it should be noted that after the coal leaves the blending facility, a substantial amount of additional mixing takes place in the materials handling system of the generating station.

The first study calculated the coal quality deviation per cut across the stockpile section. It was assumed that all coal from the three shovels employed in the production simulation study was deposited in one stockpile.

The following values for average grades, standard deviations, and the ratios of correlation time to production time were calculated.

Shovel No.	No. of $500m^3$ Blocks	Average Btu/lb	Standard Deviation	Ratio of Correlation Time to Production Time
1	126	5810	1480	0.036
2	103	6010	1482	0.500
3	108	8646	1289	0.133

Total weighted average calorific value is 6780 Btu/lb. Using these data, it was calculated that in a stockpile with 50 layers a blended product with a deviation of 75 Btu/lb above or below average value could be obtained.

However, the size of the pile and normal stacker travelling speed should permit the use of a much higher number of layers in the blending pile during actual operations. Therefore, using 100 layers in the stockpile, it was calculated that the deviation would be reduced to 32 Btu/lb on the average of 6780 Btu/lb. This gives an indication of the improvement which may be obtained by increasing the number of layers in the blending pile.

In order to evaluate the effect of mixing high-grade coal with average-grade coal, the production of Shovel I, which was operating in lower-grade coal, was assumed to be blended separately. The average calorific value of the low-quality coal was included in the calculation giving a result of 5810 Btu/lb with a deviation of 89 Btu/lb using 100 layers.

For the high-grade blending pile, a value of 9500 Btu/lb with a deviation of 30 Btu/lb was assumed. Coal from both stockpiles was then mixed in the proportions necessary to obtain an average value of 7500 Btu/lb, and the deviation was calculated to be 50 Btu/lb.

These results indicate that the blending system should be capable of delivering coal to the generating station within the limits of $\pm$150 Btu/lb while using between 100 and 200 layers in the stockpile.

684 SPONTANEOUS COMBUSTION IN STOCKPILES

Test piles of loose and compacted coal were constructed and monitored during the summer of 1977 to determine temperature change with time and the potential for spontaneous combustion.

Fires began in loose, crushed coal and uncrushed low-grade coal piles after 14-28 days and 50-70 days of exposure, respectively. Fires generally occurred near the base of piles in areas of loose coal usually after the average temperature had risen to 60-70°C.

As a result, certain provisions for coal handling to reduce the risk of spontaneous combustion have been included in the mine planning.

In the event coal must be stored for long periods, it is recommended that compaction of the piles either in total or in part be carried out as required. It is anticipated that for most of the time the two average-grade stockpiles would probably not require compaction, as the pile capacity of approximately 280 000 tonnes represents about 1.3 weeks of average coal consumption at the thermal plant.

In the case that an uncompacted stockpile cannot be emptied in time and combustion starts, it should be possible to turn over the pile by placing the stacker behind the reclaimer and operating both machines simultaneously on the same conveyor.

The material would then be reclaimed from the stockpile and stacked out in an empty space in the same stockpile area. This procedure requires that empty space is, or be made available at the east end of the blending area and that the design of the reclaimer permits the passage of the stacker.

The high-grade stockpiles would possibly be left untouched for extended periods of time and therefore the costs of compaction have been included in the estimates.

The emergency surge stockpile area should normally be kept empty. When the requirement arises to use this area it must then be determined if it is avisable to compact the coal or stockpile it loose.

685 LOW-SULPHUR COAL HANDLING

Low-sulphur coal could be supplied to the generating station as required. Selective mining and scheduling of the shovel operations should permit the mining of adequate supplies of low-sulphur coal. The transportation alternatives through the material handling system are:

(a) It should be possible for the recommended high-grade stockpile to contain sufficient low-sulphur D-Zone coal at all times since about 42% of the mine production consists of this type of coal. During the periods of time when low-sulphur coal is required this supply could be either sent directly to the generating station or mixed in with the average coal as required.

(b) In the event that the supply of low-sulphur coal contained in the high-grade stockpile is not adequate, either in quality or quantity, coal could be sent directly from the mine to the generating station by using one of the conveyors in the stockpile area to bypass the blending system.

6.9 SUPPORT FUNCTIONS

This section of the report outlines the principal mine support operating activities and the support mobile equipment recommended. General information regarding fleet size and job-function for each type of equipment, as shown on Table 6-12, is also included. Where applicable, the reasons for selection of specific equipment are presented.

691 SITE PREPARATION

Site preparation involves land clearing and topsoil removal for both the mine and dump areas, and also bench pioneering for the mine and waste dump base preparation for the waste dumps.

Land Clearing Land clearing would be carried out on a total of 1260 hectares of area in both the mine and waste dumps. This activity would be divided into three categories: clear-cutting, selective logging of merchantable timber, and clearing of scrub. The first two activities would be carried out during the pre-production period, probably by contract. The third activity would be carried out periodically in advance of the topsoil removal as mining progressed.

Topsoil Removal Removal of topsoil from the waste dump and mine area would occur progressively with the advance of mining. It would be carried out by 24 cubic metre scrapers with D-9 dozer backup. Topsoil stockpile locations have been recommended which will minimize scraper haul length. Scraper productivity was averaged at 57 bank cubic metres per operating hour.

Bench Pioneering The bench pioneering operation is intermittent and involves cut and fill work at designated bench elevations. This would provide a sufficient number of ramps and level operating space for subsequent shovel/truck excavation. The work will be performed by a fleet of 24 cubic metre scrapers with backup dozers and 32-tonne rear dump trucks loaded by 5.4 cubic metre front-end loaders also with backup dozers.

TABLE 6-12

Equipment Recommended and Job Functions

Hat Creek Project Mining Feasibility Report 1978

Item	Description	Fleet Size
24 LCM Scraper		
Capacity	12 BCM	
Production years	mine life	6
Job functions	- topsoil removal and replacement - bench pioneering - road construction and maintenance - dump buffer zone placement	
D-9 Dozers		
Production years	pre-production . . . . production	5 8
Job functions	- backup to scraper activities - road construction - bench pioneering - pit cleanup - 11.5 m^3 front-end loader support	
D-8 Dozers		
Production years	pre-production . . . . 1 to 15. . . . . 16 to 35	4 6 8
Job functions	- road construction and maintenance - dump activities: leveling of spoil moving conveyors causeway construction buffer zone leveling - 5.4 m^3 front-end loader support - topsoil dump activity - low-grade coal stockpile	
Wheeled Dozer		
Production years	mine life	2
Job functions	- shovel and bench cleanup - road maintenance - snow removal	

(Continued)

TABLE 6-12 (continued)

Item	Description	Fleet Size
16-G Grader		
Production years	pre-production to year 3	4
	4 to 15	5
	16 to 35	6
Job Functions	- road construction and maintenance - grading on dumps	
11.5 m^3 Front-end Loaders		
Production years	Year -2	1
	Year -1 to 35	2
Job functions	- shovel backup - pit cleanup	
5.4 m^3 Front-end Loaders		
Production years	pre-production	2
	Years 1 to 35	3
Job functions	- bench pioneering - low-grade coal stockpile - pit cleanup - materials to dump - dump activities - road construction and maintenance	
32-tonne Dump Truck		
Capacity	15.3 BCM	
Production years	pre-production to Year 4	7
	Years 5 to 35	10
Job functions	- bench pioneering - road construction - materials trucked to dump - materials trucked on dump - low-grade coal stockpile - pit cleanup	
1 m^3 Backhoe		
Production years	mine life	1
Job functions	- construction and maintenance of ditches - construction and maintenance of dewatering sumps	

(Continued)

TABLE 6-12 (continued)

Item	Description	Fleet Size
Gradall		
Production years	mine life	1
Job functions	- road maintenance - ditch maintenance	
Drills		
Type: Auger	mine life	1
Percussion	mine life	1
Rotary	mine life	1
Job functions	- coal drilling - baked zone drilling - hardpan and consolidated till drilling - drilling of oversized boulders	
Additional Miscellaneous Equipment		
blasting truck		1
water trucks		3
mobile crusher		1
wheeled compactor		2
vibrating compactor		2
diesel pumps		10
CaCl spreader		1
lo-boy tractor		1
hi-boy tractor		1

The scrapers would relocate approximately 312 bank cubic metres of material per hour an average distance of about 0.75 kilometres for an estimated total of about 300 fleet operating hours per year. The 32-tonne trucks move approximately 88 bank cubic metres of material per hour, giving a total of about 1755 hours per year.

The dozers would be operated for approximately 14,780 additional hours per year in carrying out cut and fill operations.

Base Preparations During Year -2 the base for the Houth Meadows embankment, dump area, and semi-permanent in-dump right-of-way would be established by hauling approximately 3.6 million bank cubic metres of construction grade material directly from the mine. This would be accomplished by utilization of the large mine truck fleet and will permit spreaders to travel on a reasonably level surface during initial waste operations. Similar operations for the Medicine Creek disposal area would be carried out during Year 16.

The fleet of 32-tonne trucks would be too small to take on this extra task, the duration of which would not warrant purchasing significant additional trucks of this size. Therefore, estimates provide for contracting haulage of approximately 2.5 bank cubic metres in Year 16 and approximately 2.0 million bank cubic metres in Year 17.

692 DRILLING AND BLASTING

The 16.8 cubic metre shovels should be able to excavate the majority of materials mined without ground preparation by blasting.

Some potential problem materials have however been identified. These include hardpan clay, D-Zone coal, baked zone, and large chunks of petrified wood. To improve excavating productivity, the following quantities of these materials have been estimated as warranting light drilling and blasting:

Hard Pan Clay	22 million cubic metres
Baked Zone	9 million cubic metres
D-Zone Coal	105 million cubic metres
Total	136 million cubic metres

Calculations to develop drill productivities and fleet size were based on the following criteria:

Drill pattern size.........7.5 metres x 7.5 metres

Hole depth.................17.5 metres

Bench height...............15.0 metres

Drilling rate..............25 metres/operating hour

Blasting powder factor.....0.42 pounds/cubic metre

The following example shows the development of drilling requirements for coal during Year 3.

Type of drill......Truck mounted auger

Volume drilled.....2.475×10^6 bank cubic metres

$$\text{Number of drill holes/year} = \frac{\text{Volume drilled}}{\text{Volume per hole}}$$

$$= \frac{2,475,000}{7.5 \times 7.5 \times 15}$$

$$= 2934 \text{ holes}$$

$$\text{Drill operating hours} = \frac{\text{Number of holes}}{\text{Drilling rate per operating hour}}$$

$$= \frac{2934 \times 17.5}{25}$$

$$= 2054 \text{ hours}$$

The drilling hours indicate that one auger drill scheduled for one full shift would have the capability of drilling the harder coal material.

Similar calculations were carried out for the hard pan clay and baked zone materials for rotary and rotary percussion type drills. The results indicate the one drill of each type has ample capacity to accommodate the quantities of materials to be drilled and blasted.

693 SHOVEL BACKUP EQUIPMENT

Shovels are supported by two 11.5 cubic metre front-end loaders and dozers working as an equipment team. Both units are required during pre-production years and utilized throughout the project life.

These units are intended to replace the shovels for short periods of mechanical failure or while the shovels are moving from one bench to another. Figure 6-2 confirms that the typical 11.5 cubic metre front-end loader can load either of the recommended 136-tonne waste trucks or the 109-tonne coal trucks. These loaders will not be utilized full time in support of the shovels and will be available to perform numerous other tasks such as pit cleanup, roadway construction and relocation, rehandling of boulders, and snow removal.

694 PIT CLEANUP

Equipment recommended for cleanup includes: 11.5 cubic metre and 5.4 cubic metre front-end loaders, large track dozers, rubber-tired dozers, and the 32-tonne haulage trucks. For sizeable cleanup jobs such as large slumps, the principal haulage trucks (109 or 136-tonne) could be used depending whether the slump is coal or waste.

The cleanup functions include: periodic spillage in the shovel loading area, smoothing out and filling of irregular surfaces around the shovel loading area, ground leveling in advance of bench road construction, and cleanup of slumps that are potential hazards.

Cleanup estimates for the combined activities listed below are based on the annual equipment usages throughout the project life:

- 11.5 cubic metre front-end loaders - one loader shift per day throughout the year = 365 loader shifts/year loading into 109 or 136-tonne class haulers.

- 5.4 cubic metre front-end loaders - one half of loader shift per day throughout the year = 183 loader shifts/ year loading into 32-tonne general duty haulers.

- additional dozer work carried out in isolation will be required and includes 5 large dozer shifts per day or 1825 shifts per year and 2190 rubber-tired tractor shifts per year.

In the Hat Creek operation, rocks that consistently cut haulage truck tires are not expected. Shovel cleanup should therefore be reduced when compared to some other operations, but work required to maintain level bench surfaces may be comparatively greater. In the softer materials, a gouging out of the bench floor can be anticipated with the attendant problems of maintaining flat shovel gradients. The cost provisions for shovel cleanup are believed to be adequate to cope with problems of this nature.

695 ROAD CONSTRUCTION AND MAINTENANCE

The road construction activity will utilize: 24 cubic metre scrapers, 35-tonne dump trucks, large dozers and graders, water wagons, compactors, 250-tonne-per-hour portable rock crusher and 5.4 cubic metre front-end loaders with backup dozers. Road maintenance would require similar equipment and also gradalls and wheel dozers.

Projected roads have been classified according to service (vehicle class) and usage intensity. Each of three road categories has been assigned a sub-base cross-section that it considered adequate for expected Hat Creek conditions. Construction material for the sub-base and road surfacing would be provided by selecting materials from areas scheduled to be stripped during the life of the project.

Pit Road Classification

Road Duty	Class	Built-up Width	Depth of Fill
Mine Haulage	1	30 metres	1 metre
Service Road	2	20 metres	0.5 metre
Light Service Road	3	10 metres	0.5 metre

The following list identifies mine road functions by class.

Class 1	Class 2	Class 3
Main Haul Roads	Mine Service Road	Shiftable Conveyor Access
Temporary Haul Roads	Safety Berm Access	Dewatering and Ditch Access
	Pit Ring Road	Dump Perimeter Access
	Dump Access	
	Overland Conveyor Access	

Table 6-13 lists the estimated length of each class of road construction required annually. As benches advance away from the pit centre and as shiftable conveyors are moved, road-building for these two functions is repetitive. On benches, it is estimated that road sub-base materials can be recovered and relocated prior to becoming contaminated with clay-like foundation materials.

The construction method for roads involves dozers to cut and fill where required and rough-level the foundation. Scrapers, aided by push dozers, would subsequently pick up run-of-mine gravels - mainly from the east side of valley - in order to build up the lower portion of the sub-base. The upper portion of the sub-base, including the running surface, would be crushed gravel or, when available, baked zone materials. The gravel crusher would be located on a gravel bench and fed by front-end loaders. It would convey crushed material to a pile for scraper pick-up. Graders and vibrating compactors would complete the final surface. Provision has been made for water trucks for both the construction phase described above and for subsequent road maintenance practice.

TABLE 6-13

Annual Road Construction By Class In Kilometres

Hat Creek Project Mining Feasibility Project 1978

Year	Class I	Class II	Class III	Total
-2	4.9	10.0	6.0	20.9
-1	6.0	10.1	11.0	27.1
1	13.6	3.6	12.0	29.2
2	13.2	3.5	12.0	28.7
3	14.2	3.4	11.0	28.6
4	16.3	5.3	11.0	32.6
5	16.9	4.9	11.0	32.8
6	15.8	4.8	11.0	31.6
7	15.8	2.7	12.0	30.5
8	15.8	2.5	12.0	30.3
9	15.8	2.5	12.0	30.3
10	15.9	2.5	12.0	30.4
11	16.7	2.5	11.0	30.2
12	16.7	2.9	11.0	30.6
13	16.7	3.5	11.0	31.2
14	16.7	5.0	11.0	32.7
15	19.1	5.0	11.0	35.1
16	19.0	6.4	16.0	41.4
17	19.0	2.5	16.0	37.5
18	19.0	2.4	16.0	37.4
19	19.0	2.4	16.0	37.4
20	19.0	2.4	16.0	37.4
21	19.0	2.4	16.0	37.4
22	20.9	2.4	16.0	39.3
23	20.9	2.7	15.0	28.6
24	20.9	2.8	15.0	28.7
25	20.9	2.8	15.0	38.7
26	20.9	2.7	15.0	38.6
27	9.6	2.8	6.0	18.4
28	9.6	2.4	6.0	18.0
29	9.6	2.4	6.0	18.0
30	9.6	2.0	6.0	17.6
31	9.6	2.0	6.0	17.6
32	9.6	2.0	6.0	17.6
33	9.6	2.0	6.0	17.6
34	9.6	2.0	6.0	17.6
35	9.6	2.0	6.0	17.6
TOTAL	555.0	126.2	414.0	1095.2

The estimate of quantities required for annual road construction materials is 250 000 bank cubic metres. This also provides for resurfacing of relocated roads. The crusher is of a size that provides 250 tonnes per hour. Estimates for out-of-pit perimeter roads, access roads, and service roads include an allowance for culverts, drains, and drainage ditches.

Road maintenance consists of grading, surface replacement, application of chemical binders, watering during summer months, and snow ploughing and gravelling in winter. Provision has been made for ditching and trimming of potential sloughs.

The periodic excavation of inter-bench ramps is treated as an excavation function. The construction and maintenance of roadways on those ramps is a road construction function.

Snow removal will also be required for four to five months of the year. The snow year will persist longer on the upper benches and on the spoil piles than in the pit. Provision has also been made to allow for snow clearing in the administration and service areas of the mine.

696 LOW-GRADE COAL STOCKPILE OPERATIONS

From Year -1 to Year 3 inclusive low-grade coal is mined intermittently and in relatively small quantities. This material is trucked direct from the mine face to the low-grade stockpile by 109 tonne trucks where it is placed and compacted.

From Year 4 onwards the low-grade coal is crushed and delivered intermittently by a conveyor system to a bin suitable for direct loading into 32-tonne and dump trucks. With the assistance of bulldozers, the 32-tonne trucks will place the material in a stockpile located close to the Houth Meadows waste embankment. In order to reduce the possibility of spontaneous combustion, self-propelled pneumatic type compactors will compact the material as it is placed in the stockpile.

697 WASTE DUMP OPERATIONS

The waste disposal conveying system requires support from the mobile equipment fleet in several areas. These have been identified as: the direct trucking of construction grade materials from the mine to build the conveyor right-of-way sub-bases; the trucking of waste materials from the end of the spreader operation to areas around the periphery of the dump which cannot be reached by the spreader; and dozing/grading activities associated with the shiftable conveyor/spreader operation.

697.1 Direct Trucking Function Direct trucking from the mine of construction grade materials will be carried out by a fleet of 32-tonne trucks assisted by 5.4 cubic metre front-end loaders and D-8 dozers.

The combined total quantities hauled over the life of the mine for constructing the conveyor right-of-way sub-bases are estimated at 13.49 million bank cubic metres. The annual quantities to be excavated and transported range between 60 and 100 000 bank cubic metres. The average haulage distance, one-way only for the trucks, was estimated to range from 3 kilometres during pre-production to 7 kilometres during Year 15 and onward.

697.2 Trucking to Inaccessible Areas Situations occur during the waste dump development where the shiftable conveyor has inadequate length for the spreader to reach the edge of the dump. Rather than interrupt the total waste operation by extending the shiftable conveyor, trucks would be utilized to place materials in these areas. These materials are loaded by 5.4 cubic metre front-end loaders into 32-tonne trucks from windrows of waste material placed by the spreader at a location convenient to both operations.

The total quantity of material hauled by this means to the periphery of the waste dumps of the 35 year life of the mine is estimated at 13.72 million bank cubic metres. Annual amounts range between 150 000 and 500 000 bank cubic metres at an average haulage distance of 600 metres.

697.3 Dozer and Grader Activities Dozers would be utilized on the dump surfaces to carry out the following activities:

- relocation of shiftable conveyors

- leveling of spoil piles

- preparation of conveyor right-of-way
- road construction and maintenance
- leveling of truck dumped materials
- assist front-end loader operations

Two dozers, having an effective utilization of 57%, were assumed to be capable of handling these activities on the Houth Meadows dump up to Year 15. During Year 15 a third dozer would be introduced with the start-up of waste disposal at Medicine Creek dump.

Graders are utilized on the waste dump to carry out the following activities:

- leveling for shiftable conveyor moves
- conveyor right-of-way construction
- road construction in locations adjacent to conveyor right-of-way, for dump access, the the dump perimeter, and service access parallel to the shiftable conveyors.

One grader, having an effective utilization of 68% for one shift per day, was assumed to be capable of handling the activities listed for Houth Meadows dump. A second grader was introduced during Year 16 with the startup of waste disposal at Medicine Creek dump.

698 RECLAMATION

The mine equipment will be involved with various reclamation activities through the life of the mine. These are listed as follows:

(a) Initial topsoil removal from disturbance areas.

(b) Final dump surface leveling and regrading using dozers assigned to the waste dump. This activity starts at approximately Years 14 or 15.

(c) The placing of buffer zone material on waste dumps and low-grade coal stockpiles would be carried out mainly by dozers, scrapers and graders. Once the final dump surface has been prepared, the fleet of 24 cubic metre class scrapers load, transport, and deposit the buffer materials. The buffer material is originally from the mine as part of its waste production to the dumps and is placed in special stockpiles by the spreader operation on the final dump surface.

(d) Topsoil may then be reclaimed from topsoil stockpile locations by the fleet of 24 cubic metre class scrapers and placed as a final layer on the dump surface. Final contouring is carried out by the mine graders.

Up to the end of Year 35 the mine equipment will have placed approximately 3.55 million bank cubic metres of buffer zone and approximately 1.4 million cubic metres of topsoil on the Houth Meadows waste dump. It is estimated that 1.85 and 0.64 million bank cubic metres, respectively, of these two materials will have been placed on the Medicine Creek dump. The balance of these materials requiring placement on the dumps beyond Year 35 may be carried out by others.

6.10 DEWATERING

The dewatering operation takes two forms; the removal of runoff water by sumps, surface pumps, and ditching, and the operation of deep wells to aid slope stability. The equipment includes small surface pumps, large sump pumps, booster pumps, a large number of small deep well pumps, backhoes, gradall, water trucks, and a pump service truck.

The dewatering activity is expected to yield insignificant quantities of water due to the low permeability of Hat Creek rocks and to the arid climate. Ditch maintenance in-pit is not expected to be a significant task for the same reasons. Maintenance of interceptors and other ditches beyond the rim of the pit may require seasonal attention during spring runoff and intermittent rainy periods. Any water that is collected during the summer months will be utilized for road maintenance and dust control.

The concept for groundwater withdrawal is a staged program.

(a) Starting in Year -5 two systems of wells should be drilled and operated: 25 wells in selected locations inside the pit perimeter at 50-metre depths, and 10 to 15 regional or extra-perimeter wells averaging 300 metres in depth. All well holes should be drilled 6" diameter and cased. Pumps should be electrical driven. The concept is to collect deep well water into boxes and sumps via a series of headers.

Wells that survive the advancing mine excavation will likely require replacement on the average of each 10 years due to silting. Replacement of both excavated and silted wells should be stepped outwards towards the second stage location for wells. Monitoring holes may also be required.

(b) Starting in production Year 10 and completing in Year 15, a final set of wells should be established beyond the projected perimeter of the 35-year pit. By Year 15 this system should increase to 75 pairs of wells - one shallow at

50 metres and one deep averaging 300 metres. By this point in time up to 100 monitoring holes should also be in use. Cost estimates provide for the installation of piezometers and a monitoring crew.

This is a tentative programm subject to further definition pending further geotechnical investigation.

Mine seepage and runoff water may be collected into a series of bench collection boxes or sumps. Water in excess of dust control requirements should be lifted out of the pit in a cascading system by staged electric pumps into the water collection lagoon east of the Houth Meadows dump.

SECTION 7

ALTERNATIVE MINING SYSTEMS

CONSIDERED

7.1 INTRODUCTION

In the process of recommending a mining system for the Hat Creek project three levels of evaluation were carried out. The first was an initial assessment of six alternative mining systems on the basis of practical operating features. This lead to the second level where the three preferred systems were examined for order-of-magnitude comparison of cost and operating capabilities. In the third level, detailed evaluations were carried out on two systems, from which one was recommended as most suitable for the Hat Creek project.

7.2 INITIAL ASSESSMENT OF SIX ALTERNATIVE SYSTEMS

The six alternative mining systems initially evaluated were:

- I - shovel/truck
- II - shovel/truck/conveyor
- III - shovel/conveyor
- IV - bucket wheel excavator/conveyor
- V - continuous excavator/truck/or conveyor
- VI - dragline

These systems are discussed briefly as follows:

721 SHOVEL/TRUCK

The capacities of shovels and trucks most commonly used in the British Columbia mining industry are 11.5 cubic metre electric shovels, and 109-tonne to 154-tonne capacity electric wheel drive end-dump trucks. Based on the desirability of utilizing proven equipment, the initial evaluation utilized 11.8 cubic metre shovels and 109-tonne capacity trucks in its assessment. The coal was transported by truck to an out-of-pit primary crushing, screening, beneficiation,and blending facility. The waste was trucked directly to disposal areas or low-grade coal stockpiles.

Some of the more significant advantages and disadvantages of this system were considered to be:

Advantages
- proven and accepted type of equipment
- flexibility to adapt to changes in mining conditions
- long equipment life of 20 years for shovels and 10 years for trucks

Disadvantages - impractical to assume that these large trucks can travel over the waste dumps even with extensive sub-grade road construction

- long haul distances will require many trucks with resulting higher capital costs

- escalating costs of diesel fuel are becoming a very important factor in system selection

722 SHOVEL/TRUCK/CONVEYOR

The concept for this system is similar to shovel/truck except that the large truck fleets are substantially reduced by truck hauling materials to in-pit unloading stations and by conveyor(s) to out-of-pit destinations. To allow the simultaneous handling of coal and waste and to provide protection against major conveyor breakdowns three parallel conveyor lines are provided. Each of the conveyors is provided with the flexibility of transferring materials to any one of the ongoing conveyors. The maximum production of the mine can be handled by any two conveyors.

In addition to the features of the shovel/truck previously described, the following advantages and disadvantages were found to apply:

Advantages - substantial reduction in the number of large trucks required

- eliminates the need for large trucks to travel on waste dumps

- should provide an operating cost advantage over an all-truck system

Disadvantages - will reduce the flexibility of the in-pit operations to adapt to changes in mining conditions

- potential material handling problems in the conveying of wet materials under winter conditions

- the initial capital cost of conveying systems should be higher than for truck systems

723 SHOVEL/CONVEYOR

In this system the large trucks are eliminated completely in the pit operation. Shovels of 11.5 cubic metre capacity excavate either coal and/or waste materials and load directly onto mobile hoppers located adjacent to the shovel. The mobile hoppers are equipped with sizing facilities in order to prepare the materials for loading directly onto one or more in-line, self-propelled short bridge conveyors. These units in turn transfer their materials onto longer shiftable face conveyors which feed into the permanent conveyors described under the shovel/truck/conveyor system. Under this system materials are transported by conveyor directly from the shovel excavator to the out-of-pit final destinations. Some of the significant advantages and disadvantages of this system were:

Advantages

- should have a significant operating cost advantage over both previous systems
- eliminates the difficulties associated with operating truck fleets in the open pit

Disadvantages

- higher initial capital costs than previous systems
- mobile loading hoppers may reduce productivity of shovel excavators
- mining systems in which materials are transported directly from shovel excavators on a multi-bench development to a conveying system have not been utilized in British Columbia to date and are found only infrequently on a world scale
- restricted flexibility in adapting to changes in mining conditions

724 BUCKET WHEEL EXCAVATOR/CONVEYOR

Bucket wheel excavators are used to mine either coal and/or waste and to transfer these materials directly onto shiftable face conveyors which in turn deliver the materials to a system of conveyors for the transportation out of the pit as described in the previous two systems.

The more important advantages and disadvantages of the system were considered to be:

Advantages

- there should be a potential for significant operating cost advantage over other systems
- no major haulroad systems are required to accommodate bucket wheels and only a few service and access roads would be constructed
- bucket wheel excavators have the ability to selectively remove small bands of waste partings
- bucket wheels have a longer service life than electric shovels

Disadvantages

- it must be recognized as a permanent system that does not easily accommodate changing mine plans or changing geological conditions
- high initial capital cost compared to all other systems evaluated
- bucket wheel excavators, while proven in Germany, are relatively unknown in Canada
- generally require wide, long bench methods of mine development and significant pre-production excavation to establish working areas
- require long lead times from purchase to commissioning and long break-in time to reach satisfactory productivity

725 CONTINUOUS EXCAVATOR/TRUCK OR CONVEYOR

This system utilizes a continuous excavator such as the Holland loader or plow that loads directly into 109-tonne end-dump trucks. The major advantage was considered to be the potential for lower initial capital costs of the loading equipment. The significant disadvantage was that the unit was designed to operate on shallow bench heights of 3 to 5 metres. This was considered impractical for the Hat Creek open pit and was, therefore, only briefly considered.

726 DRAGLINE

The configuration of the Hat Creek deposit was viewed as unsuitable for dragline operations since:

(a) from large capacity draglines, it would be impractical to dump materials directly into mobile transfer hoppers for subsequent transfer to conveyor systems and double handling of materials would likely be required. This would require additional reclaim equipment such as shovels or bucket wheel reclaimers and result in higher capital and operating costs.

(b) coal quality control would be extremely difficult to maintain

(c) the depth of the pit requires equipment systems capable of moving easily in a vertical direction from one work location to another

727 CONCLUSIONS

After determining the applicability of the six mining systems to conditions at the Hat Creek deposit and weighing the advantages and disadvantages of each system, three systems were selected for further study and order-of-magnitude comparisons and are listed below:

System II - Shovel/truck/conveyor

System III- Shovel/conveyor

System IV - Bucket wheel excavator/conveyor

In order that the system providing the lowest estimated cost could be determined, the three systems selected were studied in detail with regard to a comparison of the pit design, and to the type, size, and suitability of equipment in handling the various mining materials. A more detailed description of these studies is presented in the following section.

7.3 ORDER-OF-MAGNITUDE COMPARISONS

731 INTRODUCTION

It was decided that engineering for each of the three selected systems would be carried to the conceptual stage, thus allowing order-of-magnitude cost estimates to be prepared.

The development of the mine for the three systems investigated was based on a deep narrow mine approach where continuous exposure of high-grade D-Zone coal was maintained throughout the 35-year production period. This would then provide the generating station with the most favourable average run-of-mine coal quality. All out-of-pit operating functions and support facilities were assumed to be the same for the three systems. The design of these facilities was based on judgement as to what was considered to be practical and proven for the Hat Creek area with prototype designs avoided as much as possible. Capital and operating cost estimates were in unescalated 1977 dollars. To compare alternatives all unit costs were levellized at a discount rate of 10%.

732 PIT CONFIGURATIONS STUDIED AND PRODUCTION SCHEDULE

A total of 10 trial and error, 35-year pit plans were prepared. Pit bottom elevations were varied from 700 metres ASL to 600 metres ASL. The quantity and quality results from these various pit plans suggested that the best pit depth which would provide the maximum average grade of run-of-mine coal was an elevation of 632 metres ASL.

In addition, several incremental mine plans covering the 35-year production period were developed to reflect a two-mine broad approach, versus a single-mine, narrow, deep approach. The two-mine approach was abandoned in favour of the single approach as it was estimated that the annual total quantities to be mined in each of the first 10 years were up to 10 million bank cubic metres per annum higher.

A production schedule reflecting a single approach was prepared using the Phase II computer routines in order to analyze the contents of a series of pits designed for the shovel/truck/ conveyor and bucket wheel excavation systems. Economic comparisons of the three systems were carried out in this portion of the study, based on two production schedules. Although annual material quantities varied, both came to the same approximate total volume of materials mined during the 35-year period.

733 ALTERNATIVE SYSTEMS COMPONENTS

The three mining systems under consideration differed primarily in that each had an alternative method of feeding the central mine conveyor. The shovel/conveyor and bucket wheel excavator/ conveyor systems relied mainly on bench conveyors, belt wagons, bridge conveyors, and transfer conveyors to feed the central mine conveyor. The shovel/truck/conveyor system incorporated large capacity trucks, travelling on prepared roadways in the pit for a similar purpose. Both the shovel/conveyor and bucket wheel excavator/conveyor systems did, however, employ shovel/ truck fleets to develop the centre of the pit during initial years and to expand the pit laterally thereafter. The three alternative systems are discussed briefly below.

733.1 Shovel/Truck/Conveyor System

The operating components used for this system included the following features:

(a) Pit materials were excavated using 11.5 cubic metre electric shovels, segregated by class or quality, and delivered via 109-tonne capacity end-dump trucks to one of four central inclined conveyors at the north end of the mine.

(b) Prepared roadways, 30 metres wide, were constructed on each of the operating benches from gravel materials available in the pit. These were relocated periodically as the mine faces advanced.

(c) 10% of both the coal and waste materials were blasted.

(d) The benches were generally 15 metres high and 50 metres wide except that at each fourth bench, an additional 5 metres in width was allowed for access and stability.

(e) Total quantities mined over the 35-year life were 361 million tonnes of coal and 464 million cubic metres of waste. The pre-production quantities of waste were 7 million bank cubic metres. The average annual quantities mined were 9.3 million tonnes of coal and 12 to 15 million bank cubic metres of waste.

733.2 Shovel/Conveyor System

The operating components used for this system included:

(a) An operating unit consisting of an 11.5 cubic metre shovel which fed a mobile hopper, designed to reject over-size material not suitable for conveyors. The shovel/hopper combination transferred material either upwards or downwards to a shiftable bench conveyor by means of belt wagons or bridge conveyors, depending on the distance involved in the inclined transfer. The shiftable bench conveyors then transported the materials to one of four central mine conveyors at the north end of the mine.

(b) The bottom bench, because it was in a constant state of development, was primarily worked by a shovel/truck system with the materials being transported to an unloading station on the central inclined conveyors.

(c) 10% of both the coal and waste materials were blasted.

(d) Prepared roadways were limited to only the bottom benches on which large truck operations were necessary. All other benches were serviced by 10-metre wide light duty access roads.

(e) Oversized materials rejected at the mobile hopper were hauled to one of two in-pit, mobile crushing facilities near the central inclined conveyor system.

(f) Auxiliary operations such as ramp construction, bench clean-up, and road maintenance were included in the overall mining scheme.

(g) The total quantities mined over the 35-year life were 361 million tonnes of coal and 464 million bank cubic metres of waste. The pre-production quantity of waste increased to 18 million cubic metres. The average annual quantities mined were 10 million tonnes of coal and 13 to 20 million cubic metres of waste.

733.3 Bucket Wheel Excavator/Conveyor System

The operating components for this system were based on current mining practice at various bucket wheel excavator operations throughout the world and adapted to Hat Creek conditions.

The evaluation was based on conceptual mine planning to assess the applicability of the bucket wheel excavator/conveyor systems to develop the deposit, with specific attention paid to the selective mining of waste parting materials. The system, as recommended by a specialist consultant, utilized four bucket wheel excavator/conveyor units, each conveying materials independently to the main transfer facility outside of the mine. The bucket wheel excavators were considered to be the only primary excavating equipment.

A pit bottom elevation of 670 metres ASL was reached at Year 35 with an overall volume of materials removed of approximately 700 million bank cubic metres using the recommended final slope requirement. Incremental mine development plans, showing the periodic materials movement and run-of-mine coal grade variation, were not outlined at this stage of the study.

734 FACILITIES COMMON TO ALL SYSTEMS

The order-of-magnitude comparisons were based on the assumption that all out-of-pit operating functions and support facilities were the same for all three systems even though it was recognized that there would in fact be minor differences. Their common facilities are described briefly below:

(a) Central Mine Conveyor System - This system comprised four conveyors, each 600 metres long, with a belt width of 1200 millimetres. The conveyors transported materials from an in-pit elevation of 730 metres ASL to an out-of-pit distribution plant at elevation 900 metres ASL. Incorporated in the system were three major truck unloading stations, each with the ability to distribute materials onto one of the two conveyor belts.

(b) Waste Conveyor System - This system consisted of two conveyors, approximately 14 400 metres in total length, with a belt width of 1200 millimetres. These conveyors were to transport waste from the distribution plant to the Houth Meadows and Medicine Creek waste dumps, construction grade materials to the major waste dump embankments, and low-grade coal to the stockpile area adjacent the Medicine Creek disposal area. Materials were placed in the waste embankments and dump areas by two large mobile spreaders, each equipped with a 40-metre discharge boom.

(c) Overload Coal Conveyor System - A single conveyor, with a total length of 4300 metres and a belt width of 1200 millimetres, transported blended coal from the blending facility to the generating station.

(d) Coal Beneficiation Plant - This facility provided for the crushing, screening, washing, and tailings handling equipment necessary to beneficiate the coarse sizes of the A, B, and C Zone coals prior to their transportation to the adjacent blending area. All the D-Zone coals, after crushing, would bypass the beneficiating plant and proceed directly to the blending area. The capacity of the beneficiating plant was 1000 tonnes per hour.

(e) Coal Blending Facility - This was designed to receive raw coal directly from the mine and the coal beneficiating plant, and to place this in two active piles for blending and reclaim purposes. The capacities of each pile, or blending unit, were 280 000 tonne (1 week) and 140 000 tonne (1/2 week). The equipment consisted of two stackers, two bucket wheel reclaimers, and approximately 3200 metres of conveyor with a belt width of 1200 millimetres.

(f) Administrative and Maintenance Buildings - The total area of all buildings and support facilities for the mine was in the order of 20 000 square metres. The three major facilities included equipment maintenance shops and warehouses (10 000 square metres), the administrative complex (1600 square metres), and the mine dry (2000 square metres).

(g) Power Supply Distribution System - This system received primary power at 60 kV. At the mine sub-station, voltage was reduced to 12.5 kVa and distributed to all the mine facilities. Further voltage reduction to 4160 and 550 V took place as required by the use of mobile transformers.

(h) Mine Dewatering, Surface Water Drainage, and Water Treatment - An extensive system of deep wells encircling various areas of the mine would enable depressurization of the mine walls in order to improve slope stability. A system of surface ditches was also developed to collect surface water, drain existing lakes, and pick up water from the deep wells. Most of the water collected was directed to a tailings pond settlement area. All leachate water draining from this tailings pond was to be treated before discharge into the Hat Creek.

735 CONCLUSIONS

The order-of-magnitude comparison carried out for the shovel/truck/conveyor and the shovel/conveyor was much more comprehensive than those carried out for the total bucket wheel excavator/conveyor system. However, the assessment was considered adequate for all three systems studied at this stage of the project development. For the first two systems, cost and operating capability were the key issues. For the total bucket wheel excavator/conveyor system, the paramount issue was the technical problem of adapting the system for mining the unique deposit at Hat Creek.

The order-of-magnitude cost analysis carried out examined the operating and capital costs on the basis of 1977 constant dollars for the total system. For comparison purposes the unescalated costs derived were discounted to present worth at 10%.

The resulting levellized costs per tonne mined were estimated to be:

System II - Shovel/Truck/Conveyor......$10.44/tonne of coal

System III - Shovel/Conveyor............$10.48/tonne of coal

System IV - Bucket Wheel Excavator/
Conveyor ....$10.02/tonne of coal

The differences are regarded as within the accuracy of the estimates but as noted the bucket wheel excavator system indicated the most favourable cost. However, several technical problems were identified in adapting the bucket wheel system to the structural and geotechnical nature of the deposit and are summarized as follows:

(a) The out-of-pit transfer conveyors would be located within the active slide areas for the mine life, assuming a north mine exit was retained. To overcome this, an additional 45-70 million cubic metres of slide material may have to be removed during the life of the mine.

(b) The alternative of a southern mine exit by the conveyor systems would, however, require further detailed investigations for all the out-of-pit facilities. This could influence the economic comparisons.

(c) To meet the average quality requirements of the generating station, it appeared that it would be necessary to adopt a narrower and deeper mining approach than normal for bucket wheel operations. This could require the bucket wheels to mine several benches simultaneously rather than fully developing one bench at a time.

(d) Because of their operating mode, higher stripping ratios occur during preproduction and the first 10 years of mining if bucket wheel excavators are used exclusively.

As a result of these technical problems it was decided to limit further bucket wheel excavation studies to a combined system of bucket wheels and shovel/trucks using a southern mine exit.

It was also decided that further work on the shovel/conveyor system was not warranted, and that only the shovel/truck/conveyor system would undergo further study as an alternative mining method to the combined bucket wheel excavator/shovel system. Reasons for this decision were:

(a) There was no significant cost advantage for the shovel /conveyor system as compared to shovel/truck.

(b) The shovel/conveyor system is not as flexible in meeting periodic coal quality requirements, and the semi-fixed conveyors are more vulnerable to ground movements and sloughs.

(c) The shovel/truck is a more proven system than the shovel/conveyor system. The latter system would require a considerable number of custom designed components.

7.4 DETAILED EVALUATIONS OF TWO SYSTEMS

741 INTRODUCTION

As a result of the conclusions reached in the previous level of studies the detailed evaluations concentrated on two mining systems: the shovel/truck/conveyor system, and the combined bucket wheel excavator/shovel/truck/conveyor system. Changes in operating components and objectives compared to previous levels of study are described briefly as follows:

(a) production and quality planning was carried out on the basis run-of-mine coal would not undergo beneficiation;

(b) the target for delivered fuel quality was lowered from 7875 to 7375 Btu/lb (dry basis);

(c) the moisture content of as-delivered coal was increased from 20% to 25%; and

(d) the mine development approach and location of supporting facilities for the combined bucket wheel excavator/shovel/truck/conveyor system were altered to allow a southern mine exit.

742 SHOVEL/TRUCK/CONVEYOR SYSTEM

Detailed evaluations of the two systems led to a recommendation that the shovel/truck/conveyor system be adopted for the development of the Hat Creek deposit.

A detailed description of this system and reasons for its selection have been dealt with in detail under Section 6 of this report.

743 COMBINED BUCKET WHEEL EXCAVATOR/SHOVEL/TRUCK SYSTEM

Included as Appendix A of the overall feasibility report is a detailed report produced by a specialist consultant entitled, "Study on the Application of Bucket Wheel Excavators for the Exploitation of the Hat Creek Deposit".

The total system planning and cost estimating for the combined bucket wheel/shovel/truck/conveyor system was not carried to the same level of detail and confidence as that for the shovel/truck conveyor system. Figure 7-1 illustrates the revised locations of the materials handling and support facilities to accommodate the southern mine exit on which the cost estimates were based. For the bucket wheel system cost estimate, a lower level of confidence is reflected by the recommendation and inclusion in this study of a slightly higher contingency factor. The economic studies were carried to a level where it was possible to effect a comparison resulting in the selection of the shovel/truck/conveyor system and the elimination, for the purposes of further study, of the combined bucket wheel/shovel/truck/conveyor system.

744 CONCLUSIONS

1. On a total system basis, there appears to be no economic advantage in utilizing bucket wheel excavators for the Hat Creek Project. Estimated capital and operating costs of the combined bucket wheel/shovel/truck/conveyor system and the recommended shovel/truck/conveyor system are compared in Section 4 of Volume VI - Capital and Operating Costs. These comparisons indicate that the initial investment for the recommended system during Years -3 to 4 is approximately $100 million lower than that for the combined system. Were the two systems to be assessed using the same contingency, the economic advantage would still remain with the recommended system.

2. In addition to its economic disadvantage, the combined bucket wheel system has the following limitations:

   (a) It seems unlikely that a production schedule could be developed to reflect a reasonably level, annual quantity of coal and waste materials excavation. This is due to a combination of the following factors:

      - to maintain a low annual rate of materials movement while meeting the coal quality objectives, the pit is best suited geologically to development by a deep, narrow approach. Standard bucket wheel operations require wide, long rotating benches with limited flexibility for moving the excavators from one bench to another.

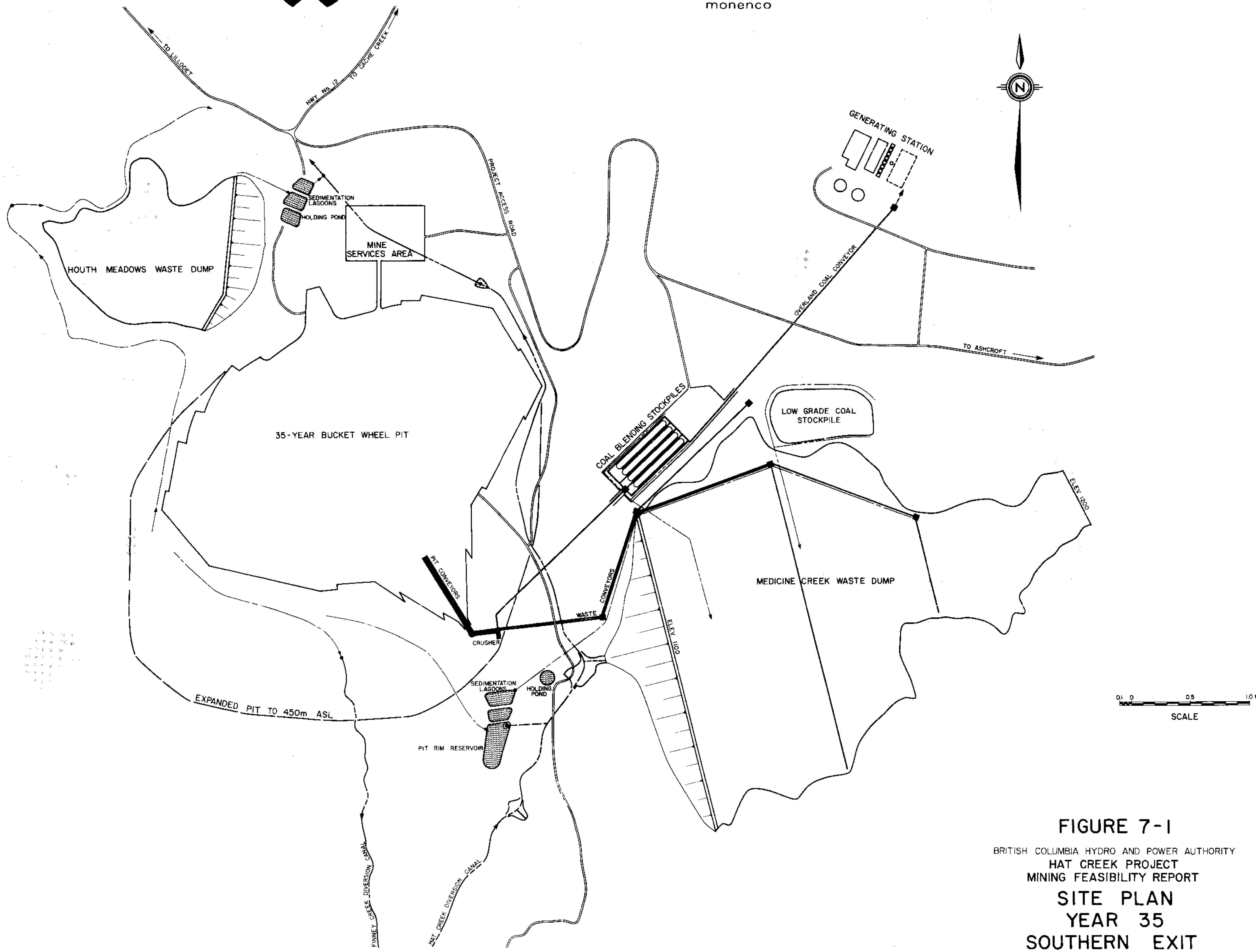

FIGURE 7-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
MINING FEASIBILITY REPORT

SITE PLAN
YEAR 35
SOUTHERN EXIT

- with the southern exit, initial pit development commences in an area where the coal is located at a much greater depth below the original ground suface than would be the case with a northern approach. Using the southern exit, the bucket wheel production schedule reflects the following average annual quantities of coal and waste excavation:

  Years 1 to 10 ... 30 million bank cubic metres per annum

  Years 11 to 25 ... 22 million bank cubic metres per annum

  Years 26 to 35 ... 9 million bank cubic metres per annum

  When compared to the annual production statistics for the recommended system given on Table 5-5 in Section 5 of this volume it can be seen that from the points of view of mining and cash flow, these results are undesirable.

(b) With a southern exit, an average grade coal quality is more difficult to achieve during the early years of mine development due to the minimum of better-grade coals that are encountered. The opposite conditions occur if the mine is developed from the north.

(c) Variations in coal quality from an average grade could be more difficult to avoid than with the recommended system. This is due primarily to the normal excavation routines of the bucket wheel excavator which do not encourage a multitude of mining faces at various locations in the mine or on the same bench.

(d) The complexity and length of conveyor systems in the pit is increased by the use of bucket wheel excavators. The combined system requires six in-pit conveyors, exiting from the mine through a sophisticated central distribution point, and extending to a total system length of 32 600 metres. By comparison, the recommended shovel/truck system requires only three in-pit conveyors and a total system length of 27 455 metres.

(e) The necessity of a southern exit for the combined system makes it desirable to maximize use of the Medicine Creek waste area with little, if any, utilization of the Houth Meadows waste area. This may impose limitations and increased project costs if a portion of the Medicine Creek waste area should ever be required for the needs of the generating station, or if future mine operations do not find it practical to use the 35-year pit for back-filling purposes and must still resort to waste disposal in Houth Meadows area.

3. The risk associated with selection of a mining system that requires a very early and significant commitment, and which is relatively inflexible to changes in mining conditions, cannot be minimized. The bucket wheel excavator system, as proposed, required a commitment to purchase four bucket wheel excavators and belt wagons in Year -3 in order to allow the commissioning of two bucket wheels systems in Year -1, and an additional two systems in Year 1.

4. The study presented in Appendix A is based entirely on the geological data contained in the Phase II computer model. Time constraints and other factors did not allow the re-calculation of all the mine development quantities and qualities to reflect the latest geological data contained in the Phase III computer model. Based on the experience gained in other comparisons of Phase II and Phase III data results it is unlikely that for the bucket wheel system study the Phase III data would affect the coal quantity and quality results developed for the 35-year mine life. However in the annual increments of the first five years where variations in coal quantity and quality can be expected to occurr, the mine plans can be modified without difficulty to meet the objectives of average coal quantity and quality.

5. The study presented in Appendix A emphasizes that a more detailed and favorable presentation of the geological data would considerably improve the possibility of selecting bucket wheels for the development of Hat Creek. To have provided this data to the level requested, it appeared very likely that considerably more geological data from a close-spaced drill program would have been required. This additional drill data could not have been made available within the time frame of this study.