BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

PRELIMINARY ENGINEERING

COMPOSITE REPORT

APPENDIX A

MINING

September 1978

Copy No. 9

APPENDIX A

MINING

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SECTION A1.0 - REVIEW OF MINING WORK

This Appendix covers mining work during the period from project inception in April 1974 to the end of the preliminary engineering phase in 1978.

A1.1 EXPLORATION DRILLING

The purpose of exploration drilling is to define the structure, size, limits and quality of the resource. To this end, diamond drilling programmes were initiated in mid-1974 and have been conducted each year since to investigate the coal deposit at the north end of the Upper Hat Creek Valley. The most recent of these programmes was completed in September 1978. With the exception of 1975, when the emphasis was placed on investigating an anomaly that subsequently was identified as the No. 2 Deposit, work has been concentrated on the No. 1 Deposit.

Since 1974, 264 holes have been drilled in the Upper Hat Creek Valley totalling approximately 89 500 m (294,000 ft) in length (see Plate A1-1). The No. 1 Deposit has been well defined by 220 holes on a 150 m by 150 m (500 by 500 ft) grid pattern. At the completion of the current drilling programme the holes drilled in the No. 1 Deposit will have yielded data on 59 500 m (195,000 ft) of hole comprising 11 600 m (38,000 ft) in surficial material, 47 900 m (157,000 ft) in coal and waste rock.

Cores obtained from drilling have been carefully logged by geologists and stored. The cores from coal horizons have been sampled and submitted for analysis of thermal, chemical and combustion properties.

A1.1 EXPLORATION DRILLING - (Cont'd)

As a result of these drilling programmes, conducted under the supervision of Dolmage Campbell and Associates Ltd. and the B.C. Hydro Mining Department, a good understanding of the geological structure and the quality distribution of this complex coal deposit has been developed. Reserves in excess of 700 Mt (770 million tons) have been established for the No. 1 Deposit. The No. 2 Deposit has been identified as a potentially much larger resource.

A1.2 GEOTECHNICAL STUDIES

A geotechnical assessment programme was initiated and assigned to Golder Associates. Extensive field investigations commenced in 1976 are now being completed. The major purpose of the programme has been to establish safe working slopes for the open pit mine in the No. 1 Deposit. The stability of these slopes is controlled by the strength of the materials and the groundwater conditions in the area. Testing has also been completed on the waste dump foundations and on the behaviour of waste materials.

For the conceptual design phase (see Sub-section A1.3(a) below) an overall pit slope angle of 16^{0} was recommended, which was necessarily conservative because of the limited testing completed at that time. Based on additional data developed in the preliminary engineering phase it was possible to vary the slope angles between 16^{0} and 25^{0} in different pit wall materials. This has substantially reduced the volume of waste materials to be removed over the life of the project. Waste dump design parameters have also been established.

These studies indicate that the Hat Creek rock formations are geotechnically complex and difficult. However, the results obtained from the extensive investigations have established mine design parameters with a satisfactory level of confidence.

A1.2 GEOTECHNICAL STUDIES - (Cont'd)

It is clear that it will be necessary to maintain a geotechnical monitoring and control function throughout the life of the mine.

A1.3 MINING STUDIES

(a) Conceptual Design

In September 1975 Powell-Duffryn National Coal Board Consultants Limited (PD-NCB) were commissioned to conduct mining studies on the Hat Creek coal deposits.

Preliminary reports were issued on No. 1 and No. 2 Deposits in March 1976 and June 1976 respectively. It was recommended that future work should be concentrated on the No. 1 Deposit. Further reports dated March 1977 were prepared: "Revised Report on Hat Creek Openpit No. 1" and "Reclamation Study Hat Creek Openpit No. 1".

(b) Preliminary Engineering Phase

Preliminary engineering design studies were assigned to the Cominco-Monenco Joint Venture (CMJV) in May 1977. The results of these studies were reported in draft form in June 1978.

An open pit mine has been designed to supply 350 Mt (385 million tons) of coal averaging 17.0 MJ/kg (7327 Btu/lb) on a dry basis to the powerplant over a period of 35 years. This will require the removal of 450 Mm^3 (585 million yd^3) of waste material.

CMJV investigated alternative mining systems and approaches to developing the deposit. As presently proposed an open pit mine would be developed to 3 km (2 mi) by 2.5 km (1.5 mi) by 265 m (870 ft) deep using a shovel-truck-conveyor mining

A1.3 MINING STUDIES - (Cont'd)

system. Coal crushing, blending and stockpiling facilities would be installed at the mine mouth. Blended coal would be transported by conveyor to the powerplant 4 km (2.5 mi) away and 500 m (1640 ft) above the valley floor. Waste material would be delivered by conveyor from the mine mouth to waste disposal areas at Houth Meadows and the Medicine Creek Valley.

(c) Bulk Sample Programme

A bulk sample programme was conducted during the summer of 1977 under the supervision of B.C. Hydro's Thermal Division staff. Coal was excavated from two trenches in the No. 1 Deposit to provide 6300 t (7000 tons) of coal for a burn test at the Battle River thermal generating plant operated by Alberta Power Ltd. This pilot scale mining operation provided valuable data on the mining, handling and storage of the coal and waste materials.

A1.4 COAL BENEFICIATION

The purpose of coal beneficiation is to raise the heating value of the run-of-mine coal to make a better fuel for the powerplant. This can only be achieved at the expense of some process losses. Most beneficiation methods currently employed are wet gravity separation processes.

In 1975 Birtley Engineering (Canada) Limited performed both bench and pilot scale tests on Hat Creek coal samples obtained by bucket-auger. These tests were performed using slightly modified standard industry procedures. Because of the clay in the coal the results of the two types of test were not well correlated, but did clearly establish the difficulty of washing Hat Creek coal.

A1.4 COAL BENEFICIATION - (Cont'd)

Further bench scale tests were conducted in 1977 under the direction of Simon-Carves (Canada) Ltd., sub-consultants to CMJV. These tests were performed on samples obtained from the bulk sample programme trenches using further modified procedures and a recently developed Australian Standard attrition test. These tests confirmed and explained the results previously obtained by Birtley.

In 1977 a 73 t (80 ton) sample from the bulk sample programme was tested in the pilot plant at the Western Research Laboratory of Energy, Mines and Resources in Edmonton. This test has proven the practicability of coal washing and provided data on the production and treatment of the tailings that could not have been gathered in any other way.

Although these studies have indicated that Hat Creek coal can be beneficiated, it cannot be justified at this time due to technical and economic constants. raints The subject is discussed in detail in Appendix D. The recommended scheme for the production of powerplant fuel relies on blending rather than beneficiation.

cominco-monenco joint venture

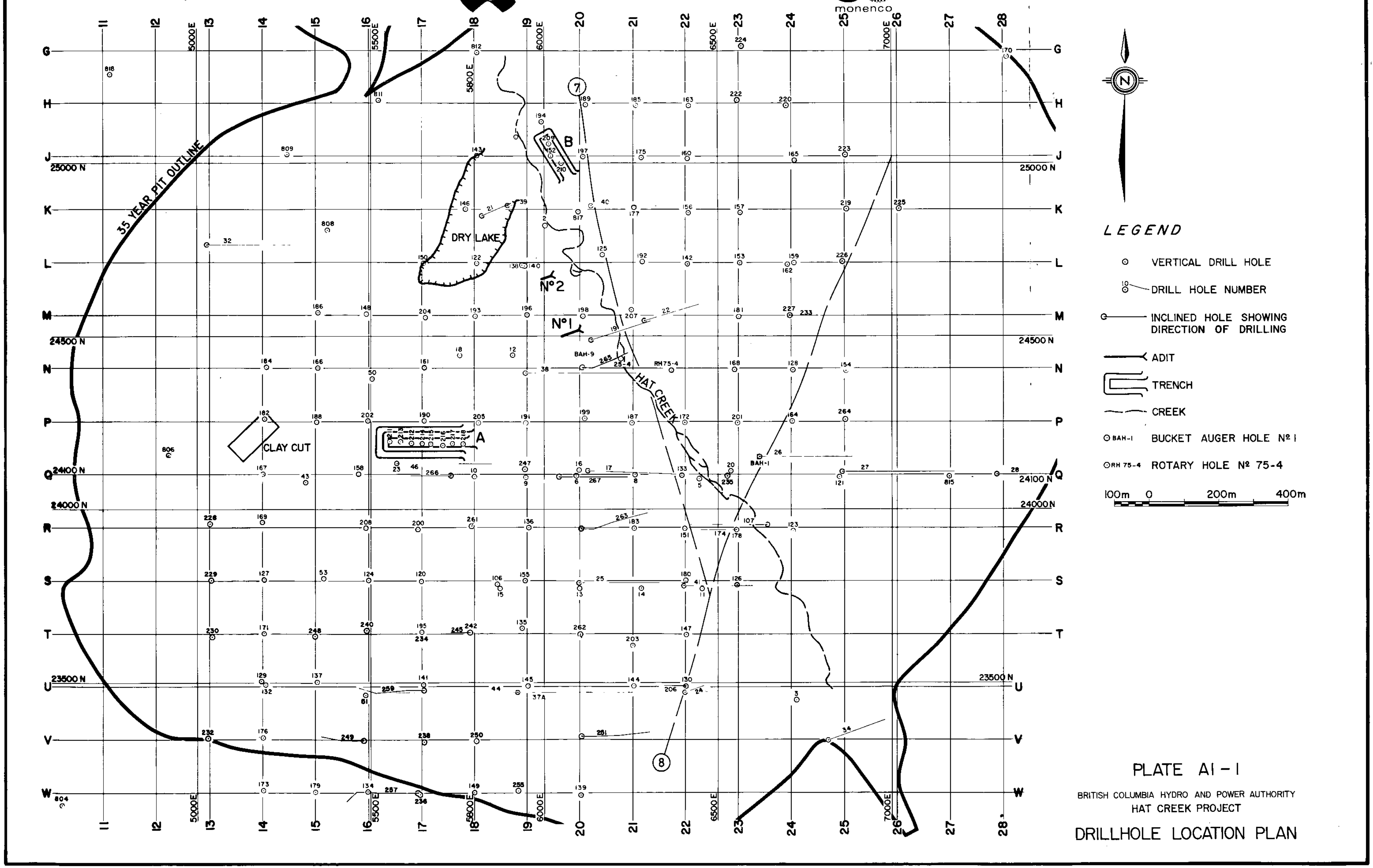


PLATE A1-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

DRILLHOLE LOCATION PLAN

SECTION A2.0 - GEOLOGY

Introduction

Coal in Hat Creek was first reported by G.M. Dawson of Geological Survey of Canada in 1877. Sporadic development continued until 1974 when B.C. Hydro began a systematic exploration and evaluation programme.

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

Samples were also provided for washability studies at the laboratories of Energy, Mines and Resources in Edmonton, Birtley Engineering (Canada) Limited and Warnock Hersey Professional Services Limited in Calgary. Warnock Hersey also conducted wet attrition tests to simulate size degradation in wash plant.

A2.1 STRATIGRAPHY

The coal bearing section belongs to the Hat Creek Formation of the Eocene epoch in the Tertiary Period deposited 36 to 42 million years ago. It is underlain by the Coldwater Formation consisting of mixed detrital material and overlain by poorly consolidated bentonitic claystone and siltstone beds of the Medicine Creek Formation. Volcanic activities have indurated these rocks as well as the coal. During the

Pleistocene epoch these beds were subjected to glaciation and subsequently overlain by glacio-fluvial material. The regional stratigraphy is summarized in Table A2-1.

Table A2-2 illustrates a scheme for the development of stratigraphic subdivision of the Hat Creek Formation. These subdivisions were useful for stratigraphical and quality correlations.

Based on lithology and coal quality the Hat Creek Formation was subdivided into 14 subzones. Two of these subzones, A-2-1 and C-1-1, are essentially waste and coaly shale units, while the remaining 12 represent coal of varying qualities.

The four subzones in A-1 coal zone are defined by the tops of the three most continuous partings in the zone. The parting markers are A-4, A-9 and A-11 as labelled on the included sample geophysical log (Plate A2-1) for DDH 76-136. Characterisic gamma ray and density log peaks are used to identify these marker horizons throughout most of the deposit. The basic principle is that the geophysical logs reflect the varied coal zone lithologies and provide a means of subzone correlation between drill holes. Gamma ray log peaks essentially reflect claystone interbeds (partings) with relatively high radioactive K ion content. The corresponding density log reflects the variation in density of the rock and coal or coaly material.

Two subdivisions of B-1 Zone are based on subzones of nearly equal thickness that are identifiable on geophysical logs. C-2 coal zone is divided into two subzones that are also identified and correlated using geophysical logs. C-2 often contains lenticular waste partings of substantial thickness but of limited continuity. D-1 coal zone is subdivided into four quality subzones to provide narrower units

A2.1 STRATIGRAPHY - (Cont'd)

of generally better quality coal, with calorific value greater than 18.6 MJ/kg (8000 Btu/lb). These D subzones are also characterized and correlated using the geophyscial logs.

In the western limb of the main syncline, Zones B-1 and C-2 exhibit a marked increase in shale content. Some of the seams and partings used as marker beds in this region are impossible to identify.

A2.2 BAKED ZONES

Burning effects have been noted not only as surface exposure but also in the cores. This effect was noted in trench A where the burnt clay section (baked clay) is in direct juxtaposition with coal, without any evidence of burning in the coal. From this, it is evident that the present placement is post-burning.

Differential magnetic anomalies existing between the baked clay and the country rock led to the successful application of magnetometer survey for outlining the burnt zones.

The Dry Lake has been suggested as an eroded, collapsed structure that may have resulted from a burned former coal outcrop.

The resultant burnt effect could be attributed to the coal seam being ignited by spontaneous combustion or forest fires, though the volcanic activity in the adjoining area could also be partially responsible.

A2.3 STRUCTURE

The primary structure consists of two synclines separated by an anticline, plunging at an average of 15^{o} to 17^{o} towards the

A2.3 STRUCTURE - (Cont'd)

south-southwest. It is truncated on the south and east by steeply dipping boundary faults (Plate A2-2). Repetition of sections has been observed in some of the drill cores. Such overturning is due to thrust faulting which probably is also responsible for the anomalous thickness of detrital material encountered in the western sector. Undoubtedly, the general facies change in this direction has significantly contributed to the thickening (Plates A2-3, A2-4 and A2-5).

A2.4 QUALITY

Coal sampling was conducted on all cores. Large quantities of auger and trench samples were obtained for combustion and washability studies.

With the application of geophysical logging in all drilling programmes from 1974 on, it has been possible to determine the seam structure including the interlayered coal and shale bands and correlate them to the analytical data for that section. Sample lengths were adjusted to correspond to the geophysical reading.

The regression plot (Plate A2-6) shows the linear relationship between percent dry ash and heating value for 309 samples from drill holes 76-135 and 136. It is noted that this relationship remained practically unchanged for the various zones taken individually. The boiler fuel specification (Table A2-3) takes the expected variation in the many parameters into account. This specification discussed in detail in Appendix D, was developed as follows:

1. The mean and standard deviations for proximate, ultimate and ash analysis were developed for each of the four zones (A, B, C and D).

A2.4 QUALITY - (Cont'd)

2. Each zone was weighted in the proportions to be mined over the 35-year pit life.

3. The results obtained in 1 and 2 above were checked against samples selected from limited heating value ranges for each zone in the whole deposit. There is no significant difference between the parameters of an individual fuel of a given heating value and a blended fuel of the same heating value.

4. The values presented in the fuel specification are weighted mean values and the confidence ranges are ±1 standard deviation.

A2.5 EXPLORATION IN PROGRESS

The objectives of the current programme (July to September 1978) are to confirm the disposition of the postulated faults especially the Creek Fault and to replace or confirm the geological and chemical information obtained from the 1925 to 1959 drill holes, so that it conforms to the current data.

(a) Drilling

Diamond drilling NQ (core diameter 1 7/8-inch or 47.5 mm) with wire line for continuous coring was adopted for this programme. Special stress was laid on the engineering properties of the unconsolidated overburden.

(b) Logging

All holes are geophysically logged (Gamma Ray and Density) on a scale 1:250. Geolographs provide data on the rate of penetration versus bit pressure and bit rpm versus pump pressure.

A2.5 EXPLORATION IN PROGRESS - (Cont'd)

The core recovery is determined with minimal loss of time, keeping in mind that part of the losses can be compensated by the swelling effect of clay.

Geophysical logs are used to identify critical boundaries and the nature of materials. The density logs are of particular help in identifying carbonaceous shale, shaly coal, coal etc.

(c) Sampling and Analysis

All sampling and anlaytical procedures are in accordance with the following ASTM standards:

D-2013-72	For Preparation of Coal Samples
D-3173-73	For Moisture
D-3174-73	For Ash
D-2015-66	For Thermal Value
D-3177-75	For Total Sulphur

A2.6 GEOSTATISTICAL STUDIES OF THE VARIATION OF HEATING VALUE AND SULPHUR

B.C. Hydro initiated a geostatistical analysis of the Hat Creek No. 1 Deposit under the direction of Dr. Michel David of Institute de Recherche en Exploration Minerale - Mineral Exploration Research Institute (IREM-MERI). The purpose of this study was twofold:

1. To determine the variations in heating value and sulphur content in the subzones.

2. To estimate the average heating value and sulphur content in 75 m by 75 m (250 ft by 250 ft) blocks by kriging.

A2.6 GEOSTATISTICAL STUDIES OF THE VARIATION OF HEATING VALUE AND SULPHUR - (Cont'd)

Heating value content was examined in all the 14 subzones, A-1-1 to A-1-4, A-2-1, B-1-1, B-1-2, C-1-1, C-2-1, C-2-2, D-1-1 to D-1-4. Sulphur content was analyzed in three zones, A-1-1, B-1-2 and D-1-3, where appreciably higher sulphur has been recorded.

The variation in heating value and sulphur content was investigated in each of the subzones by computing variograms in four directions, northsouth, northeast-southwest, eastwest and northwest-southeast. The variograms demonstrated the degree of continuity in these parameters.

The average values and standard error of heating value and sulphur content were computed by kriging equations.

The following conclusions were drawn from the initial studies:

1. Coal zone variograms (A-2-1, C-1-1 are considered waste zones) demonstrated low variance in heating value, corresponding to good continuity of the heating values.

2. Sulphur variograms in A and B zones showed a high variance corresponding to a lack of trending in the sulphur content. This can be observed in the irregular distribution of pyrite, which causes the total sulphur content to appear as abnormal or erratic.

3. D zone exhibited better continuity in sulphur content than the other two zones.

4. No direct relationship was found to exist between sulphur and heating value.

TABLE A2-1

REGIONAL STRATIGRAPHY - HAT CREEK COAL BASIN

Period	Epoch	Million Years		Formation or Group	Thickness (m)	Rock Types
Quaternary	Recent				Not Determined	Alluvium, Colluvium, fluvial sands and gravels, slide debris, lacustrine sediments.
	Pleistocene	1.5 - 2				Glacial till, glacio-lacustrine silt, glacio-fluvial sands and gravels, land slides.
	Unconformity					
	Miocene	7 - 26		Plateau Basalts	Not Determined	Basalt, olivine basalt (13.2 m.y.), andesite, vesicular basalt.
	Unconformity (?)					
Tertiary	Miocene or Middle Eocene ?		Kamloops Group	Finney Lake Formation	Not Determined	Lahar, sandstone, conglomerate.
				Unconformity		
	Late Eocene			Medicine Creek Formation	600+	Poorly consolidated bentonitic claystone and saltstone.
				Paraconformity ? (McCullough 1978)		
	Late Eocene to Middle Eocene	* 36 - 42		Hat Creek Coal Formation	550	Mainly coal with intercalated siltstone, claystone, sandstone and conglomerate.
				Coldwater Formation	375	Unconsolidated or semi-consolidated siltstone, claystone, sandstone, conglomerate, minor coal.
				Fault Contact (McCullough 1978) or Nonconformity		
	Middle Eocene	43.6-49.9			Not Determined	Rhyolite, dacite, andesite, basalt and equivalent pyroclastics.
	Unconformity (McKay 1925; Duffell & McTaggart 1952)					
Cretaceous or Later	Coniacian to Aptian **	88.3±3 m.y.		Spences Bridge Group	Not Determined	Andesite, dacite, basalt, rhyolite; tuff breccias, agglomerate.
	Erosional Unconformity (Duffell & McTaggart 1952)					
		98		Mount Martley Stock	Not Determined	Granodiorite, tonallite.
	Intrusive Contact (Duffell & McTaggart 1952; McCullough 1978)					
Pennsylvanian to Permian or earlier		250-330		Cache Creek Group:		
				Marble Canyon Formation	Not Determined	Marble, limestone, argillite
				Greenstone	Not Determined	Greenstone, chert, argillite; minor limestone and quartzite, chlorite schist, quartz-mica, schist.

* Based on palynology by Rouse 1977

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

TABLE A2-2

DEVELOPMENT OF STRATIGRAPHIC SUBDIVISION IN

HAT CREEK COAL FORMATION

Stage I	Stage II	Stage III
A	A-1	A-1-1 A-1-2 A-1-3 A-1-4
	A-2 (waste zone)	A-2-1
B	B-1	B-1-1 B-1-2
C	C-1 (waste zone)	C-1-1
	C-2	C-2-1 C-2-2
D	D-1	D-1-1 D-1-2 D-1-3 D-1-4
Recognition of four broad zones in the normal depositional sequence.	Identification of two waste zones - A-2 & C-1.	Coal bearing zones A-1 - subdivided into four sub-zones separated by three partings. B-1 - subdivided into two sub-zones varying in quality. C-2 - subdivided into two sub-zones separated by lenticular waste partings of limited continuity. D-1 - subdivided into four sub-zones of varying quality.

TABLE A2-3

BOILER FUEL SPECIFICATION DATA

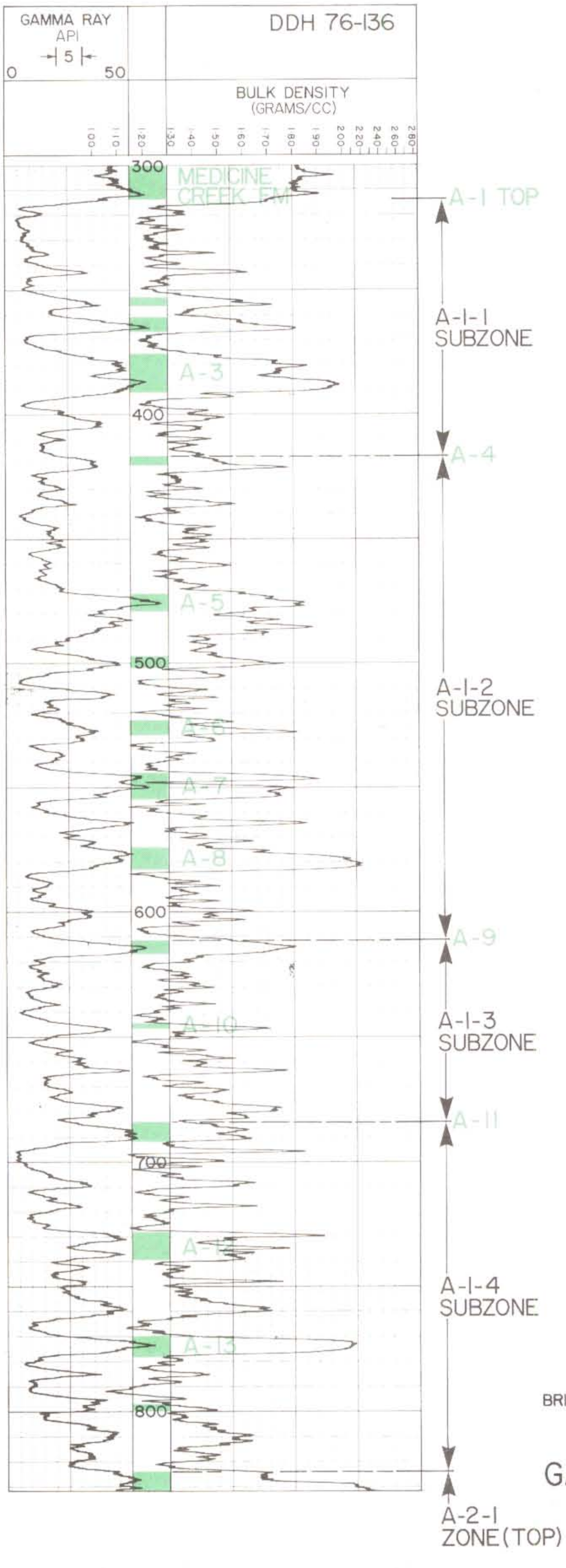
	WEIGHTED AVERAGE	STANDARD DEVIATION
Ultimate Analysis		
% Carbon	43.90	±1.49
% Hydrogen	3.74	±0.56
% Nitrogen	0.89	±0.15
% Oxygen	14.58	±1.44
% Sulphur (dry basis)	0.48	±0.25
% Chlorine	0.03	±0.02
% Ash (dry basis)	36.30	±1.80
Calorific Value (dry basis)	7327 Btu/lb	±300
	17 043 kJ/kg	±700
% Moisture (run-of-mine)	25.0	±10.0
Ash Analysis (% dry ash)		
SiO_2	53.72	±6.02
Al_2O_3	28.85	±5.01
CaO	2.63	±1.99
MgO	1.41	±0.65
Fe_2O_3	7.62	±4.97
K_2O	0.52	±0.21
Na_2O	1.18	±0.51
Mn_3O_4	0.11	±0.13
V_2O_5	0.05	±0.03
P_2O_5	0.29	±0.30
SO_3	1.82	±0.90
TiO_2	0.92	±0.26
Undetermined	0.88	±0.94
Proximate Analysis (dry basis)		
% Ash	36.30	±1.80
% Volatile Matter	32.20	±4.17
% Fixed Carbon	31.40	±4.20
Carbon Dioxide (dry basis)	1.77	n.d. (not determined)
Water Soluble Alkalies		
as Na_2O	0.24	n.d.
as K_2O	0.03	n.d.
Ash Fusion Temperatures		
Reducing Atmosphere:		
Initial Deformation	1330°C	±200°
Ash Softening (H=W)	1325	
Ash softening (H=1/2 W)	1340	
Fluid	1400+	

Approximately 8.6% of the average fuel indicates an I.D.T. $<$ 1200°C.

Approximately 4.2% of the average fuel indicates an I.D.T. $<$ 1150°C.

Oxidizing Atmosphere:		
Initial Deformation	1340°C	±200°
Ash Softening (H=W)	1350	
Ash Softening (H=1/2 W)	1360	
Fluid	1400+	
Hardgrove Grindability Index	50	±10

cominco-monenco joint venture



NOTES.

GAMMA RAY-DENSITY LOG IS USED TO ILLUSTRATE THE IDENTIFIABLE PARTINGS IN A-1 COAL ZONE.

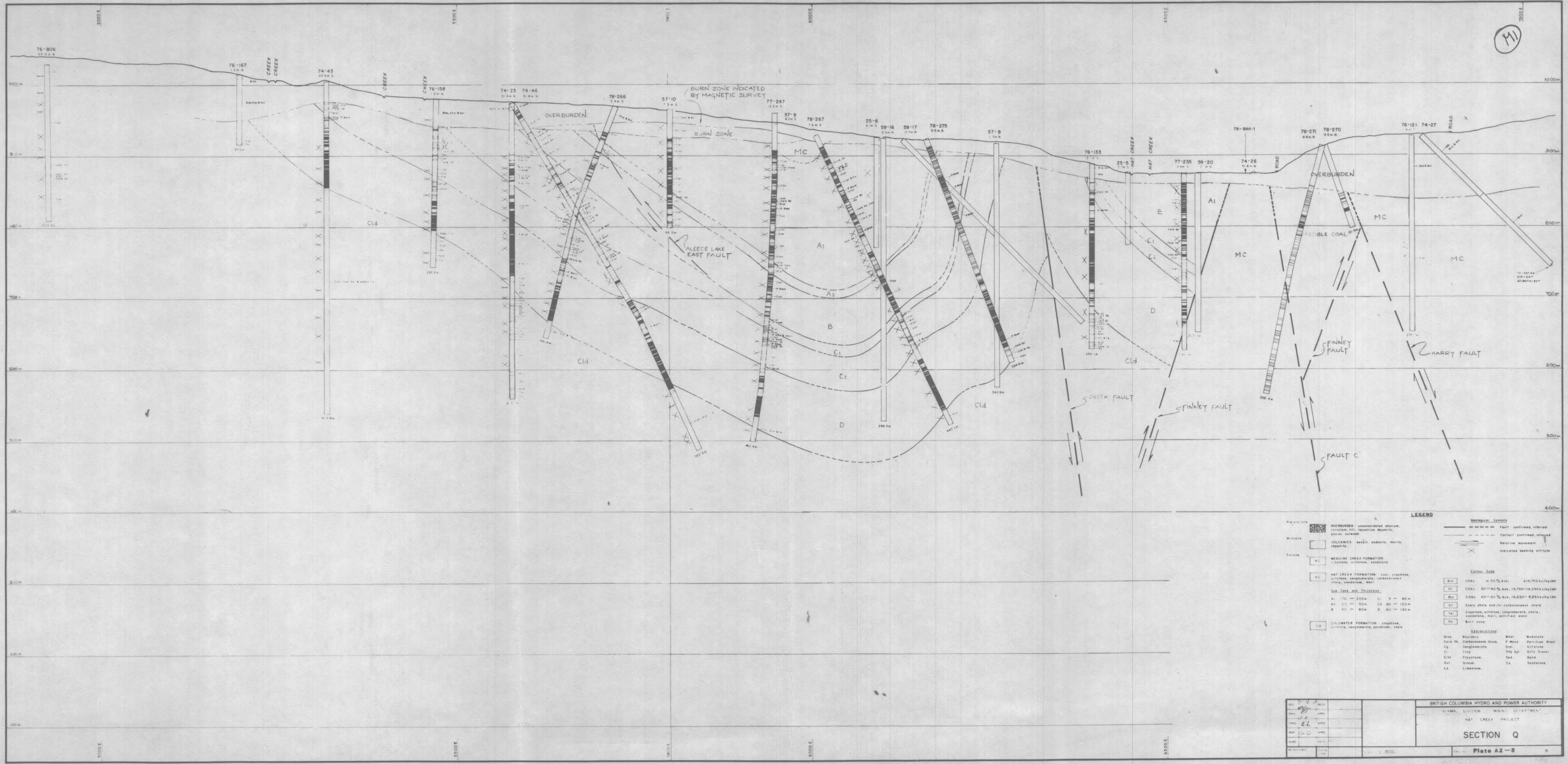
TOP OF PARTINGS A-4, A-9, A-11, ARE CONTOURED TO DEFINE SUBZONES A-1-1, A-1-2, A-1-3 AND A-1-4.

DEPTHS SHOWN ARE IN FEET FROM HOLE COLLAR.

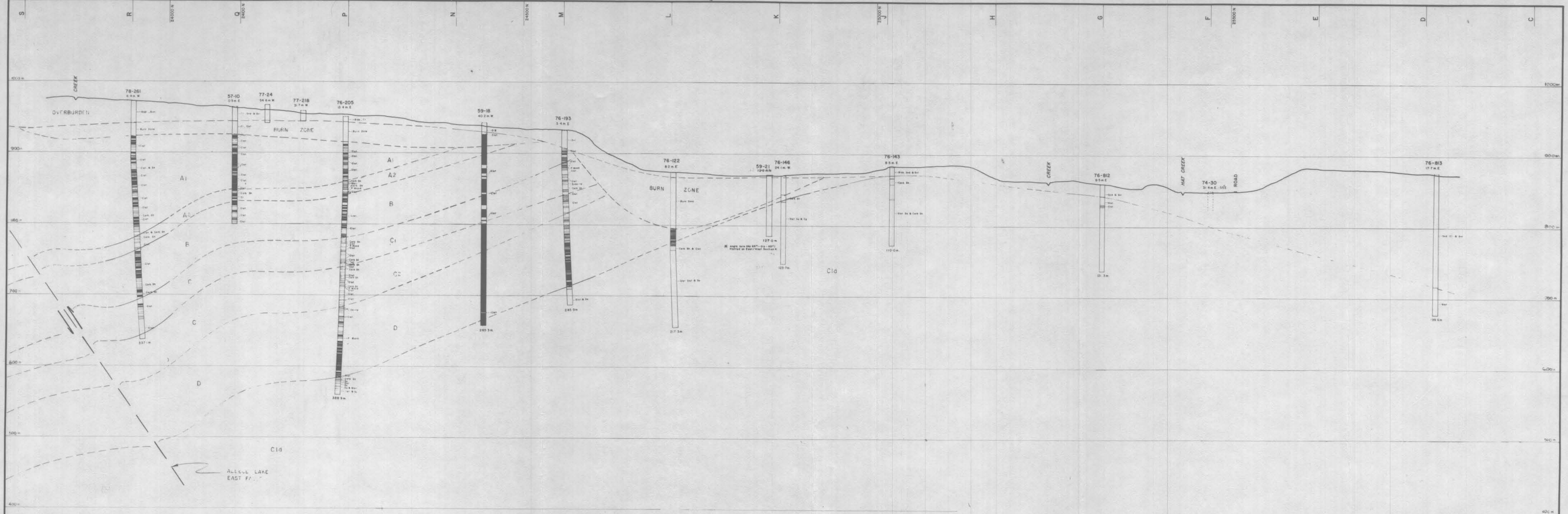
PLATE A2-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

GAMMA-RAY DENSITY LOG OF A-1 ZONE DDH 76-136



M1
5000 E
5500 E
6000 E
6500 E
7000 E
1000 m
900 m
800 m
700 m
600 m
500 m
400 m
76-806
76-167
CREEK
CREEK
74-43
CREEK
CREEK
76-158
74-23
74-46
78-266
57-10
BURN ZONE INDICATED
BY MAGNETIC SURVEY
77-247
57-9
78-267
25-6
59-16
59-17
78-275
57-8
76-133
25-5
HAT CREEK
HAT CREEK
76-BAH-1
77-235
59-20
74-26
ROAD
78-271
78-270
76-121
74-27
ROAD
OVERBURDEN
BURN ZONE
MC
A1
A2
B
C1
C2
D
E
Cld
HARRY FAULT
FAULT C
FINNEY FAULT
CREEK FAULT
POSSIBLE COAL
LEGEND
Pleistocene
OVERBURDEN unconsolidated alluvium, colluvium, till, lacustrine deposits, glacial outwash
Miocene
VOLCANICS basalt, andesite, dacite, rhyolite.
Eocene
MC
MEDICINE CREEK FORMATION claystone, siltstone, sandstone
HC
HAT CREEK FORMATION coal, claystone, siltstone, conglomerate, carbonaceous shale, sandstone, marl
Sub Zone and Thickness
A1 170 — 200m C1 5 — 80 m
A2 20 — 50m C2 60 — 120 m
B 50 — 80m D 60 — 130 m
Cld
COLDWATER FORMATION claystone, siltstone, conglomerate, sandstone, shale
Geological Symbols
Fault confirmed, inferred
Contact confirmed, inferred
Relative movement
Indicated bedding attitude
Colour Code
Blk COAL < 30 % Ash; +19,750 kJ/kg (db)
Dr COAL 30 — 40 % Ash; 19,750 — 16,250 kJ/kg (db)
Blu COAL 40 — 60 % Ash; 16,250 — 9,250 kJ/kg (db)
Gr Coaly shale and/or carbonaceous shale
Yel Claystone, siltstone, conglomerate, shale, sandstone, marl, petrified wood
Pk Burn zone
Abbreviations
Blds Boulders
Carb Sh Carbonaceous Shale
Cg Conglomerate
Cl Clay
Clst Claystone
Gvl Gravel
Ls Limestone
Mdst Mudstone
P Wood Petrified Wood
Stst Siltstone
Slty Gvl Silty Gravel
Sd Sand
Ss Sandstone
DES
DWN
CHKD E.L.
INSP
SUBD
REC'D
APPD
APPD
APPD
DATE
MICROFILMED
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
THERMAL DIVISION MINING DEPARTMENT
HAT CREEK PROJECT
SECTION Q
1:1500
Plate A2 — 3
R



LEGEND

Pleistocene

OVERBURDEN : unconsolidated alluvium, colluvium, till, lacustrine deposits, glacial outwash

Miocene

VOLCANICS : basalt, andesite, dacite, rhyolite.

Eocene

MC — MEDICINE CREEK FORMATION : claystone, siltstone, sandstone

HC — HAT CREEK FORMATION : coal, claystone, siltstone, conglomerate, carbonaceous shale, sandstone, marl

Sub Zone and Thickness :

A1	170 — 200m	C1	5 — 80 m
A2	20 — 50m	C2	60 — 120 m
B	50 — 80m	D	60 — 150 m

Cld — COLDWATER FORMATION : claystone, siltstone, conglomerate, sandstone, shale

Geological Symbols

Fault : confirmed, inferred.

Contact : confirmed, inferred.

Relative movement.

Indicated bedding attitude.

Colour Code

Blk	COAL : < 30 % Ash; +19,750 kJ/kg(db)
Or	COAL : 30 — 40 % Ash; 19,750 — 16,250 kJ/kg(db)
Blu	COAL : 40 — 60 % Ash; 16,250 — 9,250 kJ/kg(db)
Gr	Coaly shale and/or carbonaceous shale.
Yel	Claystone, siltstone, conglomerate, shale, sandstone, marl, petrified wood.
Pk	Burn zone

Abbreviations

Blds.	Boulders.	Mdst.	Mudstone.
Carb. Sh.	Carbonaceous Shale.	P. Wood	Petrified Wood.
Cg.	Conglomerate.	Slst.	Siltstone.
Cl.	Clay.	Silty Gvl.	Silty Gravel.
Clst.	Claystone.	Snd.	Sand.
Gvl.	Gravel.	Ss.	Sandstone.
Ls.	Limestone.		

DES		RECD	
DWN	S. W.	APPD	
CHKD		APPD	
INSP		APPD	
SUBD		DATE	
MICROFILMED		DIST'N CODE	

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

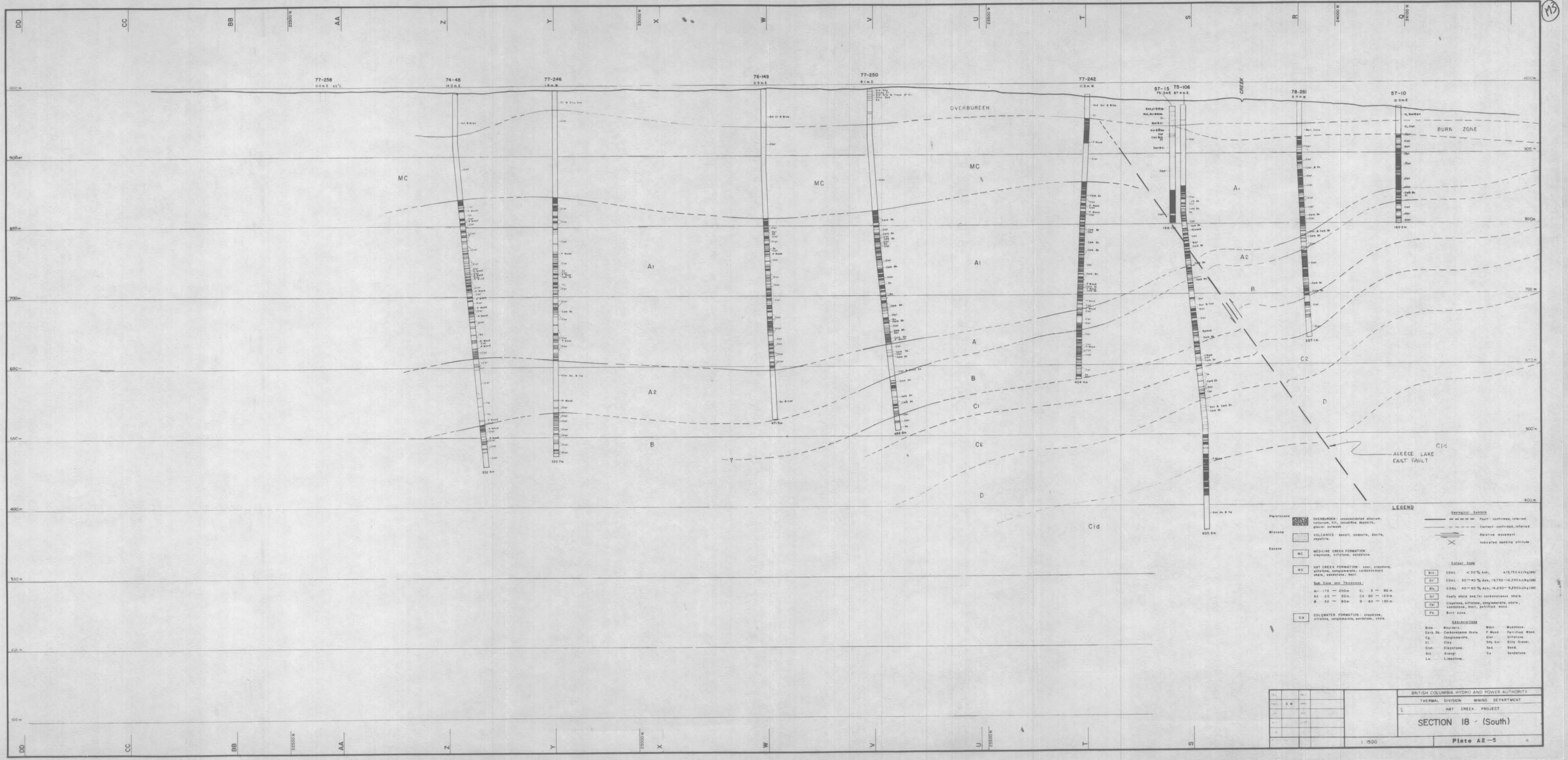
THERMAL DIVISION MINING DEPARTMENT

HAT CREEK PROJECT

SECTION 18 (North)

SCALE 1:1500

DWG. No. **Plate A2-4** R



DD
CC
BB
AA
Z
Y
X
W
V
U
T
S
R
Q
77-258
74-48
77-246
76-149
77-250
77-242
57-15
75-106
CREEK
78-261
57-10
OVERBURDEN
BURN ZONE
MC
A1
A2
A
B
C1
C2
D
C1d
ALEECE LAKE
EAST FAULT
LEGEND
Pleistocene
OVERBURDEN: unconsolidated alluvium, colluvium, till, lacustrine deposits, glacial outwash
Miocene
VOLCANICS: basalt, andesite, dacite, rhyolite
Eocene
MC
MEDICINE CREEK FORMATION: claystone, siltstone, sandstone
HC
HAT CREEK FORMATION: coal, claystone, siltstone, conglomerate, carbonaceous shale, sandstone, marl
Sub Zone and Thickness:
A1 170 — 200m
A2 20 — 50m
B 50 — 80m
C1 5 — 80m
C2 60 — 120m
D 60 — 150m
Cld
COLDWATER FORMATION: claystone, siltstone, conglomerate, sandstone, shale
Geological Symbols
Fault: confirmed, inferred
Contact: confirmed, inferred
Relative movement
Indicated bedding attitude
Colour Code
Blk
COAL: <30% Ash; +19,750 kJ/kg(db)
Or
COAL: 30—40% Ash; 19,750—16,250 kJ/kg(db)
Blu
COAL: 40—60% Ash; 16,250—9,250 kJ/kg(db)
Gr
Coaly shale and/or carbonaceous shale.
Yel
Claystone, siltstone, conglomerate, shale, sandstone, marl, petrified wood
Pk
Burn zone.
Abbreviations
Blds. Boulders.
Carb. Sh. Carbonaceous Shale.
Cg. Conglomerate.
Cl. Clay.
Clst. Claystone.
Gvl. Gravel.
Ls. Limestone.
Mdst. Mudstone.
P. Wood Petrified Wood.
Slst. Siltstone.
Silty Gvl. Silty Gravel.
Snd. Sand.
Ss. Sandstone.
S.W.
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
THERMAL DIVISION MINING DEPARTMENT
HAT CREEK PROJECT
SECTION 18 (South)
1:1500
Plate A2-5

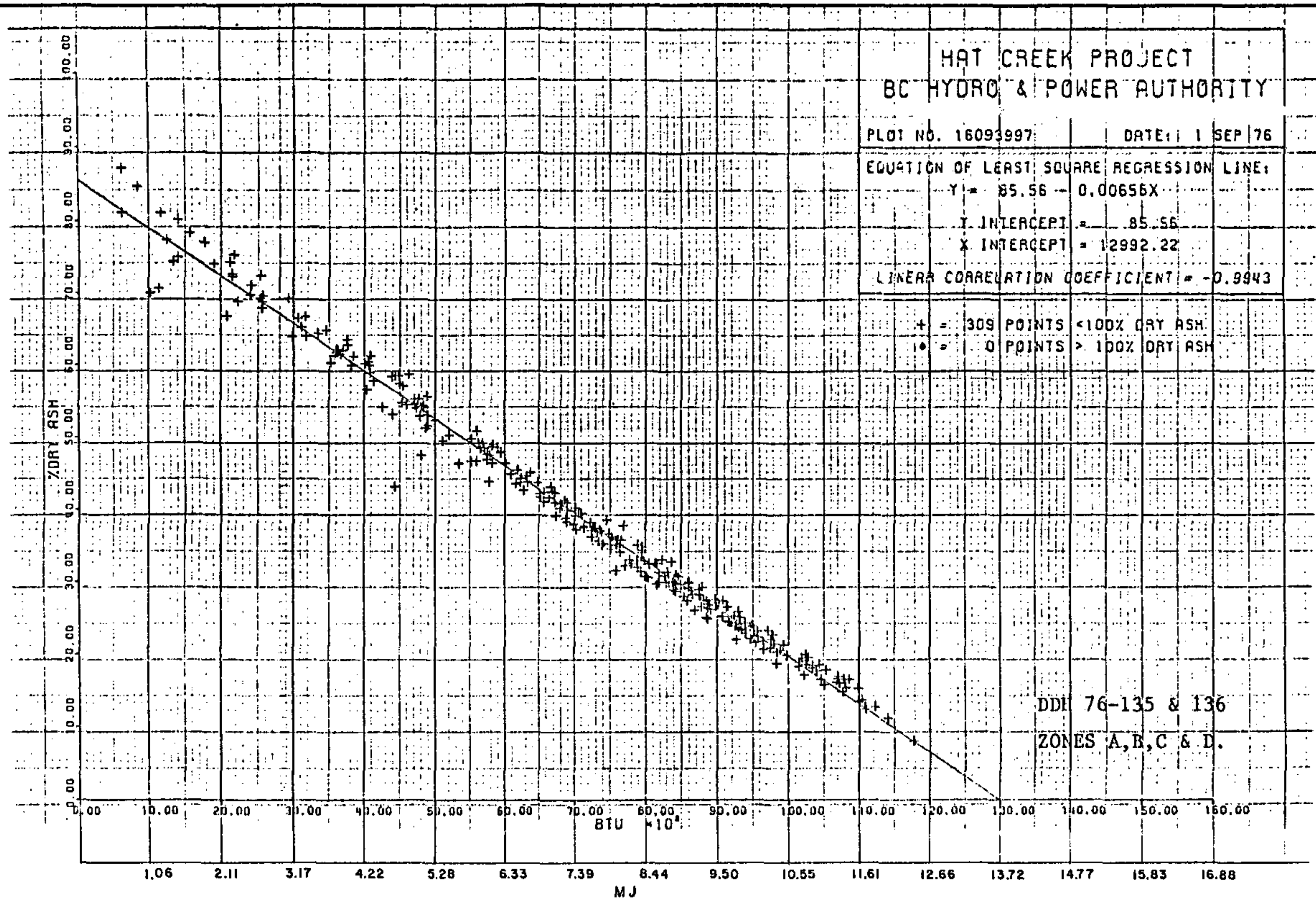


PLATE A2 - 6

ASH - HEATING VALUE REGRESSION CURVE

SECTION A3.0 - COAL RESERVES

All reserve data have been derived from a computer model in which coal quality interpolations were calculated by CMJV on the basis of a modified inverse distance squared procedure.

Table A3-1 summarizes the in situ proven and probable pit reserves, totalling 717 Mt (790 million tons) above the 9.3 MJ/kg (4000 Btu/lb) cut-off grade. According to CMJV these reserves are considered to be well-defined for blocks of coal greater than 20 Mt (22 million tons) but are subject to errors of geological correlation in smaller tonnages.

The total reserves of marginal grade coal between 7.0 and 9.3 MJ/kg (3000 and 4000 Btu/lb) are estimated to be 83 Mt (90 million tons). Of this material 16 Mt (17.6 million tons) are contained in the proposed 35-year pit.

A3.1 RESERVE ESTIMATING PARAMETERS

(a) In Situ Moisture

An in situ moisture content of 25 percent was estimated.

(b) Specific Gravity

A regression equation relating specific gravity of coal to ash content was developed as follows:

Specific gravity of coal = 1.1704 + (.009577 x % ash, dry basis)

(c) Cut-off Grades

Coal	9.3 MJ/kg (4000 Btu/lb)
Low Grade Coal	7.0 to 9.3 MJ/kg (3000 to 4000 Btu/lb)
Waste	7.0 MJ/kg (3000 Btu/lb)

A3.1 RESERVE ESTIMATING PARAMETERS - (Cont'd)

(d) Dilution and Mining Losses

1. Dilution - 2.5 percent weight of material having no heat value.

2. Mining Losses (mining and handling) - 1 percent of mined tonnage.

(e) Partings

The waste partings in each sub-zone are included in the coal reserve estimations, selective mining of partings having been studied and the results reported elsewhere in this report.

A3.2 AS MINED RESERVES IN 35-YEAR PIT (DILUTED)

In applying the above factors to the proposed 35-year pit, the estimated minable reserves by zone are as follows:

Coal Zone	Million tonnes	(tons)	% of Total	Dry Basis Heating Value MJ/kg	(Btu/lb)	Ash %	Sulphur %
A-1	78.61	(86.5)	22.5	12.72	(5473)	49.3	0.70
B-1	58.09	(63.9)	16.6	16.71	(7188)	37.0	0.66
C-1	9.32	(10.3)	2.7	12.53	(5390)	46.8	0.48
C-2	51.99	(57.2)	14.9	13.95	(6002)	44.2	0.41
D-1	151.48	(166.2)	43.3	20.73	(8918)	26.1	0.31
Total	349.49	(384.5)	100.0				
Average				17.03	(7327)	36.3	0.48

Corresponding waste and low-grade coal quantities are 443 million bank cubic metres (MBCM) and 9 MBCM (16 Mt or 17.6 million tons).

The average stripping ratio for the 35-year mine is 1.3 BCM of waste and low-grade coal per tonne of coal delivered.

A3.3 VERIFICATION OF COAL RESERVE CALCULATIONS

Independent manual checks of computer produced data were performed for CMJV and tonnages and grade differences were found to be within acceptable limits by CMJV.

A statistical study indicated that the actual average heating values for the various zones will fall within the following confidence ranges of the average values given above:

A Zone	±700 kJ/kg (±300 Btu/lb)
B and C Zones	±1400 kJ/kg (±600 Btu/lb)
D Zone	±470 kJ/kg (±200 Btu/lb)

Regional coal quality trending has been recognized and incorporated in the computer model by the method chosen to interpolate the data between drill holes. Thus, the actual deviations are expected to be less than those noted in those zones displaying pronounced trends.

TABLE A3-1

In Situ Proven and Probable Pit Reserves in No. 1 Deposit

	Million Tonnes*	Million Tons	% of Total	MJ/kg	Calorific Value (Btu/lb)	Ash Content (%)	Sulphur Content (%)
PROVEN PIT RESERVES							
35-year pit reserves cut-off > 9.3 MJ/kg (4000 Btu/lb), undiluted, dry basis.							
Zone A-1	77.5	85.3	22.5	13.0	5613	47.8	0.72
A-2						—	—
B-1	57.2	62.9	16.6	17.1	7373	35.6	0.68
C-1 C-2	60.4	66.4	17.6	14.1	6061	44.4	0.44
D-1	149.1	164.0	43.3	21.3	9147	24.5	0.31
Total	344.2	378.6					
Weighted Average				17.5	7515	35.1	0.49
PROBABLE PIT RESERVES							
beyond 35-year pit, calorific value cut-off > 9.3 MJ/kg (4000 Btu/lb), undiluted dry basis.							
Zone A	139.5	153.5	37.4	12.1	5227	50.0	0.69
B	66.8	73.5	17.9	14.7	6310	43.6	0.72
C	31.6	34.8	8.5	12.0	5157	51.1	0.43
D	134.9	148.4	36.2	20.1	8627	27.9	0.30
Total	372.8	410.2					
Weighted Average				15.4	6645	40.9	0.53
TOTAL PROVEN + PROBABLE PIT RESERVES							
Calorific value cut-off 9.3 MJ/kg (4000 Btu/lb), undiluted, dry basis.							
Zone A	217	239	30.3	12.5	5365	49.2	0.70
B	124	136	17.3	15.8	6800	39.9	0.70
C	92	101	12.8	13.4	5750	46.7	0.42
D	284	312	39.6	20.7	8900	26.1	0.31
Total	717	788					
Weighted Average				16.4	7060	38.0	0.51

*Specific gravities used to compute tonnages reflect in situ moisture. The average in situ moisture is 25% for the total in place reserves.

SECTION A4.0 - MINE DESIGN

A4.1 MINE DESIGN PARAMETERS

(a) Powerplant Requirements

Based on the powerplant operating regime, the corresponding fuel requirements to cover the period from pre-production to the end of operating year 35 were established.

(i) Powerplant Needs at Target Quality

The powerplant needs based on a target quality of 17.1 MJ/kg (7375 Btu/lb), dry basis and 24 percent moisture are as follows:

Year	Boiler Units	Net Capacity (MW)	Average Capacity Factor (%)	Million Tonnes At 17.1 MJ/kg Dry Basis and 24% Moisture
Pre-Production				1.0[1]
1	1	500	69	3.0
2	2	1000	60	5.3
3	3	1500	60	8.4
4	4	2000	61	10.4
5	4	2000	65	11.0
6-15	4	2000	70	11.1/year
16-25	4	2000	65	10.3/year
26-35	4	2000	55	8.8/year

[1] Required for commissioning Unit No. 1 and for initial stockpile.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

It should be noted, however, that the powerplant equipment would be capable of continuous operation for periods of up to 6 months at maximum continuous rating on all four units. The mine design should be such that this full-load coal demand could be satisfied.

(ii) Allowable Coal Quality Variations

The allowable variations in the quality of coal delivered to the powerplant are summarized as follows:

1. Instantaneous fluctuations of ±350 kJ/kg (150 Btu/lb) without prior notice are acceptable.

2. Fluctuations of greater than ±350 kJ/kg (150 Btu/lb) with a minimum of 1 hour's notice are also acceptable provided the quality of the blended coal is not less than 16.3 MJ/kg (7000 Btu/lb) dry basis.

To determine the equivalent annual coal tonnages for varying heating values and/or moisture contents, the heating value input multiplier factor must be used. See Plate A4-1.

(b) Material Delivery Points and Plant Site

(i) Coal

The responsibility of the mine with regard to the preparation and delivery of fuel grade coal ends at a clearly identified delivery point at the powerplant (i.e. the discharge of the second flight of the overland conveyor at the powerplant fence).

(ii) Low Grade Coal

There is a possibility of beneficiating lower grade coal in the future. The low grade coal stockpile and its location would be based upon economic and environmental considerations and proximity to the powerplant.

(iii) Waste Material

The Houth Meadows and Medicine Creek areas, located outside the limits of the No. 1 and No. 2 Deposits were identified as the main waste dumps. Relatively small areas around No. 1 Deposit would be utilized for the temporary storage of topsoil.

(iv) Plant Site

Any permanent structure for plant and maintenance should be 300 m approximately (1000 ft) from the rim of the ultimate pit and not overlying any coal. Potential areas are located between the No. 1 Deposit and the Indian Reservation and an area flanked by the base of the Medicine Creek dump, the eastern limb of No. 1 Deposit and Medicine Creek.

(c) Geotechnical Parameters

Four main categories of slopes were investigated - final pit walls, dynamic slopes, waste dumps, and total resource pit slopes. Recommended slopes for each of these categories are shown graphically on Plates A4-2 and A4-3.

(i) Final Mine Wall Slopes

Based on the concept of rebound from series relief and assuming stability monitoring, the following

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

pit slopes were found geotechnically acceptable and have been adopted in mine planning.

1. Bedrock in the pit, clockwise from Section Q on the west around to A zone in the southwest, 20^{o}.

2. B, C and D zone coal in the southwest, 25^{o}.

3. On the west side of the D zone coal, between Section Q and the D zone, a transition from 25 to 20^{o}.

4. In the A zone coal on the west side of B zone coal, a transition from 25 to 20^{o}.

5. Surficials, except in active slide areas, 25^{o}.

6. Surficials in active slide areas, 16^{o}.

(ii) Dynamic Slopes

A. Truck and Shovel Pit

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

2. Dynamic slopes angles of 30^{o} are considered to be acceptable in strong bedded material where the bench alignment follows the above guidelines.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

3. In the event the dip of the bedding is less than 30^{o} and out of the face, and the strike of the bedding is parallel to or within 20^{o} of the face alignment, the dynamic slopes should be reduced to the slope of the bedding. This precaution is not necessary where the dip of the bedding is less than 20^{o}.

4. Dynamic slopes of 20^{o} in weak waste materials are considered to have adequate short-term stability up to a period of 1 or 2 years.

B. Bucket Wheel Pit

1. Three benches with an aggregate height of 45 m (approximately 125 ft) and face angles of 52^{o} to 55^{o} and an overall slope of 14^{o} to 15^{o} are feasible for dynamic slope design.

2. Maximum height of a mining face that would be excavated at 52^{o} is 20 m (55 ft).

3. Using a southern pit access would allow slopes to be advanced radially, and consequently benches would normally lie oblique to the strike of the bedding.

4. Operating characteristics of bucket wheels requiring flat dynamic slopes, would not pose any problems on short-term stability.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

C. Waste Dumps

1. As a large portion of the waste is made up of very weak materials, retaining embankments must be constructed at the dumping areas.

2. Retaining embankments should be constructed of free draining, homogeneous sand and gravels and should be uncontaminated by bentonitic clay mine wastes. Some size gradation of the materials would be required.

3. In constructing an embankment, the sands and gravels could either be dumped in 10 m (33 ft) lifts from a spreader or placed in thin layers less than 0.3 m (1 ft) using trucks and bulldozers.

4. To prevent weak waste material from sloughing over embankments, slopes should not be greater than 10 horizontal to 1 vertical, where dump materials exceed crest height by 80 m (260 ft) or less, and 20 horizontal to 1 vertical, where greater than 80 m.

5. Due to the weakness of waste materials, it has been assumed that trucks cannot be driven over the dumps, thus requiring the latter to be developed almost entirely through conveyor-spreader operations.

6. Using the graph in Plate A4-3 showing the relationship between bench height and

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

anticipated angle of repose of dumped Coldwater waste rock as a guide, design criteria for determining spreader size are as follows:

a. Based on a safety factor of 1, the spreader would deposit material 20 m (66 ft) below its floor level, operating from the crest at a minimum of 4 m, or 1/5 the 20 m lift, the accepted nearest allowable position from the crest. In addition, it would deposit material to a maximum of 15 m (50 ft) above its occupied floor level making a total of 35 m (115 ft) dump bench height.

b. During normal conditions, the shiftable belt conveyor working in combination with the spreader would be relocated periodically to within 10 m of the lift height from the crest of the newly constructed 20 m lift. A 1 to 2 months' material stabilization period would be required prior to each relocation.

(iii) Ultimate Pit (to El. 450)

Based on the assumptions that an ultimate pit to El. 450 (1475 ft) would be backfilled with waste materials from the No. 2 Deposit and that either the pit slopes would be depressurized or that pore pressure reductions are possible from rebound of materials, the following pit slopes were adopted for planning purposes:

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

1. Bedrock coal and waste for the total circumference of the pit except where the conglomerate formation on the northwest side is encountered, 20^{o}.

2. In the northwest quadrant, the 20^{o} 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.

3. East site surficials in till, sand and gravels, 25^{o}.

4. East side volcanic deposits, 25^{o}.

5. West and south side surficials, 16^{o}. – Slide areas

(d) Hydrology

Provisions are to be made for mine and waste dumps dewatering which are imperative for wall slope stability and environmental considerations. These comprise three functions:

1. Diversion of surface water from working areas.

2. Removal and treatment of surface run-off from operating areas.

3. Removal of groundwater affecting pit walls and other operating areas.

Various dewatering tests were performed and an assessment of all the available data led to the following conclusions:

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

1. It would 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.

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

3. 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.

4. 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/siltsone formations. The low permeability of these rocks could mean that the equalization of the pore pressures would only occur some considerable time after the end of mining.

5. 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.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

(e) Material Characteristics

(i) Specific Gravity

The following linear regression equation was established between ash (as a percentage of coal on a dry weight basis) and the specific gravity of Hat Creek coal:

1. Coal Specific Gravity = 1.1704 + (0.009557 x % ash, dry basis).

2. The specific gravities used for waste materials are:

   a. Waste above bedrock - 2.2.

   b. Bedrock waste (including major partings between coal zones) - 2.0.

(ii) Swell Factors

	As Dug	Uncompacted Stockpile	Machine Compacted
Coal	35%	35%	20%
Waste above bedrock			
- granular surficials	20%	15%	-
- cohesive surficials	30%	25%	-
Bedrock waste	30%	25%	

(iii) Bearing Capacity

A. Conveyorways, Crushing House, Auxiliary Facilities

The in-situ strengths of both surficial materials and bedrock are expected to exceed the minimum specification of 5 kg/cm^2 (70 psi) for foundation support, however further localized tests would be required for final foundation design. Semi-permanent conveyorways close to the northwest slide would not be practical due to the high probability of "soil creep".

B. Haul Roads

Roads on surficial materials have relatively high bearing capacity and would only require minimal preparation to attain uniform gradient. Normal road topping and grading would be required.

Roads on waste rock and coal in-situ are capable of supporting 136 t (150 ton) trucks provided that an adequate sub-base is constructed. As the effective moisture in most bedrock material is below derived values of plastic limits, heavy traffic would likely compact rather than liquify the material.

Roads within the northwest slide area would need special road building technology, taking into consideration "soil creep" and localized "boils" in the bentonitic clays. Soil creep requires construction of a higher standard sub-base and more frequent upkeep resulting in higher localized road maintenance costs. Bentonitic boils will not support heavy loads and must be avoided.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

(f) Cut-off Grades

The following factors influenced the selection of cut-off grades for Hat Creek coal:

1. Maximum utilization of the resource.

2. Supplying an acceptable boiler fuel, especially with regard to heating and sulphur values.

3. Minimizing the total costs to the mine and powerplant.

Two cut-off grades were determined resulting in three categories of mine production. The higher cut-off differentiates fuel from non-fuel and the lower cut-off further subdivides non-fuel according to its ultimate disposal in either waste dump or a low-grade coal stockpile. The higher non-fuel cut-off determined was 9.3 MJ/kg (4000 Btu/lb) and the lower cut-off between low-grade coal and waste was 7.0 MJ/kg (3000 Btu/lb).

(i) Fuel/Non-Fuel Cut-Off Grade

The table below shows the heating value of the delivered fuel as the cut-off grade is varied from 9.3 to 13.9 MJ/kg (4000 to 6000 Btu/lb), the shortfall in coal production from the proposed 35 year pit design, and the resource utilization.

Cut-off Grade		Deliverable Fuel Millions		Deliverable Fuel Heating Value		Present 35 Yr Pit Production Shortfall Millions		% Heating Value Utilized
MJ/kg	(Btu/lb)	Tonnes	(Tons)	MJ/kg	(Btu/lb)	Tonnes	(Tons)	
13.9	(6000)	251.3	(276.4)	19.0	(8184)	59.4	(65.3)	76.3
11.6	(5000)	322.9	(355.2)	17.6	(7563)	14.3	(15.7)	90.6
10.5	(4500)	336.7	(370.4)	17.3	(7444)	6.6	(7.3)	93.0
9.3	(4000)	349.1	(384.0)	17.0	(7327)	-	-	94.9

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

An inspection of this table reveals the following points:

1. A significant improvement in resource utilization as the cut-off grade is reduced from 13.9 to 11.6 MJ/kg (6000 to 5000 Btu/lb).

2. The heating value of the delivered fuel increases from 17.0 to 17.6 MJ/kg (7327 to 7563 Btu/lb) as the cut-off grade increases from 9.3 to 11.6 MJ/kg (4000 to 5000 Btu/lb).

3. The heating value of 17.0 MJ/kg (7327 Btu/lb) is an acceptable quality for the manufacturer's design of the boilers.

4. The increase in heating value from 17.0 to 17.6 MJ/kg (7327 to 7563 Btu/lb) would not result in significant cost savings in the boiler design.

5. At a cut-off grade of 9.3 MJ/kg (4000 Btu/lb) the total fuel demand of the powerplant during the 35-year period would be satisfied while at the higher cut-off grade additional coal must be mined resulting in higher mining costs.

From these observations it can be seen that a cut-off grade of 9.3 MJ/kg (4000 Btu/lb) would produce an acceptable fuel to the powerplant and reflects the

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

lowest cost to the mine and powerplant. At this cut-off grade, 95 percent of the resource would be utilized. Studies of the effect on sulphur values by varying the cut-off grades also revealed that there is no significant effect on sulphur levels as the cut-off grades are varied:

Fuel Cut-off MJ/kg	(Btu/lb)	Diluted Fuel % Sulphur	Coal Quality Heating Value MJ/kg	(Btu/lb)	Lbs. of Sulphur Per Million Btu
8.1	(3500)	0.48%	16.8	(7248)	0.66
9.3	(4000)	0.48%	17.0	(7327)	0.65
10.5	(4500)	0.48%	17.3	(7444)	0.64

(ii) Low-Grade Coal/Waste Cut-off Grade

Low-grade coal was determined as that portion of mine production with a heating value between 7.0 and 9.3 MJ/kg (3000 and 4000 Btu/lb) dry basis. It would be stockpiled for possible future use.

The sensitivity of the 7.0 MJ/kg (3000 Btu/lb) waste cut-off level was examined by evaluating the effects on the quantity and quality of the low-grade coal stockpile as the cut-off level was varied. The following table demonstrates that raising the cut-off grade to 8.1 MJ/kg (3500 Btu/lb) may commit an additional 8.1 Mt (8.9 million tons) of coal with about 1 percent of the total thermal content, to the waste dumps. Conversely, lowering the cut-off to 5.8 MJ/kg (2500 Btu/lb) would add about 9 Mt (9.9 million tons) with less than 1 percent of the total thermal content to the low-grade coal stockpile at considerable cost. The average quality of the low-grade coal stockpile varies from 7.4 to 8.7 MJ/kg (3200 to 3745 Btu/lb) for this range of low-grade coal/waste cut-offs.

Low-Grade Coal-Waste Cut-off Grade		Low-Grade Coal Stockpile Millions		Heating Value of Low-Grade Coal Stockpile		Percentage of Total Heat Resources
MJ/kg	(Btu/lb)	Tonnes	(Tons)	MJ/kg	(Btu/lb)	
5.8	(2500)	24.7	(27.1)	7.4	(3200)	2.8%
7.0	(3000)	15.7	(17.3)	8.1	(3491)	2.0%
8.1	(3500)	7.6	(8.4)	8.7	(3745)	1.0%

The merits of maintaining the low-grade coal stockpile will be reassessed, in view of the low heating value of the low-grade coal stockpile, and the fact that after 35 years of operation, the unmined Hat Creek energy resource would still be far greater and less expensive to recover than the stockpile.

(g) Dilution and Mining Loss

Factors determined for mining loss and dilution are:

1. Dilution at 2.5 percent by weight of diluted coal. The diluent is assumed to have zero heating value.

2. Mining loss at 1 percent of the diluted coal as mined.

Diluents are defined as unavoidable waste materials mined with fuel-grade coal excluding partings within those subzones classified as coal; and the shaled-out coal, contained primarily in C zone, occuring in the west limb of the deposit which is classified as low-grade coal or waste.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

(i) Dilution

The study of dilution applied the concept that the weight of diluent that would be mixed with the coal is a function of the surface area of the coal/waste subzone interface and the angular attitude of this interface. All contact areas were measured and classified as to attitude - either less than or greater than 15^0. 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	0.3 m (1.0 ft)
greater than 15^0	0.7 m (2.3 ft)

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

1. All sloughs of waste coal benches and not completely cleaned up - 20 000 t/yr (22,000 tons/yr).

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

2. All road building materials laid down on coal benches and incompletely removed - 10 000 t/yr (11,000 tons/yr).

The results of diluent quantity calculations are provided in the following table. Nominal time periods are not directly related to the powerplant schedule.

Period of Pit Development	Coal Million Tonnes	(Tons)	Diluent Tonnes	(Tons)	% Dilution
-1- 5 yrs	22.1	(24.3)	.6	(.66)	2.6
6-10 yrs	44.8	(49.3)	1.2	(1.32)	2.6
11-15 yrs	62.2	(68.4)	1.5	(1.65)	2.4
16-25 yrs	115.4	(126.9)	2.5	(2.75)	2.1
26-35 yrs	98.3	(108.1)	2.2	(2.42)	2.2
Total	342.8	(377.0)	8.0	(8.8)	2.3

An additional 10 percent 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 an average value of 2.5 percent dilution would be applied throughout the pit development.

(ii) Mining Loss

The following day-by-day operating situations were considered to constitute mining losses of the coal reserves:

1. A token loss of oxidized coal.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

2. Coal lost with waste removed at coal/waste interfaces.

3. Errors in dispatching coal to waste dumps.

4. Degrading of coal during ground sloughs to such an extent that it would be dispatched to the waste dumps.

5. Losses from dusting of fine coal and spillages during transportation.

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

A. Oxidized Coal

West side of pit - a 1 m (3 ft) 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 m (10 inch) loss over all coal subcrops.

B. Coal at Interfaces with Waste

A 0.25 m (10 inch) loss was assumed where coal interfaces with hanging wall rocks, footwall rocks, A-2 and C-1 zone waste bands.

C. Operational Losses

The combined loss from all other causes, i.e. human errors, wind losses, admixing with waste sloughs, etc., was estimated at 25 000 t/yr (27,500 tons/yr).

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

From the study results shown below it was decided that the average value of 1 percent loss would be applied throughout the pit development.

Development Period Years	Coal Produced	Coal Loss (Mt)				% Loss
		Oxidized	Dilution	Other	Total	
0- 5	22.1	0.2	0.1	0.1	0.4	2.1
6-10	44.8	0.2	0.3	0.1	0.6	1.3
11-15	62.2	0.2	0.3	0.1	0.6	1.0
16-25	115.4	0.3	0.5	0.3	1.1	1.0
26-35	98.3	-	0.6	0.2	0.8	0.8
Total	342.8	0.9	1.8	0.8	3.5	1.0

(h) Partings Removal

Within the zones the coal is interlain with waste partings of variable thickness. If these partings could be removed separately from the coal by selective mining and disposed of as waste then the quality of the run-of-mine coal would be improved. A study was undertaken to assess the possible improvement in coal quality by selective mining. This investigation was limited to the A zone which had the greatest potential to effect fuel upgrading.

The improvement in coal quality in the A zone by the removal of all partings is shown in the following table. This evaluation was carried out on the A zone coal to be mined during five periods of the recommended pit development.

A4.1 MINE DESIGN PARAMETERS - (Cont'd)

Pit Development Period (Years)	A Zone Coal Mined Million Tonnes	(Tons)	Heating Value of A Zone Coal With No Partings Removed Dry Basis MJ/kg	(Btu/lb)	Heating Value of A Zone Coal With All Partings Removed Dry Basis MJ/kg	(Btu/lb)	Grade Improvement by Removal of All Partings Dry Basis MJ/kg	(Btu/lb)
1- 5	10.32	(11.35)	12.51	(5384)	13.79	(5931)	1.27	(547)
6-10	10.33	(11.36)	12.59	(5418)	20.12	(8658)	7.53	(3240)
11-15	14.21	(15.63)	12.84	(5522)	18.61	(8007)	5.78	(2485)
16-25	30.28	(33.31)	12.74	(5482)	16.94	(7290)	4.20	(1808)
26-35	13.47	(14.82)	12.80	(5507)	23.10	(9941)	10.31	(4434)
Total	78.61	(86.47)	12.72	(5472)	18.30	(7875)	5.59	(2403)

It would be impractical to achieve 100 percent removal of partings because of the following reasons:

1. Some difficulty would be experienced in recognizing and differentiating coal from waste in certain areas of the deposit.

2. The thickness and slope of some partings may render them impractical and/or uneconomical to be selectively mined.

3. There would be some degree of inaccuracy in the correlation of partings from the drill hole data available.

To accurately evaluate the degree of partings removal that could be attained during mining operations a comprehensive study would be required to assess such factors as:

1. The additional mining loss to be experienced due to the increased number of coal/waste interfaces.

A4.1 <u>MINE DESIGN PARAMETERS</u> - (Cont'd)

2. The impact on estimated coal tonnages within coal zones. With composite heating values presently assigned to zones, some blocks now considered as all waste may prove to have minable quantities of coal, and some blocks presently considered as all coal may in fact have significant quantities of waste.

3. The additional cost of mining if selective mining of partings is undertaken.

4. With a possible reduction in the range of coal qualities mined, as a result of selective mining, there would be an effect on the design and cost of the stockpiling and blending facilities.

5. The development of the mine may have to be altered for more effective selection of waste partings.

6. The approach to the mine development may also be altered if significant improvements are possible in the coal grades of the A and C zones. The need to mine significant quantities of the higher grade D zone coal in the earlier years may be reduced. This could have an impact on the mining method employed for the mine development and also improve the operating conditions of the mine from a geotechnical viewpoint.

Since potential benefits may be derived from partings removal and a more detailed geological representation of the deposit will be available at a later stage of the project, a comprehensive evaluation of the selective mining of partings would be carried out at that time.

A4.2 MINING METHODS

The following six alternative mining systems were identified:

1. Shovel/truck.
2. Shovel/truck/conveyor.
3. Shovel/conveyor.
4. Bucket wheel excavator/conveyor.
5. Continuous excavator/truck and/or conveyor.
6. Dragline/truck and/or conveyor.

From this list two systems were determined to be the most practical. They were the shovel/truck/conveyor and the bucket wheel excavator/conveyor systems.

In order to deliver a consistent fuel quality to the power-plant, coal from the upper zones has to be blended with the relatively higher grade D zone coal. Since most of the higher grade D zone coal occurs at depth, this necessitates a rapid deepening of the pit during pre-production and the first 5 years of production. To maintain a relatively low stripping ratio the pit was designed with the minimum possible lateral excavation of the upper benches, while satisfying the recommended pit slopes and other mining and geotechnical constraints. Such a mining approach requires that the bucket wheel excavator/ conveyor system be augmented by a shovel/truck system to enable simultaneous mining operations on a number of working benches and in the higher grade coal at the bottom of the pit. The alternatives for further study were therefore altered to a pure shovel/truck/conveyor system and a combined bucket wheel excavator/shovel/truck conveyor system. North American Mining Consultants (NAMCO) were retained to assess the feasibility of the bucket wheel excavator and conveyor system for developing the deposit.

NAMCO was unable to optimize their study on the application of bucket wheel excavators and conveyor systems to develop the Hat Creek mine, due to the level of planning data developed at this stage of the study. The study of the combined system was more involved than that of a system employing primarily one type of mining equipment. Since NAMCO only studied the application of bucket wheel excavators and conveyor systems, further work is required to carry out a sensitivity analysis to achieve the optimum mix of the two subsystems and the correct timing and location of each of the subsystems. Nevertheless, the combined system that was investigated gave similar costs to the shovel/truck/conveyor system which was selected based on a greater degree of confidence in the more detailed study of the shovel/truck/conveyor system. The possibility of incorporating bucket wheel excavator systems to work with the shovel/truck system should be investigated further. This combined system is not expected to alter significantly the overall economics of the project, however it could provide operating advantages and improve selective mining capability.

(a) Combined Bucket Wheel Excavator/Shovel/Truck/Conveyor System

A 35-year open pit mine for the combined bucket wheel/shovel/truck/conveyor system was designed with a pit bottom elevation of El. 682 (2224 ft). The pit was designed in accordance with the final slope angles of Plate A4-2. The mass calculations of this pit provided the following results:

A4.2 MINING METHODS - (Cont'd)

Total volume of materials	-	726 Mm^3
Total volume of waste	-	483 Mm^3
Tonnage of coal	-	332 Mt (365 M tons)

(with an average heating value of 17.2 MJ/kg (7390 Btu/lb) undiluted dry basis)

Waste:coal ratio	-	1.46:1 (m^3:t)
Total volume of low grade coal	-	10 Mt (11 million tons)

(heating value of 7.0 to 9.3 MJ/kg (3000 to 4000 Btu/lb) dry basis)

The distribution of total material quantities to be mined by the two subsystems were as follows:

Bucket wheel excavator and conveyor subsystem	-	442 Mm^3
Shovel/truck subsystem	-	284 Mm^3

The production schedule had a pre-production period of 2 years during which time 22 Mm^3 of material would be mined. The annual material movements during the 35-year life of the mine are summarized as follows:

1. Years 1 to 5 - annual material movement varied from 24 to 35 Mm^3.

2. Years 6 to 10 - average annual material movement of 31 Mm^3.

A4.2 MINING METHODS - (Cont'd)

3. Years 11 to 25 - average annual material movement of 21 Mm^3.

4. Years 26 to 35 - average annual material movement of 9 Mm^3.

This production schedule requires the mining of a very large volume of material during the first 10 years of operation. A rescheduling of mining could produce a more even annual production volume.

The main equipment employed during the peak production years would be:

(i) Bucket Wheel Excavator Subsystem

Four bucket wheel excavators with a nominal bucket size of 850 L and 51 discharges per minute.

Bucket wheel diameter across cutting lips	-	10.5 m (34.5 ft)
Number of buckets	-	12
Theoretical output	-	2600 m^3/h loose
Belt width	-	1400 mm (55 in)
Belt speed	-	4 m/s (780 fpm)
Travel speed	-	9 m/min (30 fpm)
Mean ground pressure	-	16.5 N/cm^2
Annual output	-	5.5 MBCM

Belt wagons would be designed to work in conjunction with each bucket wheel excavator. These belt wagons would bridge a distance of 80 m (262 ft) between the bucket wheel discharge and the conveyor loading point. This would be accomplished by two independent hydraulically operated booms each 41 m (135 ft) in length. Belt width and speed would be the same as for the bucket wheel excavators.

(ii) Shovel/Truck Subsystem

The shovel/truck subsystem would be employed primarily for the preparation of working levels for the bucket wheel excavator system, the development of the uppermost benches and for mining operations at the bottom of the pit where bucket wheel excavators cannot be effectively utilized.

Two shovels with 16.8 m^3 (22 yd^3) buckets supplemented in peak production periods by two large front end loaders with 11.5 m^3 (15 yd^3) buckets would be the main loading equipment scheduled for this system. Seven 136 t (150 ton) waste trucks and three 109 t (120 ton) coal trucks would be scheduled to haul material to two sets of loading pockets located on the inclined conveyors constructed on the main exit ramp.

Material mined by the bucket wheel excavator and the truck/shovel systems would be transported via steel-cord belt conveyors to the waste dumps, low grade coal stockpile and the stockpile/blending facilities. Plate A4-4, shows the general layout of this combined system. The combined system would be designed with six belt conveyor systems on the central outgoing ramp, two of which would be for the materials mined by the shovel/truck system. Each of these six conveyors would be capable of handling the theoretical output rates of the bucket wheel excavators, i.e. 2600 m^3 (loose)/h (3400 yd^3/h). The conveyor belts would be 1200 mm (48 in) wide and have a speed of 5.2 m/sec (1000 fpm).

A distribution point would be located at the top of the central outgoing ramp. At this point material from the six inclined conveyors would be distributed to either of three conveyors by installing extendable head stations on the inclined conveyors. Of the three conveyors leaving the distribution point one would carry coal and the other two waste. The coal conveyor would deliver all coal from the mine to the stockpile/blending area via the crushing plant, and the two waste conveyors would handle either waste, embankment material or low grade coal. These three conveyors would be sized at approximately twice the capacity of the inclined conveyors, since a maximum of two loading machines would be expected to be mining coal while two to four loading machines could be mining waste, embankment material and/or low grade coal.

A major difference between the combined system and the pure shovel/truck system is that Medicine Creek would be the main waste disposal area for the combined system. Material from the slide area and the upper benches in the northern end of the pit would be placed in the Houth Meadows dump. A detailed plan was not developed for the out-of-pit conveying systems, the dump construction and the support facilities of the combined system. This plan would be completed when further investigations are carried out on the combined system.

(iii) South Exit

During the preparation of the bucket wheel planning data in February 1978, new geotechnical information was received. This information referred to the slide area in the northwestern end of the No. 1 Deposit and defined the slide material by means of a topographic map and cross-sections. This slide material was shown to extend down to the top of the underlying conglomerate. A design constraint was also imposed on NAMCO's planning which stipulated that during the development of the mine, conveyor transfer points and stationary conveyors could not be located in this slide material.

The implication of this design constraint was that it would no longer be possible to develop the No. 1 Deposit with the main exit ramp in the northern end of the pit. It would be necessary to locate the main exit ramp in the southeastern end of the pit. The development of the pit from this new exit ramp would not impose any added difficulties to the pit design itself and would offer the following advantages.

1. The development of the Hat Creek No. 1 Deposit by a combined system with a southeastern exit ramp necessitates a large portion of the development to be carried out by an advancing mine face, the orientation of which would not be parallel to the

strike of the coal thereby resulting in more more stable mine faces.

2. Mining benches could be simultaneously established north and south of the Hat Creek Valley with limited preparation work and a simple and relatively short conveyor layout.

3. The main access to the mine would not be endangered by its proximity to the active slide area. Also, the face conveyor would be perpendicular to the active slide area and therefore not subject to the dangers of instability of the active slide area.

4. The primary waste dump would be located in the Medicine Creek waste disposal area. This dump would require much less embankment material than the Houth Meadows waste disposal area and therefore would not impose a restriction on the method of pit development in order to satisfy the embankment material requirements.

5. The layout of the conveying system and the use of the Medicine Creek waste disposal area could be maintained if the deposit is developed beyond the 35 year period. Back dumping in the mined out area of the No. 1 Deposit could be possible after the Medicine Creek waste disposal area is filled.

6. The out-of-pit mining facilities would not be as congested if located at the southern end of the

pit. The administration office, blending facilities, maintenance facilities, etc. would all be centrally located with respect to the coal deposit.

7. With a southern exit ramp the main waste dump, coal blending stockpiles and other mining facilities would be further removed from Highway 12.

8. Overland transportation of coal to the site selected for the powerplant would be no less favourable than with the northern access.

The major disadvantage of the southern exit would be that significant quantities of better grade coal are closer to surface at the northern end of the deposit. The earlier development of this portion of the deposit from the south would therefore result in higher initial stripping ratios. However, with the combined system the additional waste stripping required in the south would be reduced by utilizing working benches which slope downwards to the north and by rotating the face conveyors about the southeastern end of the pit.

More detailed studies of selective mining may also reveal that the initial development of the better grade coal in the north would not be as important as it now appears.

The relocated major exit ramp would have a more significant impact on other aspects of the mine design such as the location of the waste dump and blending facilities, the location of the maintenance facilities and the design of the conveyor system, roadways and drainage systems (in particular the Hat Creek diversion outside the pit area). To evaluate and incorporate fully this major design change into all aspects requires significant additional work at a later stage.

(b) Shovel/Truck/Conveyor System

A number of manual long range pit plans were developed and the selected scheme shown in Plate A4-5 incorporates the following major concepts:

1. A shovel/truck system working 15 m (50 ft) high benches to feed into unloading stations located over the central conveyor system:

2. 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.

3. The sufficient and continuous exposure of better grade D zone coal necessary for coal blending.

4. A northern exit from the mine with three conveyor systems for waste, coal and construction material/low-grade coal, respectively.

5. A crushing, sampling, and blending facility adjacent to the mine services area close to the northern pit exit.

6. Delivering blended coal to the powerplant by overland conveyor.

A4.2 MINING METHODS - (Cont'd)

7. Conveying and spreading systems to dispose of waste in the Houth Meadows and Medicine Creek dump areas.

8. Separate low-grade coal stockpile ranging from 7.0 to 9.3 MJ/kg (3000 to 4000 Btu/lb), dry basis, for future use.

Basic pre-production plans were prepared to open up the deposit and to ready the mine for coal production. A series of incremental pits were then designed to obtain sufficient suitable coal to satisfy powerplant requirements while at the same time maintaining a practical and economic stripping ratio. From the incremental pits, a production schedule was developed.

The average blended coal quality over the 35-year life is listed below:

Ash content, dry basis	36.3 percent
Heating value, dry basis	17.0 MJ/kg (7327 Btu/lb)
Moisture, as delivered	25 percent
Heating value, as delivered	12.8 MJ/kg (5495 Btu/lb)
Ash content, as delivered	27.2 percent

This coal quality is slightly below target, however, the total heat requirements of the powerplant have been maintanined by an increase in the overall coal tonnage. The table below reflects the quantity of run-of-mine coal, 350 Mt (385 million tons) required by the powerplant after the variations for coal quality have been considered.

A4.2 MINING METHODS - (Cont'd)

Year	Powerplant Needs at Target Quality 17.1 MJ/kg (7375 Btu/lb) Dry Basis and 24% Moisture Mt	Powerplant Needs at Predicted Quality Mt	Total Heat Required at Predicted Quality MJ x 10^{12}	(Btu x 10^{12})
Pre-Production	1.0	1.03	13.2	(12.5)
1	3.0	3.08	39.0	(37.0)
2	5.3	5.43	69.0	(65.5)
3	8.0	8.20	104.5	(99.0)
4	10.4	10.66	135.5	(128.5)
5	11.0	11.30	144.5	(137.0)
6-15	111.0	113.76	145.5	(138.0)
16-25	103.0	105.96	140.0	(128.0)
26-36	88.0	90.07	115.0	(109.0)
Totals	340.7	349.49	4559.2	(4324.2)

Alternative material handling systems were developed. The scheme selected is a conveyor complex linking the mine to the various dumps, stockpiles and powerplant and is illustrated in Plate A4-6.

(i) Pre-Production

Several alternatives were considered to open up the coal deposit and prepare the mine for production. These were:

1. Exposing the western limb of the A, B, C and D zones at higher levels.

2. Uncover better-than-average grade D zone coal between Nos. 7 and 8 fault at the north end of the No. 1 Deposit at lower elevations.

3. Uncover the eastern limb of the No. 1 Deposit.

4. A combination of the above three alternatives.

Opening up in better-grade coal, scheme 2, was selected as the most suitable alternative. It would provide the least amount of pre-production waste since the coal in the north is near surface. In anticipation of mine and plant site construction requirements more of the surficial granular materials would be scheduled for mining during the pre-production period. Much of this material would be used to construct development access roads and pads for the Houth Meadows waste conveyors.

The first unloading station and the corresponding first leg of the main conveyor belt were scheduled for construction during this period.

The pre-production period was estimated to be 3 years prior to commissioning the first generating unit and allowed for start-up problems, training of personnel, delivery of equipment and production build-up.

1 Mt (1.1 million tons) of coal with a heating value of 17.4 MJ/kg (7484 Btu/lb), dry basis, would be mined during pre-production. A portion of this would be used to commission the first 560 MW unit and

the remainder stockpiled. 20 Mm^3 of waste and overburden would be mined within the 3 year pre-production period.

(ii) 35-Year Pit

The 35-year pit, Plate A4-5, includes a suitable road network, safety berms, conveyor belt ramps and delivery points. The design parameters were adhered to and the dominant features (i.e. nature of coal deposition and slide areas) were either utilized or practical schemes were provided to reduce possible operational problems. The pit bottom would be at El. 647.5 (2125 ft) and contain sufficient reserves to satisfy powerplant fuel requirements for 35 years.

A. Pit Access

A northern access is recommended for the pit for the following reasons:

1. The better quality coal which is close to surface at the north, could be developed in the initial years at a low stripping ratio.

2. The better quality coal at the north overlies competent footwall rocks, and the angle of plunge of the structures in this area is less than the maximum wall slopes recommended by the geotechnical consultants.

3. The natural structural ridge of footwall waste between No. 7 and 8 faults is an ideal site for the main conveyor ramp and the truck

unloading stations, since these would not be affected by future mining operations.

4. The northern exit would permit maximum utilization of the Houth Meadows dumping site which would involve the least vertical transportaton of material.

Disadvantages of the northern exit are:

1. Proximity to the northwest active slide with its potential for failure and subsequent extensive clean-up requirements.

2. The northwestern truck exit would be partially located in materials comprising the inactive slide and higher road maintenance costs are anticipated due to soil creep.

The risk of the central conveyor operation being halted due to earth movements of any kind is considered to be low, but provision would be made for several truck and service road entries. In addition, haulage road widths would be increased from a pit standard of 30 m (100 ft) to 60 m (200 ft) in the slide area to provide room for clean-up should sloughs occur. Pit standards include:

1. Haul roads - 30 m (100 ft) wide - maximum ramp gradient 8 percent.

2. Service roads - 20 m (66 ft) wide - maximum gradient 10 percent.

3. Safety berms in the contact zone between surficials and bedrock 30 m (100 ft) wide - maximum gradient 10 percent (60 m (200 ft) wide below slide surficials).

B. Slide Area

It is proposed to defer mining in the active slide area as long as is practicable to facilitate a better appreciation of the problems which might arise. A semi-permanent haul road, located sufficiently south of the slide area, would be constructed during initial operating years to provide northwest access to the upper benches.

C. Truck Unloading Stations

The vertical locations of the truck unloading stations, designed to feed the central conveyor, were determined on the following factors:

1. To have minimum hauling distance from the mining faces to the hoppers.

2. To have minimum uphill truck haulage.

3. That flow of material to the loading pockets be consistent and uniform.

The elevations selected for the loading pockets are at El. 895, 820 and 730, each station being developed as the pit develops. Since these stations would be located closest to the centre-of-gravity of development pits the haulage distances would be reduced considerably.

(iii) Ultimate Pit

An ultimate pit to the El. 450 (1475 ft) floor level was developed. The El. 450 floor level was not based on a specific geological bench mark, rather it was considered to display the practical maximum pit dimensions for the No. 1 Deposit.

After 35 years, coal above El. 650 (2130 ft) in the northern portion of the No. 1 Deposit would be mined out. Future expansion, therefore, could be southward laterally and horizontally following the plunge of the coal stratigraphy. The coal supply from this expanded pit would almost double that forecast for the 35 year project. An assessment of the economics of the ultimate pit and its interrelationship with the adjacent No. 2 Deposit would be evaluated at a much later stage.

Most surficial materials suitable for constructing retaining embankments would be mined out during the initial 36 years. The design of the waste dumps provided maximum utilization of suitable granular material in the retaining embankments.

The mineral inventory contained in the ultimate pit is given in Table A4-1.

A4.3 PRODUCTION SCHEDULING

The mining strategy adopted was to initially open on better-than-average grade coal then deepen the pit rapidly. Significant amounts of the higher grade D zone coal would then be available at all times. The rate of lateral development would be controlled by exposing only sufficient amounts of lower grade coal required for blending and maintaining a relatively uniform overall material movement. Another objective was to achieve a low stripping ratio during initial operating years.

To adhere to powerplant requirements, attempts were made to provide for mining a consistent grade of coal over the production years. This goal was successfully achieved with a slight fluctuation between the stage of years 16 to 21 and 22 to 25.

Incremental pits for years 1, 2, 3, 4, 5, 9, 15, 21 and 26 were made in addition to the pre-production mine plans. The 35 year plan (Plate A4-9) shows the configuration of the pit at the end of mining. Plates A4-7 and A4-8 represent years 5 and 15 respectively. Corresponding production statistics for each of the incremental pits shown in Tables A4-2 and A4-3 were graphed; coal tonnages against the combined tonnages of low grade coal, waste and overburden. The annual production schedule, Table A4-4, was then prepared from the graphed information and summarized in Table A4-5. The estimated bench quantities are shown in Table A4-6. Plate A4-10 shows a cross-section of the pit at various stages of development.

TABLE A4-1

Summary of Production Statistics
for the Ultimate Pit
Hat Creek Project Mining Report 1978

	CMJV 35-Year Pit	Increment to El. 450	Total Resource Pit
PRODUCTION MATERIALS IN SITU (Million BCM)			
Coal(>9.3 MJ/kg)	234.32	205.20	439.52
Low-Grade Coal (6.98 to 9.3 MJ/kg)	8.96	38.27	42.23
Wastes above Bedrock	161.15	642.32	803.47
Wastes below Bedrock	281.85	215.04	496.89
COAL IN SITU ABOVE 9.3 MJ/kg			
Tonnage (X Million)	344.19	322.19	666.38
Quality			
Heating Value (MJ/kg)	17.48	14.90	16.23
Ash Content (%)	35.1	42.5	38.7
Sulphur Content (%)	0.49	0.55	0.52

Strip Ratio

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

Increment to El. 450 $$\frac{38.27 + 642.32 + 215.04}{322.19} = 2.8 \text{ to } 1$$

Total Resource Pit $$\frac{42.23 + 803.47 + 49.89}{666.38} = 2.0 \text{ to } 1$$

Relationship to No. 1 Deposit Proven Coal Resources

Total Resource Pit $$\frac{666.38}{716.50} \times 100 = 93\%$$

TABLE A4-2

Incremental Pits - Production Statistics
Without Dilution

Hat Creek Project Mining Report 1978

Year	C O A L Thousand Tonnes	MJ/kg Dry Basis	Thousand m^3	Low-Grade Thousand m^3	Waste Thousand m^3	Overburden Thousand m^3	Total Material Thousand m^3
-2	-					8 600	8 600
-1	1 018	17.85	684	11	321	11 451	12 467
1	2 821	17.41	1 881	115	740	12 589	15 325
2	5 634	17.50	3 765	139	705	12 926	17 535
3	8 211	17.65	5 503	133	1 046	10 878	17 560
4	11 255	17.46	7 517	448	1 545	10 021	19 531
5	17 739	17.44	11 844	1 089	4 884	15 014	32 831
10	46 511	17.50	31 091	621	10 480	43 554	85 746
15	63 475	17.45	42 374	2 369	25 171	45 448	115 362
21	57 773	17.30	38 447	512	38 537	57 674	135 170
26	50 522	17.71	33 846	1 078	39 785	44 568	119 277
35	79 231	17.48	52 886	2 445	42 412	9 127	106 870
Total	344 190	17.48	229 838	8 960	165 626	281 850	686 274

TABLE A4-3

Incremental Pits - Production Statistics
With Dilution

Hat Creek Project Mining Report 1978

Year	COAL Thousand Tonnes	COAL MJ/kg Dry Basis	COAL Thousand m^3	NON-COAL - Thousand Cubic Metres Low-Grade	Waste	Over-burden	Total Non-Coal	S.R. m^3/t	Total Material Moved Thousand m^3
-2						8 600	8 600	-	8 600
-1	1 034	17.41	692	11	313	11 451	11 775	-	12 467
1	2 864	16.98	1 903	115	718	12 589	13 422	4.69	15 325
2	5 721	17.06	3 809	139	661	12 926	13 726	2.40	17 535
3	8 337	17.21	5 567	133	982	10 878	11 993	1.44	17 560
4	11 428	17.02	7 605	448	1 457	10 021	11 926	1.04	19 531
5	18 012	17.01	11 983	1 089	4 745	15 014	20 848	1.16	32 831
10	47 227	17.06	31 455	621	10 116	43 554	54 291	1.15	85 746
15	64 452	17.01	42 870	2 369	34 675	45 448	72 492	1.12	115 362
21	58 662	16.86	38 898	512	38 086	57 674	96 272	1.64	135 170
26	51 299	17.27	34 240	1 078	39 391	44 568	85 037	1.66	119 277
35	80 450	17.04	53 504	2 445	41 794	9 127	53 366	0.66	106 870
Total	349 486	17.04	232 526	8 960	162 938	281 850	453 748	1.30	686 274

TABLE A4 - 4

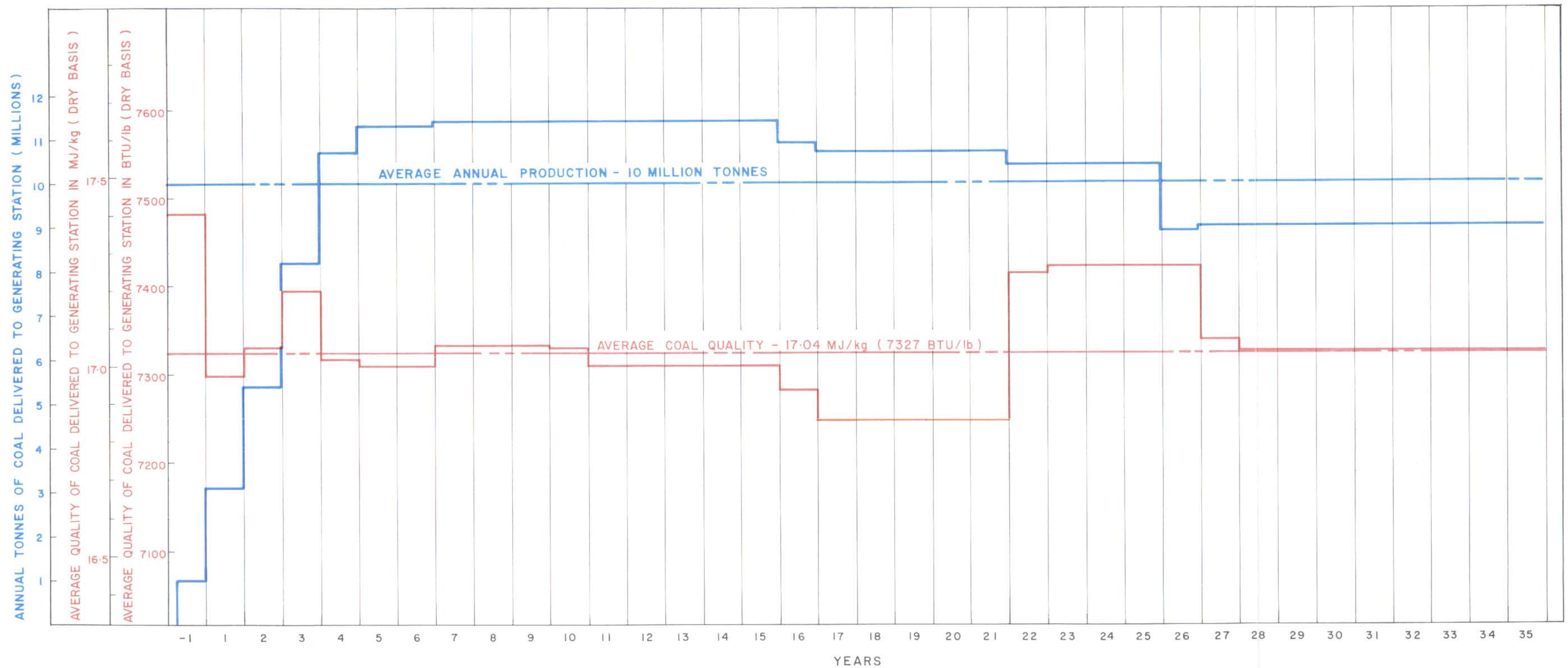
ANNUAL PRODUCTION SCHEDULE

MATERIALS MINED QUANTITIES IN MILLIONS (10^6)	PRE-PRODUCTION YEARS				PRODUCTION YEARS																																			TOTAL
	-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	
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	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)				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	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)				0·01	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	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)				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	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)		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	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)				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	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)		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
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	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)				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	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																																								AVERAGE
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	7250	7250	7250	7250	7417	7424	7424	7424	7424	7341	7328	7328	7328	7328	7328	7328	7328	7328	7327
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	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)				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	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)				0·53	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	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)				0·70	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	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}				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	12·8	12·8	12·8	12·8	12·8	12·8	12·8	12·8	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}				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	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)																																								

TABLE A4 - 5

SUMMARY OF COAL PRODUCTION & QUALITY



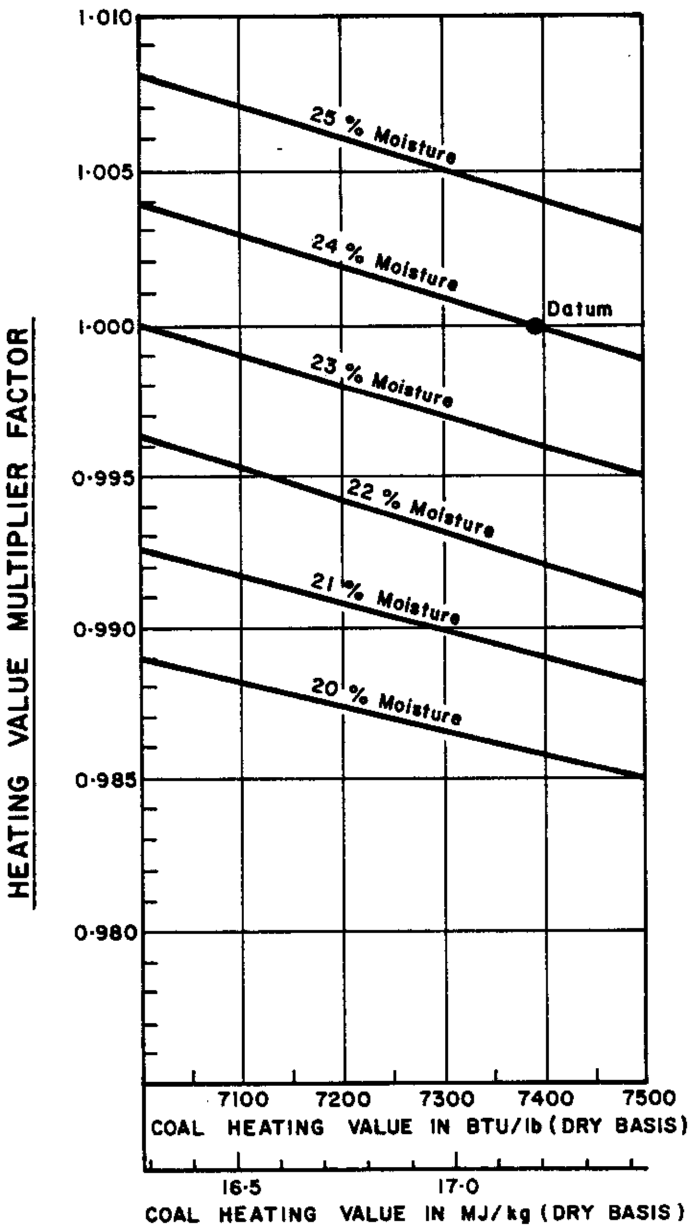
COAL – TONNES x 10^6	1·0	3·1	5·4	8·2	10·7	11·3	11·3	11·4	11·4	11·4	11·4	11·4	11·4	11·4	11·4	11·4	10·9	10·7	10·7	10·7	10·7	10·7	10·4	10·4	10·4	10·4	8·9	9·0	9·0	9·0	9·0	9·0	9·0	9·0	9·0	9·0	349·5 TOTAL
QUALITY – BTU/lb (DRY BASIS)	7484	7301	7333	7397	7319	7312	7312	7335	7335	7335	7331	7312	7312	7312	7312	7312	7284	7250	7250	7250	7250	7250	7417	7424	7424	7424	7424	7341	7328	7328	7328	7328	7328	7328	7328	7328	7327 AVERAGE
QUALITY – 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	16·9	16·9	16·9	16·9	17·2	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 AVERAGE

TABLE A4-6 - 35 YEAR PIT MATERIAL INVENTORY BY BENCH (UNDILUTED)

Bench	COAL Dry Basis					LOW GRADE COAL Dry Basis			Waste $000\ m^3$	Overburden $000\ m^3$
	000 Tonnes	MJ/kg	% Ash	% S	$000\ m^3$	000 Tonnes	MJ/kg	$000\ m^3$		
1045										3 903
1030	211	17.68	36.33	.32	136				35	6 213
1015	329	18.61	33.50	.33	211				489	7 148
1000	327	18.70	33.22	.33	210				533	7 613
985	365	17.39	36.37	.35	235				964	10 090
970	1 304	13.68	45.41	.48	.838				2 131	18 598
955	2 734	14.62	42.58	.52	1 757	551	7.94	318	5 730	24 131
940	3 839	15.37	40.25	.50	2 468	1 009	7.94	577	8 287	26 107
925	6 070	15.78	39.03	.47	3 902	1 416	7.94	808	10 386	27 757
910	9 011	15.09	41.13	.50	5 793	1 372	7.94	782	12 325	28 500
895	13 470	15.00	41.91	.52	8 600	1 514	8.21	863	9 610	30 640
880	15 889	14.92	41.54	.51	10 281	1 175	8.22	671	10 838	28 008
865	18 890	15.37	40.25	.49	12 222	1 247	8.22	709	10 324	26 756
850	27 461	16.45	37.12	.47	17 767	1 271	8.22	722	9 838	18 236
835	31 770	17.25	35.66	.45	21 130	1 027	8.17	583	11 570	10 200
820	31 820	17.51	34.68	.43	21 260	906	8.11	513	12 000	5 120
805	30 350	17.89	33.19	.41	20 420	812	8.09	460	11 660	2 310
790	27 504	18.26	31.79	.40	18 697	606	8.05	331	11 287	520
775	24 445	18.38	31.46	.40	16 618	649	8.05	348	10 252	-
760	21 864	18.52	31.10	.39	14 863	469	8.05	279	8 057	-
745	18 847	18.85	30.24	.39	12 812	426	8.05	263	6 694	-
730	16 280	19.06	29.88	.37	11 180	265	8.62	152	5 190	-
715	13 669	19.40	29.55	.36	9 391	460	8.44	235	4 092	-
700	11 277	19.56	29.08	.34	7 748	317	8.44	172	2 123	-
685	7 638	19.19	30.19	.32	5 248	81	8.44	52	559	-
670	4 034	18.49	32.11	.30	2 771	146	8.44	98	345	-
655	3 045	18.22	33.03	.31	2 080	18	8.44	24	161	-
640	1 747	17.93	33.91	.30	1 200	0	-	0	140	-
TOTAL	344 190	17.48	34.37	.42	229 838	15 737	8.12	8 960	165 620	281 850

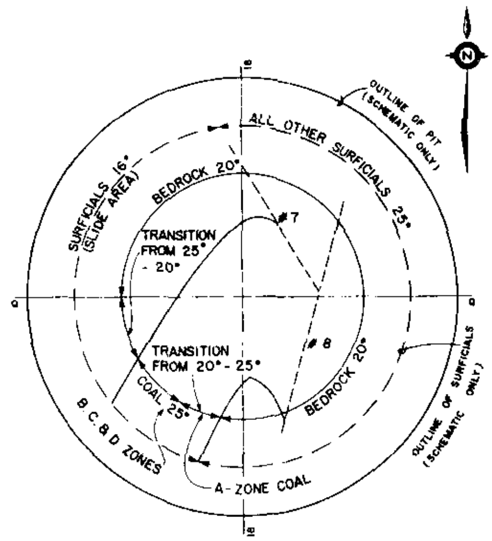
PLATE A4-1

COAL HEATING VALUE INPUT MULTIPLIER FACTOR
HAT CREEK PROJECT MINING REPORT 1978



HEATING VALUE INPUT MULTIPLIER FACTOR

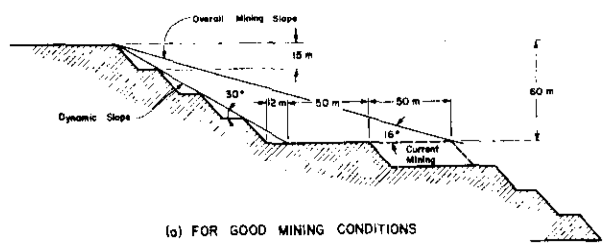
Note : Used to modify yearly H.V. Input when coal quality deviates from datum.



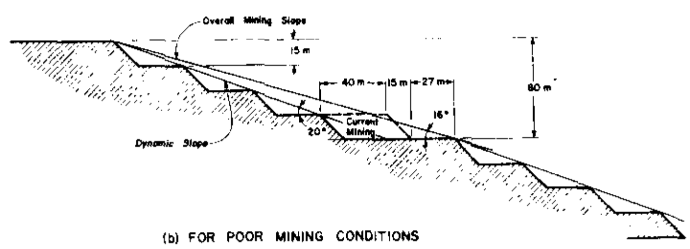
35 YEAR PIT SLOPE ANGLES

RECOMMENDED EXPANDED PIT TO 450m ASL SLOPE ANGLES

35 YEAR PIT SLOPE ANGLES



(a) FOR GOOD MINING CONDITIONS



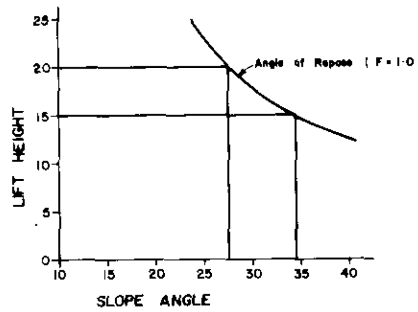
(b) FOR POOR MINING CONDITIONS

DYNAMIC SLOPE X SECTION

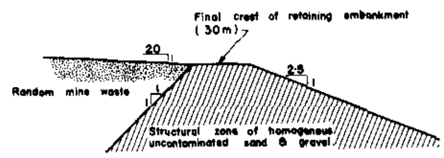
PLATE A4 - 2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

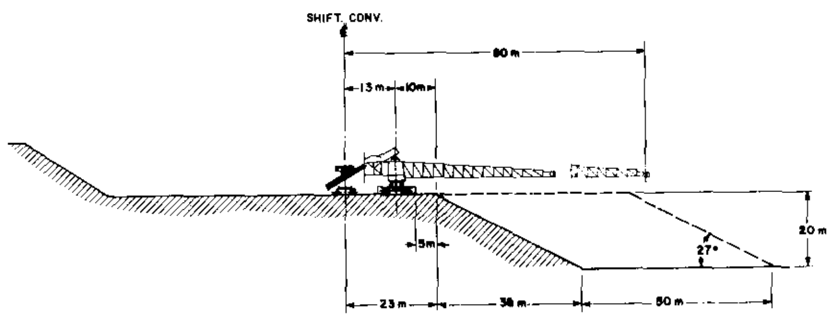
PIT SLOPES



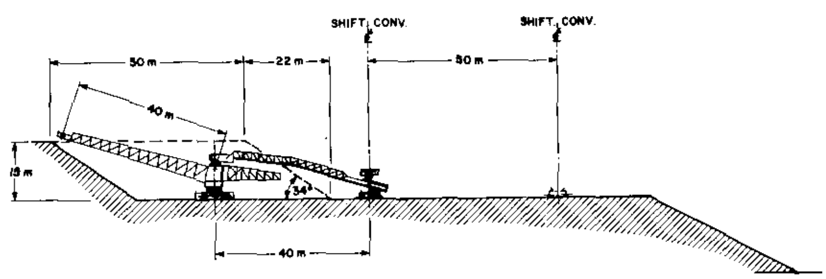
(a) WASTE MATERIAL ANGLE OF REPOSE



WASTE EMBANKMENT SLOPE ANGLES



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



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

PLATE A4-3

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

WASTE DUMP SLOPES

cominco-monenco joint venture

PLATE A4-4

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

SITE PLAN
BUCKET WHEEL PIT

Cominco cominco-monenco joint venture monenco

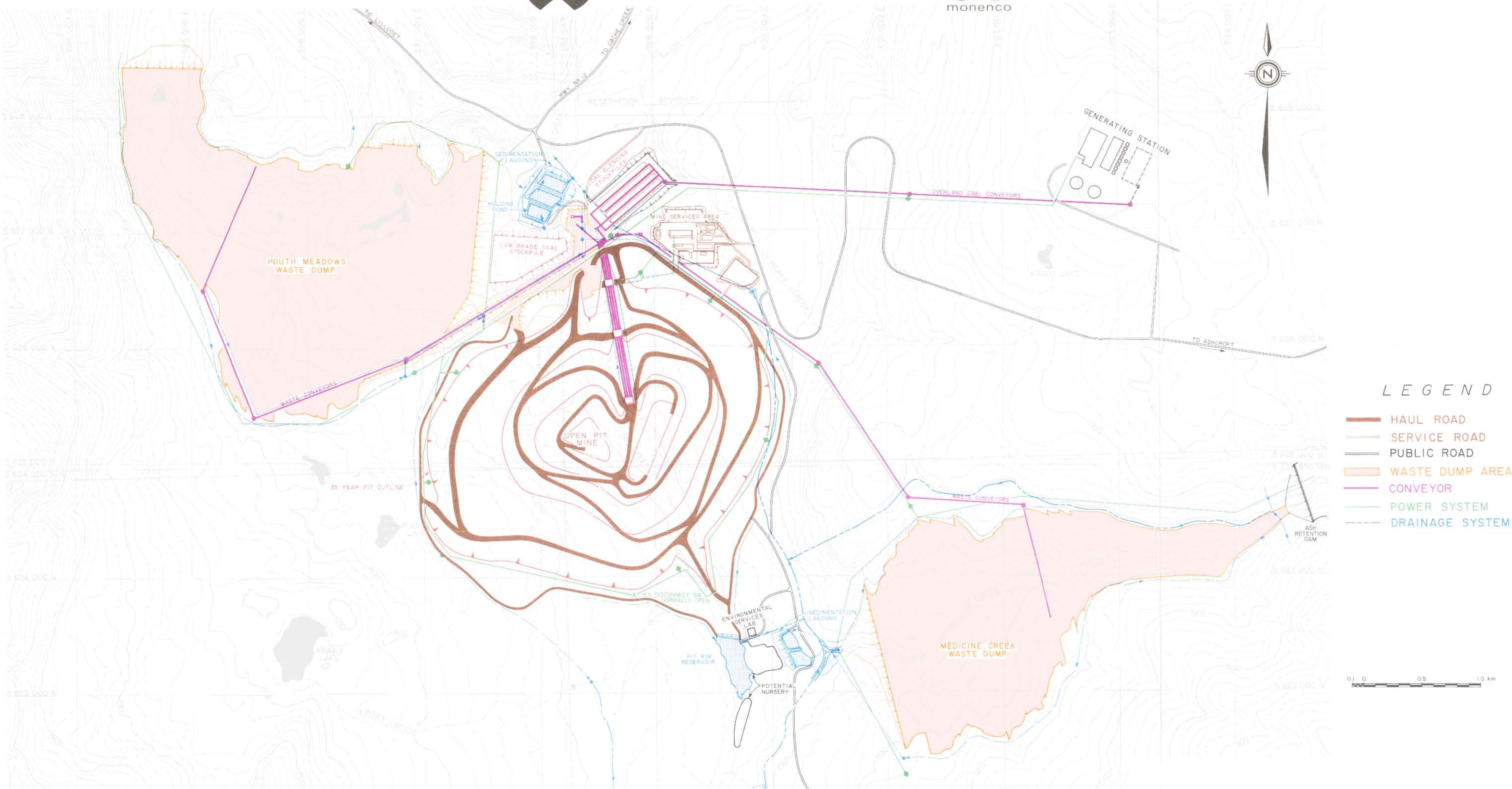


PLATE A4-5

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

SITE LAYOUT
YEAR 35
SHOVEL / TRUCK PIT

cominco-monenco joint venture

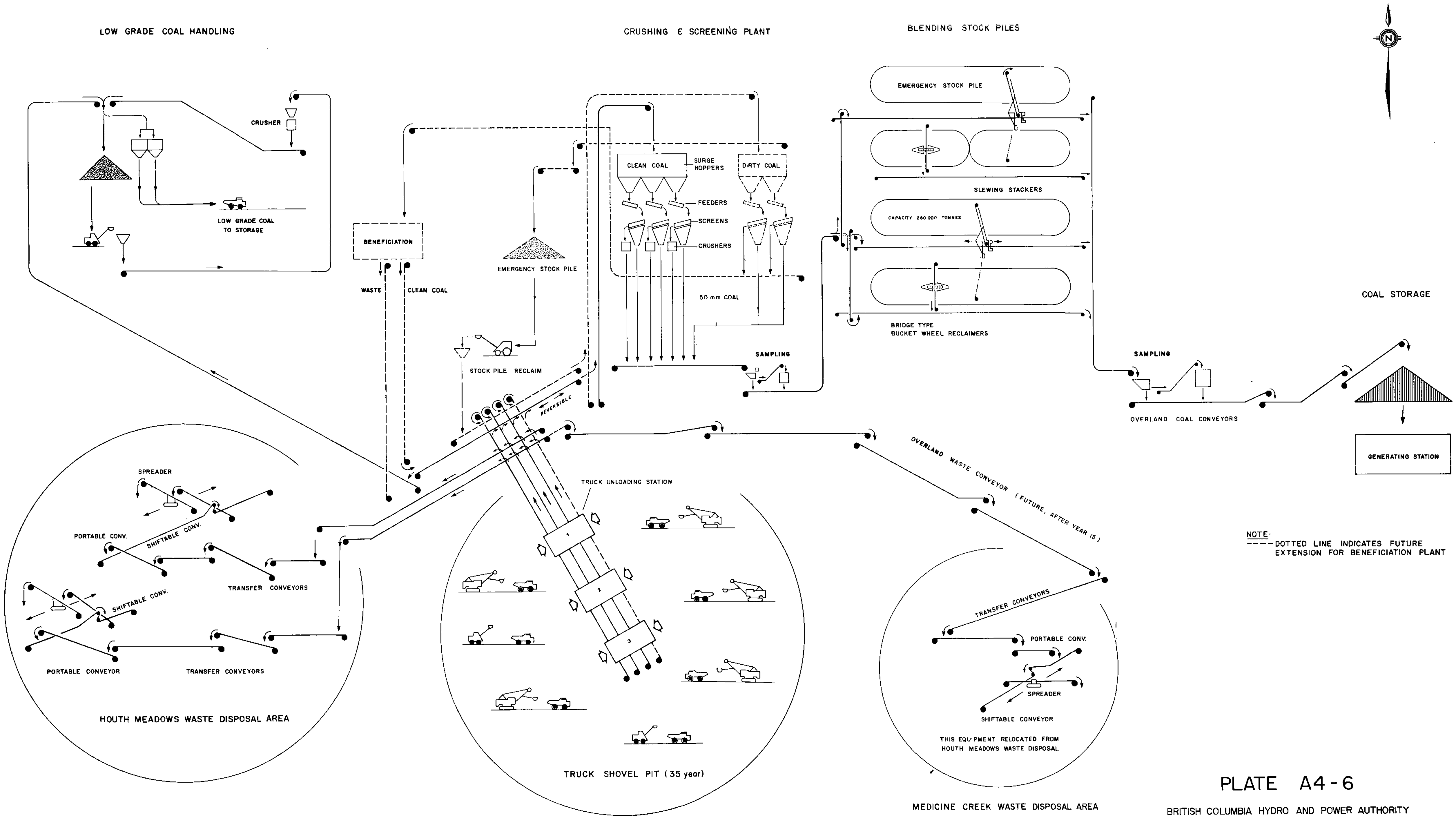
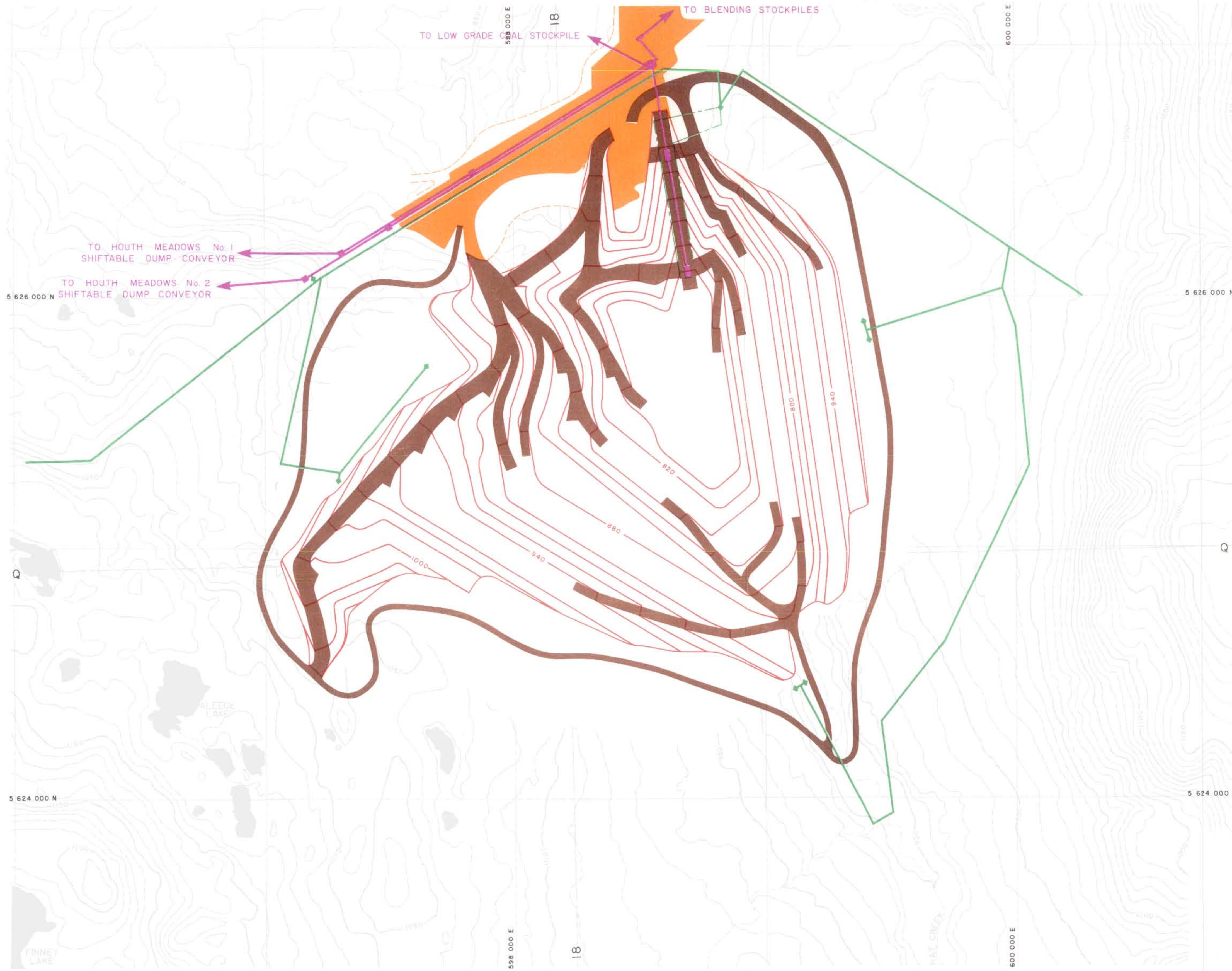


PLATE A4-6

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

SYSTEM FLOW SHEET

cominco-monenco joint venture



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

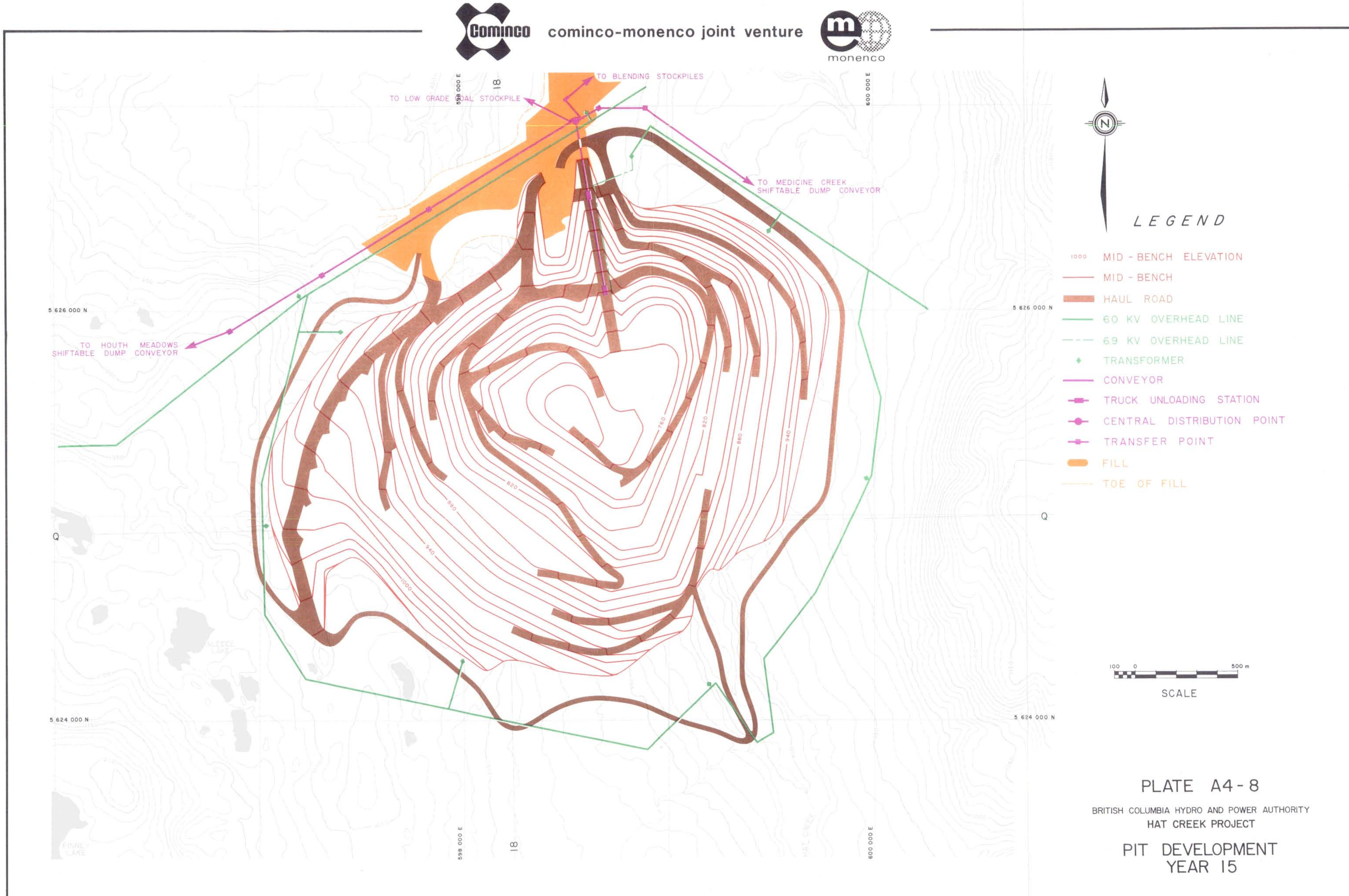
SCALE

PLATE A4-7

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

PIT DEVELOPMENT
YEAR 5


cominco-monenco joint venture
Cominco
monenco
TO BLENDING STOCKPILES
TO LOW GRADE COAL STOCKPILE
TO MEDICINE CREEK SHIFTABLE DUMP CONVEYOR
TO HOUTH MEADOWS SHIFTABLE DUMP CONVEYOR
598 000 E
600 000 E
18
5 626 000 N
5 624 000 N
Q
760
820
880
940
1000
N
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
PLATE A4-8
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT
PIT DEVELOPMENT
YEAR 15

cominco-monenco joint venture

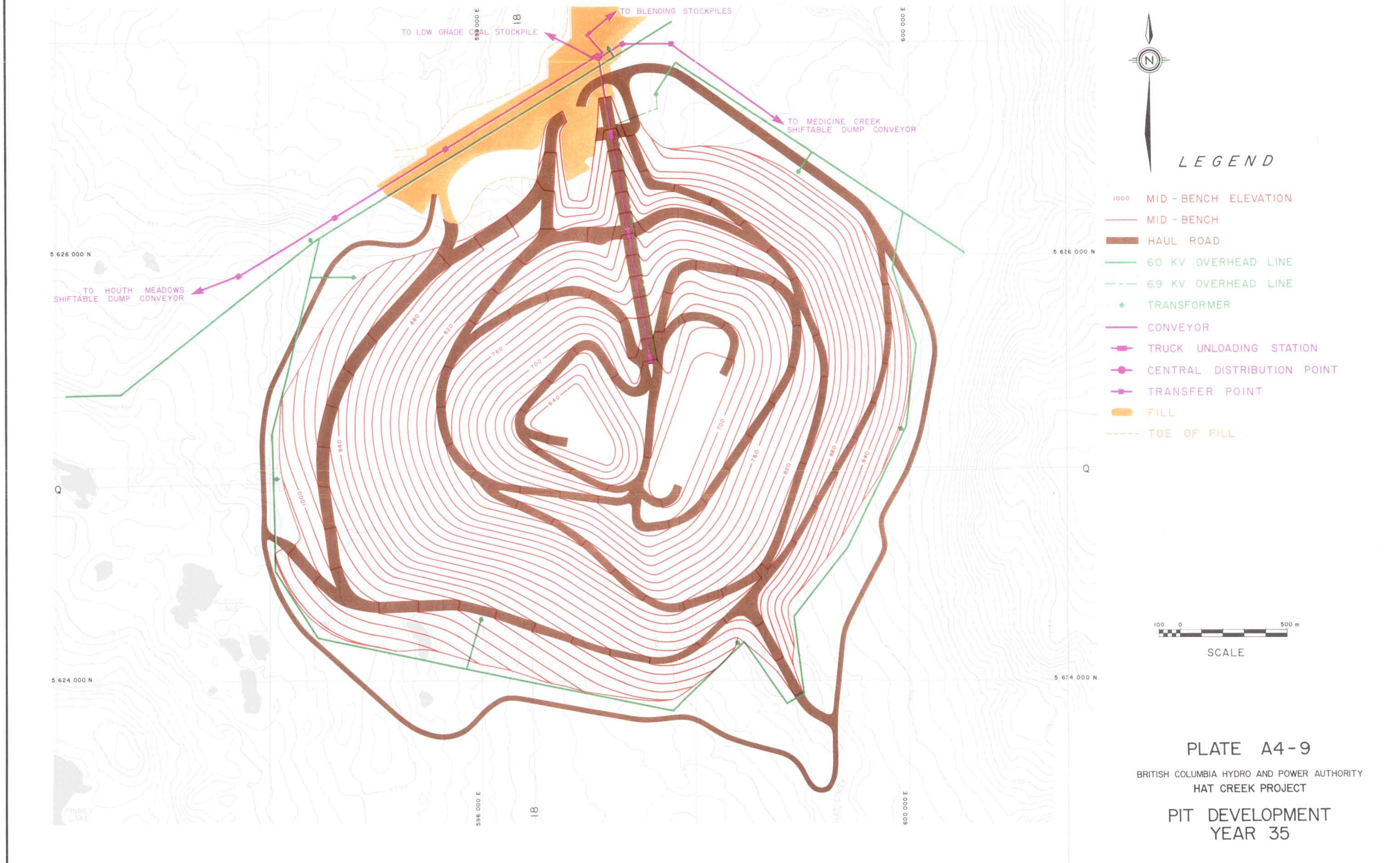


PLATE A4-9

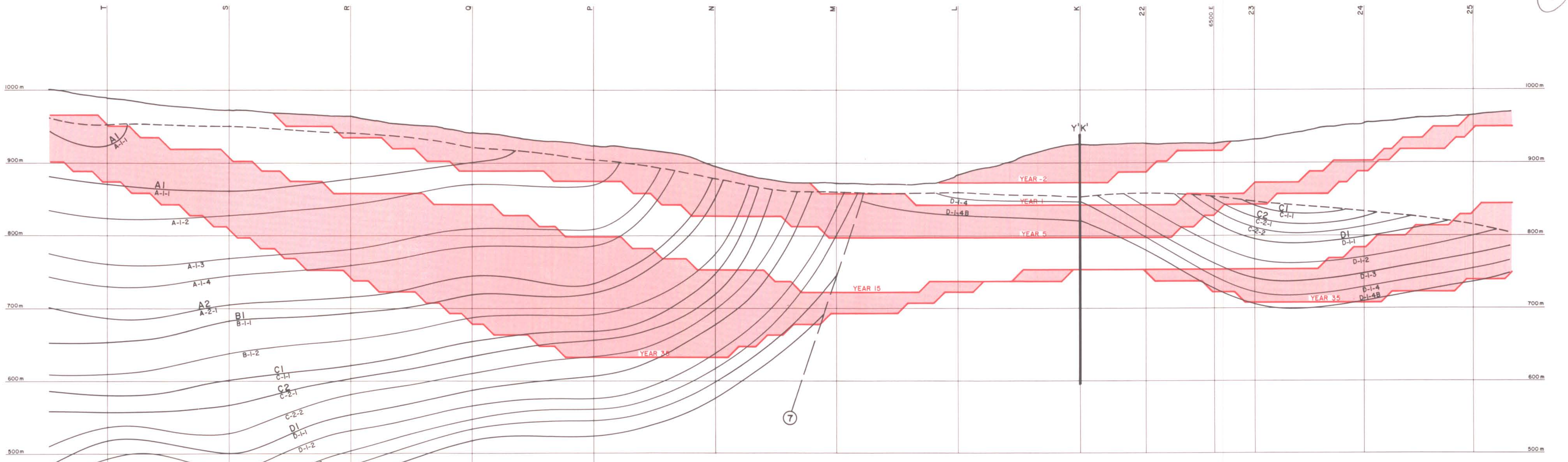
BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

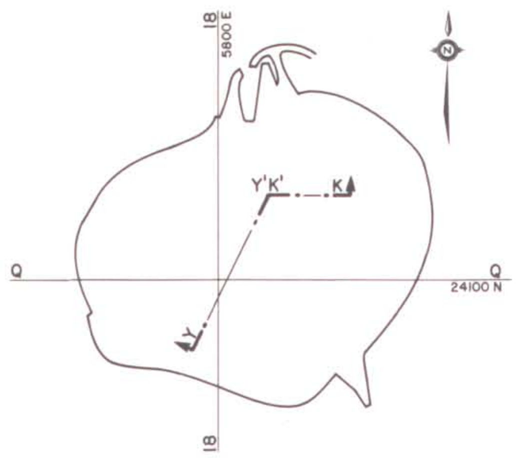
PIT DEVELOPMENT
YEAR 35

cominco-monenco joint venture

monenco

(114)





KEY PLAN
35 YEAR PIT

LEGEND

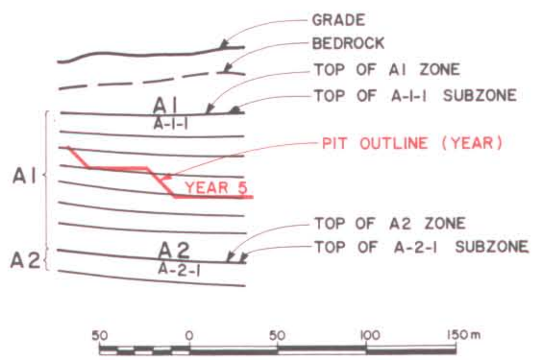


PLATE A4-10

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

MINE DEVELOPMENT
CROSS SECTION

SECTION A5.0 - EQUIPMENT SELECTION AND MANPOWER REQUIREMENTS

A5.1 MAJOR EQUIPMENT

The major equipment selected for use in the shovel/truck/ conveyor system would be proven and practical with good back-up service for parts, etc. well established in the area. Plate A4-6 illustrates the system and its major equipment components.

The following major items are discussed in this Section: loading shovels, haulage trucks and belt conveyors.

The capital costs for the major mobile mining equipment, the conveyors and crushing and blending equipment were developed based on manufacturer's listed prices and quotations in October 1977 dollars. The capital unit costs for each item of equipment include:

1. purchase cost of equipment f.o.b. factory,
2. allowance for optional extras,
3. freight and insurance to site,
4. provincial sales tax at 7 percent of f.o.b site cost, and
5. erection costs at site.

Where manufacturers' quotations were in U.S. dollars, an exchange rate of $1.08 Canadian to $1.00 U.S. was used for conversion. The capital costs, availabilities and service lives of the major mobile equipment are shown in Table A5-1. The purchase and replacement schedule was derived using the estimated service life and utilization figures and is contained in Table A5-2. The corresponding schedule of costs are shown in Table A5-3.

A5.1 MAJOR EQUIPMENT - (Cont'd)

(a) Shovels

An electric loading shovel with a bucket capacity of 16.8m^3 (22 yd^3) was chosen as the principal excavator, with a peak requirement of seven units required to meet the peak periodic generating demands. This size of shovel was selected over the 11.5 m^3 (15 yd^3) size due to its greater production capacity, greater bail pull, and its being a better match to the selected truck sizes. The shovel would be equipped with wide tracks to minimize ground bearing pressures.

Shovel Production Analysis

Bucket capacity	16.8 m^3 (22 yd^3)
Fill factor	80%
Swing cycle	36 s
Cycles per 50 min hour	83
Scheduled hours/year	8760
Physical availability	85%
Use of availability	80%
Effective utilization - 0.85 x 0.80 =	68%

	Swell Factor	Specific Gravity
Surficials (granular)	20%	2.20
Surficials (cohesive)	30%	2.20
Waste rock	30%	2.0
Coal	35%	1.55

A5.1 MAJOR EQUIPMENT - (Cont'd)

Capacities

Material	BCM/h	BCM/yr
Surficials (granular)	929	5.53
Surficials (cohesive)	858	5.10
Waste rock	858	5.10
Coal	827	4.93
Weighted average	865	5.19

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 a typical year:

Material	Volume MBCM	Shovel Productivities BCM/h	Shovel Operating Hours Required
Coal	5.63	827	6 808
Waste above bedrock			
- Granular surficials	8.30	929	8 934
- Cohesive surficials	2.58	858	3 007
Bedrock waste	0.94	858	1 096
TOTAL	17.45		19 845

The shovel fleet size to meet the specified powerplant capacity factor is:

$$= \frac{\text{Shovel operating hours required in year}}{\text{Shovel operating hours per annum}}$$

$$= \frac{19\ 845}{8760 \times .68}$$

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

A5.1 MAJOR EQUIPMENT - (Cont'd)

To allow for increased production requirements during an increase in capacity factor of the powerplant it was determined that 28 180 shovel operating hours or 4.73 (5) shovels would be required.

(b) Haulage Trucks

Three sizes of mine haulage truck were selected for use in the mine:

- 136 t (150 ton) waste truck
- 109 t (120 ton) coal truck
- 32 t (35 ton) haulage truck

The first two were selected as production trucks with the latter selected as a general purpose truck for road construction, dump development, etc.

The 136 t (150 ton) capacity electric wheel drive truck would be fitted with a 1120 kW (1600 hp) diesel engine, and a 89 m^3 (115 yd^3) heaped capacity rock box. An actual load of 128 t (140 tons) was used in the study.

The 109 t (120 ton) capacity electric wheel drive truck would be fitted with a 895 kW (1200 hp) diesel engine and a 114 m^3 (148 yd^3) heaped capacity coal box. An actual load of 104 t (115 tons) was used in the study.

The number of trucks required would fluctuate during the life of the mine with the peak being 9 x 109 t coal and 18 x 136 t waste trucks, as shown on the equipment schedule, Table A5-2. These numbers include extra trucks to allow for the increase in powerplant capacity factor.

A5.1 MAJOR EQUIPMENT - (Cont'd)

Both production trucks were considered to be a good match to the selected loading shovel, are commonly used and due to the difference in unit weight of the coal and waste materials cost savings would be possible in using different size trucks for the different materials.

(i) Hauling Analysis

A. 109 t Coal Truck

In developing the truck fleet size a computer simulation program was used to perform the haul cycle calculations for the given average haul profile for a given year or increment. An example is given for a typical year - coal haulage upward to the El. 895 unloading station.

Maximum allowable speed on flat haul	=	50 km/h (30 mph)
Maximum allowable speed on down grade	=	30 km/h (20 mph)
Rolling resistance	=	3%
Maximum grade	=	±8%
Truck effective utilization	=	57%
Truck scheduled hours	=	8760

Average Haul Profile (loaded)	Computer-simulated Haul Times in Minutes: Loaded	Empty
1. 500 m (1640 ft) flat	1.07	0.87
2. 380 m (1250 ft) ±8%	1.79	0.71
3. 220 m (720 ft) flat	0.51	0.35
4. 380 m (1250 ft) ±8%	1.80	0.71
5. 250 m (820 ft) flat	0.65	0.45
Total 1730 m (5680 ft)	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 t x 4.24 trips/hr
= 2.205 Mt per annum.

Coal required to be hauled on average profile = 6.928 Mt

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

Similar haul cycle calculations were made for the downhill and level hauls to the El. 895 unloading station. It was determined the same four trucks would handle the coal production on these hauls.

In order to allow for an increase in powerplant capacity factor the 109 t (120 ton) truck fleet size for the year would be six.

B. 136 t Waste Truck

Similar studies were carried out hauling waste to the dump pockets with the 136 t (150 ton) waste truck using the same maximum speeds, rolling resistance, grades, utilization, etc.

(c) Conveying Systems

The conveying systems, divided basically into in-pit, coal handling, and waste handling were designed to estimated peak tonnages for the various materials.

A5.1 MAJOR EQUIPMENT - (Cont'd)

The coal handling system, including the in-pit inclined coal conveyor, is discussed in detail in Appendix D.

Capital costs and cash flows for these conveyor systems are shown in Table A5-4.

(i) In-pit Conveying

All in-pit materials would be hauled by truck to loading pockets located over an inclined mine conveyor system at the northern end of the pit. Here, facilities would exist to receive, crush and feed onto any one of three 1200 mm (48 in) wide parallel existing conveyors, each with a carrying capacity of 5000 t/h (5500 tons/hr) of waste or 3200 t/hr (3500 tons/hr) of coal.

As the mine deepened the conveyors would be extended and loading pockets added to minimize truck haulage on adverse grades. A total of three pockets would be installed by year 20.

At the surface the three conveyors would discharge into a distribution point which could redirect the materials onto any one of three overland conveyors, two of which would be dedicated to waste and one, which would be reversible, for a combination of coal and low-grade coal.

(ii) Coal Handling

Coal would be conveyed from the distribution point to the crushing and sampling plant on a 1200 mm (48 in) wide belt, carrying capacity 3200 t/hr (3500 ton/hr). Crushing to minus 50 mm (2 in) would be done prior to blending.

A5.1 MAJOR EQUIPMENT - (Cont'd)

Blending would be accomplished by the windrow method of layering, as shown in Plate A5-1, through either of two rail-mounted slewing stackers into one of two piles of about 280 000 t (310,000 tons) capacity each (7 days supply for the 4-unit powerplant operating at full load). In normal operation one pile would be built while another would be reclaimed for transportation to the powerplant. In addition, provision would be made to build and reclaim two 135 000 t (150,000 ton) piles of high-grade/low-sulphur coal to enhance the blended quality if required and to deliver low-sulphur coal directly to the powerplant on occasion.

Recovery would be by either of two bridge-type bucket wheel reclaimers onto the main overland conveyor to the powerplant. This conveyor, in two flights totalling 4000 m (2 1/2 mi) in length, would also be 1200 mm (48 in) wide and have a carrying capacity matching the coal reclaiming system of 2500 t/hr (2750 tons/hr). Total lift on these conveyors would be 520 m (1700 ft), power requirement 6000 kW (8000 hp). Further work is planned to optimize the coal handling system.

(iii) Waste Handling

Two 1200 mm (48 in) wide waste conveyors are destined for the Houth Meadows area. An extensive dike 45 m, (150 ft), high x 900 m, (3000 ft), long would form part of the conveyor routing across the Hat Creek Valley.

A5.1 MAJOR EQUIPMENT - (Cont'd)

One conveyor, its shiftable extensions, tripper and 40 m (130 ft) boom spreader would handle construction grade materials for deposition into the waste retaining embankment. The second identical waste conveyor system would discharge weaker waste rock in 35 m (115 ft) lifts, and in a predetermined sequence, behind the engineered embankment. Plate A4-3 illustrates the method of operation.

Design capacity of each system is 5000 t/h (5500 tons/hr).

After approximately 15 years of operation one of the waste conveyor systems would be relocated into the Medicine Creek area for similar service.

(iv) Low-grade Coal

It is expected that all of the low-grade material during the project life could be accommodated in the Houth Meadows site, but the alternative routing to near Medicine Creek would be available in later years.

The low-grade coal would be conveyed to a preparation area, at a rate of 5000 t/h (5500 tons/hr) on 1200 mm (48 in) belts, where it would be stacked prior to crushing. The material would be reclaimed at 300 t/h (330 tons/hr) and crushed to minus 50 mm (2 in) then trucked in 32 t (35 ton) trucks to the stockpile area for spreading and then compacting to prevent spontaneous combustion.

A5.1 MAJOR EQUIPMENT - (Cont'd)

The following is a summary of peak conveyor installation plus a list of equipment required for the crushing and blending functions.

		Length in	
		Metres	(ft)
Conveyors:			
Central out-of-pit conveyors		4 290	(14,070)
Overland coal conveyor (to powerplant)		4 000	(13,120)
Waste conveyors:	Houth Meadows	13 005	(42,660)
	Medicine Creek	2 850	(9,350)
Coal crushing and blending area		3 290	(10,790)
TOTAL		27 435 m	(89,990 ft)

Crushing, Stockpiling and Blending:		
Bridge-type bucket wheel reclaimers		2
Self-propelled rail-mounted stackers		2
Crushers:	in-pit primary	9
	out-of-pit secondary	3

A5.2 SUPPORT EQUIPMENT

A fleet of support equipment would be selected to complement the major mining equipment as well as to carry out certain other functions such as: topsoil removal, road construction and maintenance, pit clean-up, bench pioneering, dump construction, conveyor moving and equipment servicing at various areas in the mine and dumps.

The capital costs and purchase replacement schedule for this equipment are shown in Tables A5-2 and A5-3. Peak requirements of the equipment are shown in Table A5-4. The following is typical of the items included:

A5.2 SUPPORT EQUIPMENT - (Cont'd)

scrapers
tracked dozers
rubber tired dozers
road grader
11.5 m^3 (15 yd^3) front end loaders
5.4 m^3 (7 yd^3) front end loaders
drills - percussion
- auger
- rotary
water wagon
compactor
crusher
service vehicles including:

- fuel and lube trucks
- cranes
- tire truck
- line truck
- blasting truck
- service trucks
- repair trucks
- ambulance
- fire truck
- pickup trucks

A5.3 MANPOWER

The manpower schedules, developed to cover both salaried staff and hourly paid personnel for the pre-production, production and where necessary the post-production years, are shown in the following six tables.

A5.3 MANPOWER - (Cont'd)

Table A5-6	Pre-production Construction and Project Management
Table A5-7	Summary of Peak Operating Manpower Requirement
Table A5-8	Administration and Services, Reclamation and Environmental Protection
Table A5-9	Mine Supervision and Engineering
Table A5-10	Mine Operating Labour
Table A5-11	Maintenance Supervision and Maintenance Labour

1. Salaried staff labour was estimated using the developed hourly labour lists as well as being based on experience of large mining operations.

2. The estimate of operating labour for the mining equipment was based on the number of annual scheduled hours for the equipment and the mine operating schedule from which are obtained the annual operating hours per man.

3. The maintenance labour was estimated using anticipated mechanical availabilities for the mine equipment. This labour force would be comprised of the various trades necessary to carry out the maintenance requirements in an operation of this type.

4. Contingencies of 5 percent for salaried staff labour and 15 percent for hourly paid operating and maintenance labour were applied to the totals as an allowance for sickness, leave of absence, absenteeism etc.

As can be seen from the summary shown in Table A5-7, the total manpower, including contingencies, to operate the Hat Creek mine peaks at 1005 persons between the years 16 and 25.

TABLE A5-1

Capital Costs and Service Lives of Major Items of Equipment

Hat Creek Project Mining Report 1978

Item	Capital Cost F.O.B. Hat Creek $ 1977	Service Life Operating Hours	Effective Utilization %	Mechanical Availability %
Drills				
Auger-truck mounted	185,000	15,000	68	75
Air-Trac c/w compressor	122,000	15,000	50	75
Shovels (rope)				
16.8 m^3	2,960,000	120,000	68	85
Front-end loader				
5.4 m^3	285,000	15,000	60	65
7.6 m^3	445,000	15,000	60	65
11.5 m^3	550,000	25,000	55	65
Haulage truck				
32 tonnes	226,500	25,000	60	70
109 tonnes (coal box)	622,000	50,000	57	70
136 tonnes (rock box)	713,000	50,000	57	70
Scraper				
24 m^3 (tandem)	358,000	15,000	57	70
Dozer (track)				
CAT D6	113,000	15,000	57	70
CAT D8	205,000	15,000	57	70
CAT D8 w/ripper	223,000	15,000	57	70
CAT D9 w/winch	300,000	15,000	57	70
CAT D9 w/ripper	312,000	15,000	57	70

Continued

TABLE A5-1 (Continued)

Item	Capital Cost F.O.B. Hat Creek $ 1977	Service Life Operating Hours	Effective Utilization %	Mechanical Availability %
Dozer (wheel)				
CAT 824B	196,000	25,000	68	70
Compactor				
CAT 825B	213,000	20,000	60	70
Grader				
CAT 16G	204,000	25,000	68	70
Crane				
15 tonnes	126,000	20,000	75	80
45 tonnes	237,000	20,000	35	85
70 tonnes	355,000	35,000	35	85
90 tonnes	477,000	35,000	35	85
Trucks (miscellaneous)				
5 tonne service	18,000	20,000	75	80
3 tonne flatdeck (c/w 2 tonne crane)	25,000	20,000	75	80
Tire truck	35,000	20,000	75	80
Line truck	65,000	32,000	75	80
Lube truck	55,000	20,000	75	80
Fuel truck (13.6 kL)	85,000	20,000	75	80
Water wagon (45.5 kL)	270,000	25,000	68	75
Dump truck (10 tonnes)	30,000	25,000	75	80
Sanding truck (10 tonnes)	32,000	25,000	75	80
Fire truck	60,000	50,000	10	95
Ambulance	17,000	20,000	10	95
Personnel bus (24 pass)	18,000	20,000	75	80
Pick-up (1 tonne)	10,500	15,000	90	80
Pick-up (3/4 tonne)	9,500	15,000	90	80

Continued

TABLE A5-1 (Continued)

Item	Capital Cost F.O.B. Hat Creek $ 1977	Service Life Operating Hours	Effective Utilization %	Mechanical Availability %
Pumps				
10 cm diesel	4,000	13,000	75	85
15 cm diesel	6,500	13,000	75	85
Welders (portable)				
600 A (diesel)	5,500.00	13,000	75	85
600 A (electric)	2,700.00	20,000	75	85
Miscellaneous				
Backhoe (1 m^3)	150,000	30,000	68	75
Compressor (17 m^3/min)	59,200	25,000	70	80
Compressor (30 m^3/min)	90,000	25,000	70	80
Steam cleaner (mobile)	60,000	20,000	50	60
Lighting plan (3 kw)	10,000	10,000	50	60
Gradall	125,000	32,000	75	80
50 kW generator	20,000	32,000	75	85
Lo-boy tractor	80,000	32,000	75	80
Hi-boy trailer	40,000	32,000	75	80
Crushing plant	300,000	25,000	70	80
CaCl spreader (box only)	7,000	20,000	75	85
Lube island	80,000	60,000	75	90
Blasting truck	62,000	20,000	75	85

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

(M5)

TABLE A5 - 2
SHEET 1 OF 3

MOBILE MINING AND AUXILIARY EQUIPMENT PURCHASE AND REPLACEMENT SCHEDULE

TABLE A5 - 2 (SHEET 1)

ACCOUNT CODE	DESCRIPTION	LIFE IN YEARS	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																
			-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
91000	**MINE PROPERTY DEVELOPMENT**																								
91300	ROADS AND BRIDGES																								
91310	HAUL ROADS																								
	16 G Grader	4					2 0 / 2				0 2 / 2	1 0 / 3			0 3 / 2	0 1 / 3			0 2 / 3	0 1 / 3			0 2 / 3	1 1 / 4	
	Water Wagon	6					2 0 / 2		1 0 / 3				0 2 / 3		0 1 / 3				0 2 / 3		0 1 / 3				0 2 / 3
	D-9 Dozer c/w Ripper	3					1 0 / 1		1 0 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 1	0 1 / 2		0 1 / 2	0 1 / 2
	824 Wheeled Dozer	3					1 0 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1
	Mobile Crusher	10					1 0 / 1										0 1 / 1								
	Gradall	6					1 0 / 1						0 1 / 1						0 1 / 1						0 1 / 1
	24 LCM Scraper	4					4 0 / 4				0 4 / 4				0 4 / 4				0 4 / 4				0 4 / 4		
	32 tonne Dump Truck	7					3 0 / 3						1 0 / 4	0 3 / 4						0 1 / 4	0 3 / 4				
	5.4 m^3 Front-End Loader	4					1 0 / 1		1 0 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2
93000	**PIT SERVICES**																								
93200	IN-PIT DRAINAGE																								
	Diesel Pumps	10					10 0 / 10										0 10 / 10								
	1 m^3 Backhoe	5		1 0 / 1					0 1 / 1					0 1 / 1					0 1 / 1					0 1 / 1	
	3-5 tonne Hiab Truck	5		1 0 / 1					0 1 / 1					0 1 / 1					0 1 / 1					0 1 / 1	
94000	**MINING EQUIPMENT**																								
94100	ELECTRIC SHOVELS																								
	16.8 m^3 Shovel	20				3 0 / 3		1 0 / 4	1 0 / 5		1 0 / 6													1 0 / 7	
94200	SPECIALIZED EXCAVATORS																								
	24 LCM Scraper	4					2 0 / 2				0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2		
94300	SPECIALIZED EXCAVATORS - PARTINGS		NOT USED																						
94400	HAULAGE TRUCKS																								
	109 tonne Coal Truck	10						4 0 / 4		1 0 / 5	1 0 / 6	1 0 / 7	1 0 / 8					0 4 / 8		1 0 / 9		0 1 / 9	0 1 / 9		
	136 tonne Waste Truck	10					10 0 / 10		3 0 / 13	2 0 / 15							0 10 / 15							3 0 / 18	

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD																		
		18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	TOTAL
91000	**MINE PROPERTY DEVELOPMENT**																			
91300	ROADS AND BRIDGES																			
91310	HAUL ROADS																			
	16 G Grader		0 2 / 4	0 2 / 4			0 2 / 4	0 2 / 4			0 2 / 4	0 2 / 4			0 2 / 4	0 2 / 4				4 27 / 31
	Water Wagon		0 1 / 3				0 2 / 3		0 1 / 3				0 2 / 3		0 1 / 3					3 15 / 18
	D-9 Dozer c/w Ripper		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2				2 21 / 23
	824 Wheeled Dozer			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1				1 11 / 12
	Mobile Crusher		0 1 / 1										0 1 / 1							1 3 / 4
	Gradall						0 1 / 1						0 1 / 1							1 5 / 6
	24 LCM Scraper		0 4 / 4				0 4 / 4				0 4 / 4				0 4 / 4					4 32 / 36
	32 tonne Dump Truck		0 1 / 4	0 3 / 4						0 1 / 4	0 3 / 4									4 15 / 19
	5.4 m^3 Front-End Loader		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2		0 1 / 2					2 15 / 17
93000	**PIT SERVICES**																			
93200	IN-PIT DRAINAGE																			
	Diesel Pumps		0 10 / 10										0 10 / 10							10 30 / 40
	1 m^3 Backhoe				0 1 / 1					0 1 / 1					0 1 / 1					1 7 / 8
	3-5 tonne Hiab Truck				0 1 / 1					0 1 / 1					0 1 / 1					1 7 / 8
94000	**MINING EQUIPMENT**																			
94100	ELECTRIC SHOVELS																			
	16.8 m^3 Shovel	0 3 / 7		0 1 / 7																7 4 / 11
94200	SPECIALIZED EXCAVATORS																			
	24 LCM Scraper		0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2					2 16 / 18
94300	SPECIALIZED EXCAVATORS - PARTINGS																			
94400	HAULAGE TRUCKS																			
	109 tonne Coal Truck			0 3 / 8				0 1 / 7	0 1 / 7					0 2 / 6						9 13 / 22
	136 tonne Waste Truck		0 10 / 18										0 4 / 9							18 24 / 42

MOBILE MINING AND AUXILIARY EQUIPMENT PURCHASE AND REPLACEMENT SCHEDULE

TABLE A5-2 (SHEET 2)

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BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

TABLE A5-2
SHEET 2 OF 3

ACCOUNT CODE	DESCRIPTION	LIFE IN YEARS	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																																				TOTAL
			-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	
94000	MINING EQUIPMENT (continued)																																											
94500	IN-PIT CONVEYORS		INCLUDED IN ACCOUNTS 95000 AND 97000																																									
94600	IN-PIT MATERIAL SIZING		INCLUDED IN ACCOUNTS 95000 AND 97000																																									
94700	AUXILIARY MINING EQUIPMENT																																											
94710	DOZERS																																											
	D-9 c/w Winch	3					1 0 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1				1 11 / 12
	D-9 c/w Ripper	3					3 0 / 3		2 0 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5		0 2 / 5	0 3 / 5				5 53 / 58
	824 Wheeled Dozer	3					1 0 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1			0 1 / 1				1 11 / 12
	D-8	3					2 0 / 2		2 0 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4		0 2 / 4	0 2 / 4				4 42 / 46
94715	CRANES																																											
	90 tonne	10					1 0 / 1										0 1 / 1										0 1 / 1										0 1 / 1							1 3 / 4
	45 tonne	10				2 0 / 2										0 2 / 2										0 2 / 2										0 2 / 2								2 6 / 8
	15 tonne	3				1 0 / 1		1 0 / 2	1 1 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3	0 2 / 3		0 1 / 3			3 32 / 35
94720	16 G GRADERS	4					2 0 / 2				0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2				0 2 / 2					2 16 / 18
94725	LIGHT VEHICLES																																											
	Fire Truck	10				1 0 / 1			1 0 / 2							0 1 / 2			0 1 / 2							0 1 / 2			0 1 / 2							0 1 / 2			0 2 / 2					2 6 / 8
	Ambulance	3				1 0 / 1		1 0 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2		0 1 / 2	0 1 / 2					2 21 / 23
	Personnel Bus (24 passengers)	3					5 0 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5			0 5 / 5				5 55 / 60
	Personnel Bus (8 passengers)	3					2 0 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2			0 2 / 2				2 22 / 24
94730	FRONT-END LOADER - 11.5 m^3	5					1 0 / 1	1 0 / 2				0 1 / 2	0 1 / 2				0 1 / 2	0 1 / 2				0 1 / 2	0 1 / 2				0 1 / 2	0 1 / 2				0 1 / 2	0 1 / 2				0 1 / 2	0 1 / 2						2 12 / 14
94735	RECOVERY VEHICLES																																											
	Hi-Boy	10					1 0 / 1										0 1 / 1										0 1 / 1										0 1 / 1							1 3 / 4
	Lo-Boy c/w Tractor	10					1 0 / 1										0 1 / 1										0 1 / 1										0 1 / 1							1 3 / 4
94740	DRILLING AND BLASTING EQUIPMENT																																											
	Air Track c/w Compressor	5					1 0 / 1					0 1 / 1																																1 1 / 2
	Truck-Mounted Drill	9				1 0 / 1								0 1 / 1									0 1 / 1									0 1 / 1												1 3 / 4
	Blasting Truck	10					1 0 / 1										0 1 / 1										0 1 / 1										0 1 / 1							1 3 / 4
	Auger Truck-Mounted Drill	5					1 0 / 1					0 1 / 1					0 1 / 1					0 1 / 1					0 1 / 1					0 1 / 1					0 1 / 1							1 6 / 7

M6

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M7

MOBILE MINING AND AUXILIARY EQUIPMENT PURCHASE AND REPLACEMENT SCHEDULE

TABLE A5-2 (SHEET 3)

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

TABLE A5-2
SHEET 3 OF 3

ACCOUNT CODE	DESCRIPTION	LIFE IN YEARS	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																
			-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
94000	MINING EQUIPMENT (continued)																								
94700	AUXILIARY MINING EQUIPMENT (Continued)																								
94745	MOBILE SERVICING EQUIPMENT																								
	5 tonne Service Truck	3				2 0/2		2 0/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4	
	Tire Truck	3					1 0/1			0 1/1			0 1/1			0 1/1			0 1/1			0 1/1			0 1/1
	Line Truck	5				1 0/1	2 0/3				0 1/3	0 2/3				0 1/3	0 2/3				0 1/3	0 2/3			
	Fuel Truck	3					2 0/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2
	3-5 tonne Hiab Truck	3				2 0/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2	
	Lube Truck	3				2 0/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2	
94750	WAREHOUSE MOBILE EQUIPMENT																								
	Forklift	5				3 0/3		1 0/4			0 3/4		0 1/4			0 3/4		0 1/4			0 3/4		0 1/4		
94755	MISCELLANEOUS EQUIPMENT																								
	Calcium Chloride Spreader	5				2 0/2		2 0/4			0 2/4		0 2/4			0 2/4		0 2/4			0 2/4		0 2/4		
	Steam Cleaner	5				2 0/2		2 0/4			0 2/4		0 2/4			0 2/4		0 2/4			0 2/4		0 2/4		
	Field Fuel and Lube Station	10								1 0/1										0 1/1					
	Skid-Mounted Light Towers	10					6 0/6										0 6/6								
	Pickup Fleet	1/3					33 0/33	11 21/32	45 24/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89
	Truck fleet - over 1 tonne	5					15 0/15	8 0/23	7 0/30			0 15/30	0 8/30	0 7/30			0 15/30	0 8/30	0 7/30			0 15/30	0 8/30	0 7/30	
97000	WASTE DISPOSAL EQUIPMENT																								
97400	MOBILE HAULING EQUIPMENT																								
	32 tonne Dump Trucks	7					4 0/4						2 0/6	0 4/6						0 2/6	0 4/6				
	5.4 m^3 Front-End Loader	4					1 0/1				0 1/1				0 1/1				0 1/1				0 1/1		
97470	DOZERS - D-8	3					2 0/2			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2	2 0/4		0 2/4
97480	COMPACTION EQUIPMENT																								
	Wheeled Type	10						1 0/1										0 1/1							
	Vibrating Type	5					2 0/2					0 2/2					0 2/2					0 2/2			

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD																		
		18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	TOTAL
94000	MINING EQUIPMENT (continued)																			
94700	AUXILIARY MINING EQUIPMENT (Continued)																			
94745	MOBILE SERVICING EQUIPMENT																			
	5 tonne Service Truck	0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4			4 44/48
	Tire Truck			0 1/1			0 1/1			0 1/1			0 1/1			0 1/1				1 11/12
	Line Truck	0 1/3	0 2/3				0 1/3	0 2/3				0 1/3	0 2/3							3 18/21
	Fuel Truck			0 2/2			0 2/2			0 2/2			0 2/2			0 2/2				2 22/24
	3-5 tonne Hiab Truck		0 2/2			0 2/2			0 2/2			0 2/2			0 2/2					2 22/24
	Lube Truck		0 2/2			0 2/2			0 2/2			0 2/2			0 2/2					2 22/24
94750	WAREHOUSE MOBILE EQUIPMENT																			
	Forklift	0 3/4		0 1/4			0 3/4		0 1/4			0 2/3								4 22/26
94755	MISCELLANEOUS EQUIPMENT																			
	Calcium Chloride Spreader	0 2/4		0 2/4			0 2/4		0 2/4			0 2/4		0 2/4						4 24/28
	Steam Cleaner	0 2/4		0 2/4			0 2/4		0 2/4			0 2/4		0 2/4						4 24/28
	Field Fuel and Lube Station					0 1/1										0 1/1				1 3/4
	Skid-Mounted Light Towers		0 6/6										0 6/6							6 18/24
	Pickup Fleet	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	0 39/89	0 69/89	0 43/89	89 1749/1838
	Truck fleet - over 1 tonne		0 15/30	0 8/30	0 7/30			0 15/30	0 8/30	0 7/30			0 15/30	0 8/30	0 7/30			0 15/30	0 8/30	30 203/233
97000	WASTE DISPOSAL EQUIPMENT																			
97400	MOBILE HAULING EQUIPMENT																			
	32 tonne Dump Trucks		0 2/6	0 4/6						0 2/6	0 4/6									6 22/28
	5.4 m^3 Front-End Loader		0 1/1				0 1/1				0 1/1				0 4/1					1 8/9
97470	DOZERS - D-8	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4	0 2/4		0 2/4				4 32/36
97480	COMPACTION EQUIPMENT																			
	Wheeled Type			0 1/1										0 1/1						1 3/4
	Vibrating Type		0 2/2					0 2/2					0 2/2							2 12/14

MOBILE MINING AND AUXILIARY EQUIPMENT
CAPITAL COST SCHEDULE
TABLE A5-3 (SHEET 1)

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BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

TABLE A5-3
SHEET 1 OF 3

ACCOUNT CODE	DESCRIPTION	UNIT COST	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
91000	MINE PROPERTY DEVELOPMENT																																											
91300	ROADS AND BRIDGES																																											
91310	HAUL ROADS																																											
	16 G Grader	204					408				408	204			408	204			408	204			408	408			408	408			408	408			408	408			408	408				6324
	Water Wagon	270					540		270				540		270				540		270				540		270				540		270				540		270					4860
	D-9 Dozer c/w Ripper	312					312		312	312		312	312		312	312		312	312		312	312		312	312		312	312		312	312		312	312		312	312		312	312				7176
	824 Wheeled Dozer	196					196			196			196			196			196			196			196			196			196			196			196			196				2352
	Mobile Crusher	300					300										300										300										300							1200
	Gradall	125					125						125						125						125						125						125							750
	24 LCM Scraper	358					1432				1432				1432				1432				1432				1432				1432				1432				1432					12888
	32 tonne Dump Truck	226					678						226	678						226	678						226	678						226	678									4294
	5.4 m^3 Front-End Loader	285					285		285		285		285		285		285		285		285		285		285		285		285		285		285		285		285		285					4845
	SUB-TOTAL						4276		867	508	2125	516	1684	678	2707	712	585	312	3298	430	1545	508	2125	720	1458		3233	1594	285	312	3298	408	867	734	2803	720	1785		2707	916				44689
93000	PIT SERVICES																																											
93200	IN-PIT DRAINAGE																																											
	Diesel Pumps	6					65										65										65										65							260
	1 m^3 Backhoe	150		150					150					150					150					150					150					150					150					1200
	3-5 tonne Hiab Truck	25		25					25					25					25					25					25					25					25					200
	SUB-TOTAL			175			65		175					175			65		175					175			65		175					175			65		175					1660
94000	MINING EQUIPMENT																																											
94100	ELECTRIC SHOVELS																																											
	16.8 m^3 Shovel	2960				8880		2960	2960		2960												2960	2960		8880		2960																32560
94200	SPECIALIZED EXCAVATORS																																											
	24 LCM Scraper	358					716				716				716				716				716				716				716				716				716					6444
94300	SPECIALIZED EXCAVATORS - PARTINGS			NOT USED																																								
94400	HAULAGE TRUCKS																																											
	109 tonne Coal Truck	622						2488		622	622	622	622					2488		622		622	622					1866				622	622					1244						13684
	136 tonne Waste Truck	713					7130		2139	1426							7130							2139			7130										2852							29946
	SUB-TOTAL						7130	2488	2139	2048	622	622	622				7130	2488		622		622	622	2139			7130	1866				622	622				2852	1244						43630

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MOBILE MINING AND AUXILIARY EQUIPMENT CAPITAL COST SCHEDULE
TABLE A5-3 (SHEET 2)

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

TABLE A5-3
SHEET 2 OF 3

ACCOUNT CODE	DESCRIPTION	UNIT COST	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD																																			TOTAL
			-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	
94000	MINING EQUIPMENT (continued)																																											
94500	IN-PIT CONVEYORS		INCLUDED IN ACCOUNTS 95000 AND 97000																																									
94600	IN-PIT MATERIAL SIZING		INCLUDED IN ACCOUNTS 95000 AND 97000																																									
94700	AUXILIARY MINING EQUIPMENT																																											
94710	DOZERS																																											
	D-9 c/w Winch	300					300			300			300			300			300			300			300			300			300			300			300			300				3600
	D-9 c/w Ripper	312					936		624	936		624	936		624	936		624	936		624	936		624	936		624	936		624	936		624	936		624	936		624	936				18096
	824 Wheeled Dozer	196					196			196			196			196			196			196			196			196			196			196			196			196				2352
	D-8	205					410		410	410		410	410		410	410		410	410		410	410		410	410		410	410		410	410		410	410		410	410		410	410				9430
	SUB-TOTAL						1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842		1034	1842				33478
94715	CRANES																																											
	90 tonne	477					477										477										477										477							1908
	45 tonne	237				474										474										474										474								1896
	15 tonne	126				126		126	252		126	252		126	252		126	252		126	252		126	252		126	252		126	252		126	252		126	252		126	252		126			4410
	SUB-TOTAL					600	477	126	252		126	252		126	252	474	603	252		126	252		126	252		600	729		126	252		126	252		126	726	477	126	252		126			8214
94720	16 G GRADERS						408				408				408				408				408				408				408				408				408					3672
94725	LIGHT VEHICLES																																											
	Fire Truck	60				60			60							60			60							60			60							60			60					480
	Ambulance	17				17		17	17		17	17		17	17		17	17		17	17		17	17		17	17		17	17		17	17		17	17		17	17					391
	Personnel Bus (24 passengers)	18					90			90			90			90			90			90			90			90			90			90			90			90				1080
	Personnel Bus (8 passengers)	12					24			24			24			24			24			24			24			24			24			24			24			24				288
	SUB-TOTAL					77	114	17	77	114	17	17	114	17	17	174	17	17	174	17	17	114	17	17	114	77	17	114	77	17	114	17	17	114	17	77	114	17	77	114				2239
94730	FRONT END LOADER - 11.5 m^3	555					555	555				555	555				555	555				555	555				555	555				555	555			555	555						7770
94735	RECOVERY VEHICLES																																											
	Hi-Boy	40					40										40										40										40							160
	Lo-Boy c/w Tractor	80					80										80										80										80							320
	SUB-TOTAL						120										120										120										120							480
94740	DRILLING AND BLASTING EQUIPMENT																																											
	Air Track c/w Compressor	122					122					122																																244
	Truck Mounted Drill	100				100								100									100									100												400
	Blasting Truck	62					62										62										62										62							248
	Auger Truck-Mounted Drill	185					185					185					185					185					185					185					185							1295
	SUB-TOTAL					100	369					307		100			247					185	100				247					285					247							2187

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MOBILE MINING AND AUXILIARY EQUIPMENT CAPITAL COST SCHEDULE
TABLE A5-3 (SHEET 3)

ACCOUNT CODE	DESCRIPTION	UNIT COST	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
94000	MINING EQUIPMENT (continued)																																											
94700	AUXILIARY MINING EQUIPMENT (continued)																																											
94745	MOBILE SERVICING EQUIPMENT																																											
	5 tonne Service Trucks	18				36		36	36		36	36		36	36		36	36		36	36		36	36		36	36		36	36		36	36		36	36		36	36		36			864
	Tire Trucks	35					35			35			35			35			35			35			35			35			35			35			35			35				420
	Line Trucks	65				65	130				65	130				65	130				65	130				65	130				65	130				65	130							1365
	Fuel Trucks	85					170			170			170			170			170			170			170			170			170			170			170			170				2040
	3-5 tonne Hiab Trucks	25				50			50			50			50			50			50			50			50			50			50			50			50					600
	Lube Trucks	55				110			110			110			110			110			110			110			110			110			110			110			110					1320
	SUB-TOTAL					261	335	36	196	205	101	326	205	36	196	270	166	196	205	36	261	335	36	196	205	101	326	205	36	196	270	166	196	205	36	261	335	36	196	205	36			6609
94750	WAREHOUSE MOBILE EQUIPMENT																																											
	Forklifts	120				360		120			360		120			360		120			360		120			360		120			360		120			240								3120
94755	MISCELLANEOUS EQUIPMENT																																											
	Calcium Chloride Spreader	7				14		14			14		14			14		14			14		14			14		14			14		14			14		14						196
	Steam Cleaners	60				120		120			120		120			120		120			120		120			120		120			120		120			120		120						1680
	Field Fuel and Lube Station	80								80										80										80										80				320
	Skid Mounted Light Towers	6					39										39										39										39							156
	Pickup Fleet	10					330	320	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	390	690	430	18380
	Truck Fleet - over 1 tonne	20					300	150	150			300	150	150			300	150	150			300	150	150			300	150	150			300	150	150			300	150	150			300	150	4650
	SUB-TOTAL					134	669	604	840	510	524	990	714	540	690	564	729	974	580	470	824	730	674	840	430	524	1029	714	540	770	564	690	974	580	390	824	769	674	840	510	390	990	580	25382
94760	INITIAL MINING EQUIPMENT SPARES																																											
	AT 6 1/2% OF CAPITAL COST					685	1191	485																																				2361
	TOTAL ACCOUNT 94700					2217	6080	1943	2399	2671	1536	3481	3550	819	2597	3684	2437	3148	3209	649	2748	3761	2036	2339	2591	1662	4465	3550	779	2269	3558	1839	3148	2741	977	3167	4459	1408	2807	2671	552	990	580	95512
97000	WASTE DISPOSAL EQUIPMENT																																											
97400	MOBILE HAULING EQUIPMENT																																											
	32 tonne Dump Trucks	226					906						453	906						453	906						453	906						453	906									6342
	5.4 m^3 Front-End Loader	285					285				285				285				285				285				285				285				285				285					2565
	SUB-TOTAL						1191				285		453	906	285				285	453	906		285				738	906			285			453	1191				285					8907
97470	DOZERS - D-8	205					410			410			410			410			410			410	410		410	410		410	410		410	410		410	410		410	410		410				7380
97480	COMPACTION EQUIPMENT																																											
	Wheeled type	213						213										213										213										213						852
	Vibrating type	213					426					426					426					426					426					426					426							2982
	SUB-TOTAL						426	213				426					426	213				426					426	213				426					426	213						3834
	TOTAL ACCOUNT 97400						2027	213		410	285	426	863	906	285	410	426	213	695	453	906	836	695		410	410	1164	1529	410		695	836		863	1601		836	623	285	410				20121

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SCHEDULE OF CAPITAL COSTS
SUMMARY CASH FLOW
TABLE A5-4

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

TABLE A5-4

ACCOUNT CODE	DESCRIPTION	PRE-PRODUCTION PERIOD -6	-5	-4	-3	-2	-1	PRODUCTION PERIOD 1	2	3	4	5	6	7	8	9	10	11	12
90000	ENGINEERING AND CONSTRUCTION	2000	4620	8280	9214	9868	9649	2000									100		
91000	MINE PROPERTY DEVELOPMENT		360	4755	12140	13885	1055	1207	508	2227	1090	1854	858	2727	882	1391	669	3348	610
92000	BUILDINGS AND STRUCTURES				5353	5119	2301	78	60		110	60		114	106	247	78	60	
93000	PIT SERVICES		175			256	165	221					175			65	82	175	
94000	MINING EQUIPMENT				11097	13926	7391	7498	7679	2874	4103	4172	1535	2597	3684	9567	5636	3925	1271
95000	COAL CONVEYING, CRUSHING, AND BLENDING				9283	15969	11648				811	2787		389					
97000	WASTE DISPOSAL EQUIPMENT				8591	16810	8629		3779	285	2908	6532	906	363	3763	426	213	695	453
98000	RECLAMATION AND ENVIROMENTAL PROTECTION			70	44	79	64	15	6	59	20	21	57	16	23	84	8	10	52
99000	CONTINGENCY	350	902	2293	9751	13285	7158	1212	1203	544	904	1542	353	620	845	1178	678	821	239
	TOTAL CAPITAL COSTS	2350	6057	15398	65473	89197	48060	12231	13235	5989	9946	16968	3884	6826	9303	12958	7464	9034	2625
	TOTAL CUMULATIVE TO FULL PRODUCTION	2350	8407	23805	89278	178475	226535	238766	252001	257990									

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD 13	14	15	16	17	18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35
90000	ENGINEERING AND CONSTRUCTION								100															
91000	MINE PROPERTY DEVELOPMENT	4040	1124	2338	720	1518		3233	1612	285	312	3298	680	995	734	2803	720	1758	20	2707	916			
92000	BUILDINGS AND STRUCTURES	78	116		114	60	46	355	60		78	60	32	114	60		130	325		78	60		110	19
93000	PIT SERVICES				175			65	82	175					175			65	82	175				
94000	MINING EQUIPMENT	2748	4383	3374	7438	2591	10542	12311	8376	779	2269	4274	2461	3770	2741	1693	3162	7311	2652	3523	2671	552	990	580
95000	COAL CONVEYING, CRUSHING, AND BLENDING		389						2980	389							389							
97000	WASTE DISPOSAL EQUIPMENT	6539	992	695		410	410	3442	7600	566		695	836		863	1601	156	836	623	285	410			
98000	RECLAMATION AND ENVIROMENTAL PROTECTION	19	32	93	22	38	56	48	32	83	12	12	84	25	6	73	7	42	79	15	21	71	20	31
99000	CONTINGENCY	1392	704	650	846	461	1105	1945	2084	228	267	834	409	490	458	617	456	1034	345	678	408	62	112	63
	TOTAL CAPITAL COSTS	14766	7740	7150	9315	5078	12159	21399	22926	2505	2938	9173	4502	5394	5037	6787	5020	11371	3801	7461	4486	685	1232	693
	TOTAL CUMULATIVE TO FULL PRODUCTION																							

ACCOUNT CODE	DESCRIPTION	POST PRODUCTION PERIOD 36	37	38	39	40	41	42	43	44	45	TOTAL
90000	ENGINEERING AND CONSTRUCTION											45831
91000	MINE PROPERTY DEVELOPMENT											79379
92000	BUILDINGS AND STRUCTURES											15581
93000	PIT SERVICES											2308
94000	MINING EQUIPMENT											178146
95000	COAL CONVEYING, CRUSHING, AND BLENDING											45034
97000	WASTE DISPOSAL EQUIPMENT											81312
98000	RECLAMATION AND ENVIROMENTAL PROTECTION	89	26	22	89	42	9	31	2	19	1	1879
99000	CONTINGENCY	9	3	2	9	4	1	3	-	2	-	59509
	TOTAL CAPITAL COSTS	98	29	24	98	46	10	34	2	21	1	508979
	TOTAL CUMULATIVE TO FULL PRODUCTION											

CANADIAN $ 000'S OCTOBER 1977

M11

TABLE A5-5

Summary of Mining Equipment
End of Year 17

Hat Creek Project Mining 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	4290 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

TABLE A5-6

MANPOWER SCHEDULE

PRE-PRODUCTION MINE CONSTRUCTION AND PROJECT MANAGEMENT

HAT CREEK PROJECT MINING REPORT 1978

	YEAR OF PROJECT						
	-6	-5	-4	-3	-2	-1	1
Project Management and Design Engineering	10	40	65	45	45	20	10
Field Engineering	-	-	20	40	40	20	10
Construction Labour	-	-	210	235	140	65	50
TOTAL MANPOWER REQUIREMENTS	10	40	295	320	225	105	70

TABLE A5-7

MANPOWER SCHEDULE
SUMMARY PEAK REQUIREMENTS

HAT CREEK PROJECT MINING REPORT 1978

	Pre-Production	1	2	3	4	5	6-15	16-25	26-35	36-45
Administration	72	90	90	90	90	90	90	90	90	-
Reclamation and Pollution Control	12	19	19	19	19	19	19	23	23	13
Mine Supervision and Engineering	72	85	85	85	85	85	85	85	85	-
Mine Operating Labour	285	328	335	339	351	370	386	412	347	-
Maintenance Supervision	46	50	50	50	50	50	50	50	50	-
Maintenance Labour	262	316	322	324	328	331	332	345	293	-
TOTAL MANPOWER REQUIREMENTS	749	888	901	907	923	945	962	1005	888	13

TABLE A5-8

Manpower Schedule
Administration and Services
Reclamation and Environmental Protection
Hat Creek Project Mining Report 1978

	YEAR OR PERIOD Pre-Prod.	1	2	3	4	5	6-15	16-25	26-35	36-45
ADMINISTRATION AND SERVICES										
Mine Manager	1	1	1	1	1	1	1	1	1	-
Assistant Mine Manager	1	1	1	1	1	1	1	1	1	-
Superintendent - Mine Production	1	1	1	1	1	1	1	1	1	-
" - Mine Engineering	1	1	1	1	1	1	1	1	1	-
" - Maintenance	1	1	1	1	1	1	1	1	1	-
" - Administration	1	1	1	1	1	1	1	1	1	-
Comptroller	1	1	1	1	1	1	1	1	1	-
Accounting	8	11	11	11	11	11	11	11	11	-
Purchasing and Warehousing	13	15	15	15	15	15	15	15	15	-
Personnel, Safety, and Security	17	19	19	19	19	19	19	19	19	-
Clerical, Typing, etc.	24	34	34	34	34	34	34	34	34	-
	69	86	86	86	86	86	86	86	86	-
5% Contingency	3	4	4	4	4	4	4	4	4	-
TOTAL ADMINISTRATION AND SERVICES	72	90	90	90	90	90	90	90	90	-
RECLAMATION AND ENVIRONMENTAL PROTECTION										
Environmental Engineer	1	1	1	1	1	1	2	2	2	2
Environmental Technician	1	1	1	1	1	1	1	1	1	1
Secretary	1	1	1	1	1	1	1	1	1	1
Pollution Control Supervisor	1	2	2	2	2	2	2	2	2	2
Pollution Control Engineer	1	2	2	2	2	2	2	2	2	2
Pollution Control Technician	1	1	1	1	1	1	1	1	1	1
Field Staff (summer only)	6	8	8	8	8	8	8	12	12	4
	12	18	18	18	18	18	18	22	22	13
5% Contingency	-	1	1	1	1	1	1	1	1	-
TOTAL RECLAMATION AND POLLUTION CONTROL	12	19	19	19	19	19	19	23	23	13

TABLE A5-9

Manpower Schedule

Mine Supervision and Engineering

Hat Creek Project Mining Report 1978

	YEAR OR PERIOD Pre-Prod.	1	2	3	4	5	6-15	16-25	26-35
MINE SUPERVISION									
Development Engineer	1	1	1	1	1	1	1	1	1
Shift Supervisor	4	4	4	4	4	4	4	4	4
Pit Foreman	4	8	8	8	8	8	8	8	8
Dumps Foreman	4	4	4	4	4	4	4	4	4
Roads and Pioneering Foreman	4	8	8	8	8	8	8	8	8
Blasting Foreman	1	1	1	1	1	1	1	1	1
Dewatering Foreman	1	1	1	1	1	1	1	1	1
Conveying and Blending Foreman	8	8	8	8	8	8	8	8	8
Mine Clerk	4	4	4	4	4	4	4	4	4
Secretary	1	1	1	1	1	1	1	1	1
	32	40	40	40	40	40	40	40	40
ENGINEERING									
Mine Engineer	1	1	1	1	1	1	1	1	1
Blasting Engineer	1	1	1	1	1	1	1	1	1
Contract Engineer	1	1	1	1	1	1	1	1	1
R & D Engineer	-	1	1	1	1	1	1	1	1
Planning Engineer	6	7	7	7	7	7	7	7	7
Geologist	2	3	3	3	3	3	3	3	3
Geological and Grade Control Technician	14	15	15	15	15	15	15	15	15
Training Officer	2	2	2	2	2	2	2	2	2
Surveyor	4	4	4	4	4	4	4	4	4
Rodman	6	6	6	6	6	6	6	6	6
	37	41	41	41	41	41	41	41	41
Combined Total	69	81	81	81	81	81	81	81	81
5% Contingency	3	4	4	4	4	4	4	4	4
TOTAL MINE SUPERVISION AND ENGINEERING	72	85	85	85	85	85	85	85	85

TABLE A5-10

Manpower Schedule

Mine Operating Labour

Hat Creek Project Mining Report 1978

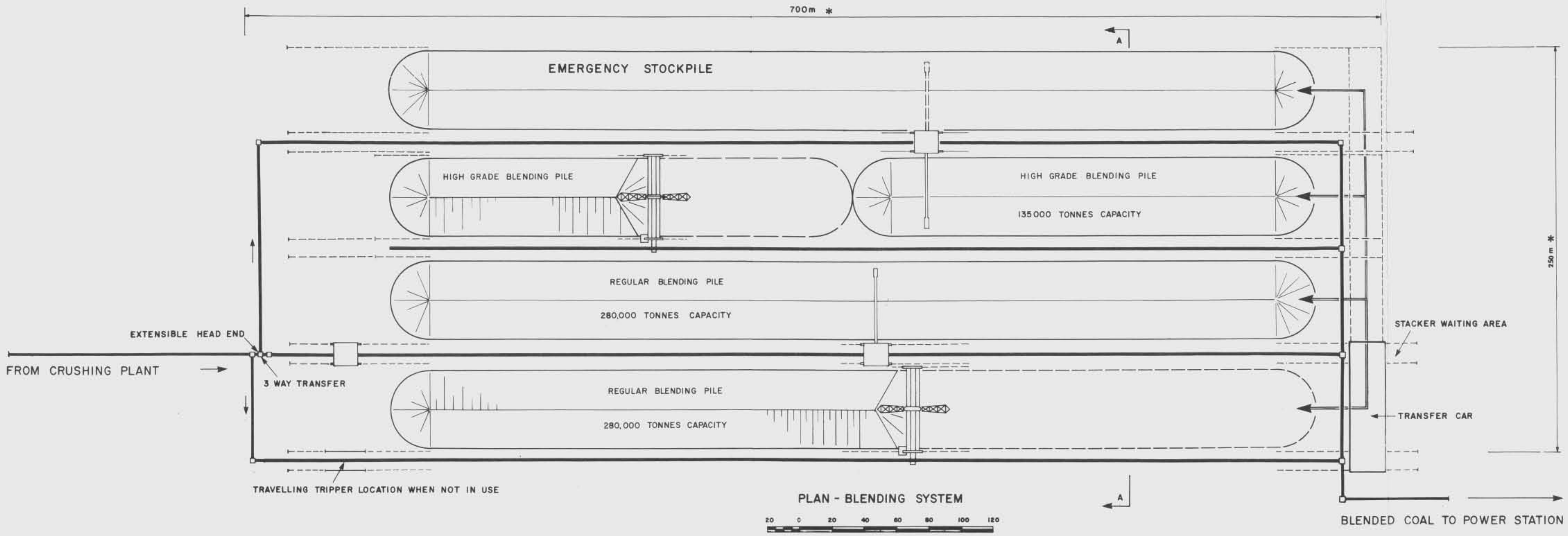
	YEAR OR PERIOD Pre-Prod.	1	2	3	4	5	6-15	16-25	26-35
Shovel Operator	12	16	16	16	16	16	16	20	12
Shovel Oiler	12	16	16	16	16	16	16	20	12
11.5 m^3 Front End Loader	8	8	8	8	8	8	8	8	8
Dozer Operator	42	44	44	44	44	45	50	52	54
Haul Truck Operator	40	58	64	64	68	80	84	88	50
Scraper Operator	8	8	8	12	12	12	12	16	18
Grader Operator	16	16	16	16	20	20	20	24	24
5.4 m^3 Front End Loader	6	8	8	8	8	8	8	8	8
Water Truck Operator	4	6	6	6	6	6	6	6	6
Crusher Operator	2	2	2	2	2	2	2	2	2
Gradall Operator	1	1	1	1	1	1	1	1	1
Backhoe Operator	1	1	1	1	1	1	1	1	1
Compactor Operator	2	2	2	2	2	2	2	2	2
Drill Operator	2	4	4	4	4	4	4	4	4
Blasting Truck Driver	2	2	2	2	2	2	2	2	2
Coal Conveyor System Operator	31	31	31	31	31	31	32	32	32
Waste Conveyor System Operator	23	26	26	26	28	32	36	36	30
Warehouse and Miscellaneous Labour	36	36	36	36	36	36	36	36	36
	248	285	291	295	305	322	336	358	302
15% Contingency	37	43	44	44	46	48	50	54	45
TOTAL MINE OPERATING LABOUR	285	328	335	339	351	370	386	412	347

TABLE A5-11

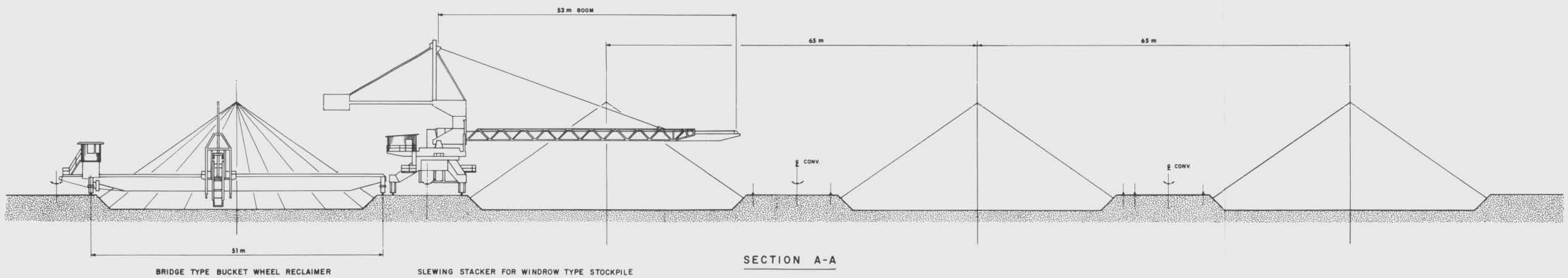
Manpower Schedule
Maintenance Supervision
Maintenance Labour
Hat Creek Project Mining Report 1978

	Y E A R O R P E R I O D								
	Pre-Prod.	1	2	3	4	5	6-15	16-25	26-35
MAINTENANCE SUPERVISION									
Equipment Shops									
Maintenance Supervisor	1	1	1	1	1	1	1	1	1
General Foreman	1	1	1	1	1	1	1	1	1
Shop Boss	8	8	8	8	8	8	8	8	8
Field Boss	8	8	8	8	8	8	8	8	8
General Shops									
Shop Supervisor	1	1	1	1	1	1	1	1	1
Shop Boss	9	9	9	9	9	9	9	9	9
Electrical Foreman	2	2	2	2	2	2	2	2	2
Electrical Boss	5	5	5	5	5	5	5	5	5
Planner	4	7	7	7	7	7	7	7	7
Training Officer	3	3	3	3	3	3	3	3	3
Clerk	2	3	3	3	3	3	3	3	3
	44	48	48	48	48	48	48	48	48
5% Contingency	2	2	2	2	2	2	2	2	2
TOTAL MAINTENANCE SUPERVISION	46	50	50	50	50	50	50	50	50
MAINTENANCE LABOUR									
Shop Labour									
H. D. Mechanic	50	70	70	70	70	70	67	71	58
Auto Mechanic	17	20	20	20	20	20	20	20	20
Tiremen	10	10	10	10	10	10	10	10	8
Welder	16	20	20	20	20	20	20	20	16
Machinist	10	12	12	12	12	12	12	12	10
Carpenter	8	8	8	8	8	8	8	8	8
Painter	2	2	2	2	2	2	2	2	2
Pipefitter	6	6	6	6	6	6	6	6	6
Sheetmetal Worker	2	2	2	2	2	2	2	2	2
Electrician	43	43	43	43	43	43	43	43	43
Labourer	11	14	14	14	14	14	13	13	12
Field Labour									
Mechanic	20	22	22	22	22	22	22	24	20
Lube-Service Operator	12	12	12	12	12	12	12	12	12
Conveyor Mechanic	15	26	31	33	36	39	44	49	30
Belt Vulcaniser	2	4	4	4	4	4	4	4	4
Crane Operator	4	4	4	4	4	4	4	4	4
	228	275	280	282	285	288	289	300	255
15% Contingency	34	41	42	42	43	43	43	45	38
TOTAL MAINTENANCE LABOUR	262	316	322	324	328	331	332	345	293

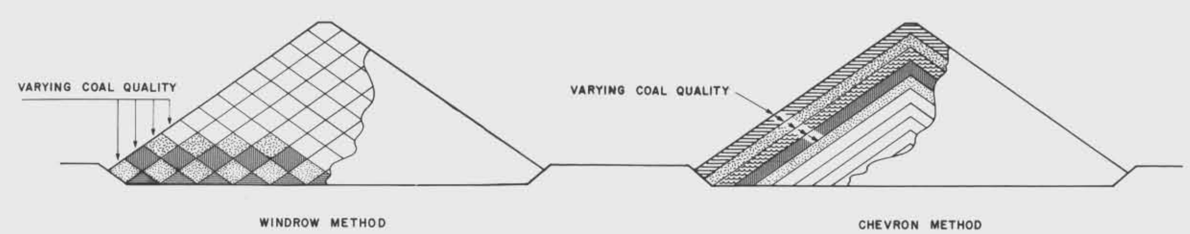
cominco-monenco joint venture



PLAN - BLENDING SYSTEM



SECTION A-A



SECTION - ALTERNATIVE STOCKPILES

* APPROXIMATE AREA DIMENSIONS

PLATE A5-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

GENERAL ARRANGEMENT
COAL BLENDING SYSTEM

SECTION A6.0 - MINE SUPPORT FACILITIES

The mine service area as shown on Plate A6-1 would be located to the northeast of the pit and all of the necessary maintenance and ancillary buildings would be located in this area. In all cases consideration has been given to functionality, safety, fire protection and provision for expansion.

A6.1 BUILDINGS AND STORAGE FACILITIES

A brief description of the facilities and their purposes is provided below. Building heating requirements were assumed to be met by electricity.

(a) Administration Building

A two-storey building would house staff for administration, accounting, data processing, personnel, purchasing, geology and mine planning.

(b) Maintenance Complex

This large structure would contain the following maintenance related functions:

1. Equipment maintenance:

   - 7 truck repair bays
   - 4 tractor repair bays
   - 8 light vehicle bays
   - 2 steam cleaning bays

2. Shops:

   - Welding and fabrication
   - Machine

A6.1 BUILDINGS AND STORAGE FACILITIES - (Cont'd)

- Electrical
- Radio and instrument repair
- Component repair

3. Warehouse and tool crib.

4. Supervising and planning offices.

5. Emergency services:

   - Fire truck
   - Ambulance
   - First aid

6. Training centre.

7. Lunchroom and kitchen facilities.

(c) Mine Services Building

The mine services building would include provisions for:

- sheet metal and pipefitting shop
- carpenters' shop
- painters' shop
- service vehicle storage
- material storage

(d) Field Maintenance Centre

The field maintenance centre would serve as headquarters for field maintenance crews.

A6.1 BUILDINGS AND STORAGE FACILITIES - (Cont'd)

(e) Rubber Repair Shop

Facilities for repairs to tires, conveyor belting and trailing cables would be located in the rubber repair shop.

(f) Laboratories

Two separate laboratories are envisaged:

1. An analytical/environmental laboratory to be used for work on coal samples, drill cores and environmental studies.

2. An environmental services laboratory located away from the central core on presently developed agricultural land to provide greenhouse and experimental growth facilities.

(g) Mine Dry

A mine dry would be provided to serve as the main point of dispatch and return for the mine workers. Provision would be made for 700 "double" lockers, shower and sanitary facilities, marshalling area and offices for mine supervisory staff.

(h) Lubricant Storage

Heated and insulated, the lubricant storage building would house bulk lube oils and greases to be pumped on demand to the various dispensing racks in the shops and mobile field service vehicles.

(i) Fuel Storage and Dispensing Area

A fuel storage and dispensing area would provide bulk loading and unloading facilities, tank farm and dispensing pumps for diesel oil, gasoline, waste oil and anti-freeze. A satellite station would be located in the pit.

A6.1 BUILDINGS AND STORAGE FACILITIES - (Cont'd)

(j) Storage Areas

Various open storage areas were located and sized to accommodate their expected use. Consideration in each case was given to accessibility, ease of materials handling, security and need for future expansion.

A6.2 MINE AREA DRAINAGE

Preliminary engineering studies developed an overall area drainage plan, the objective of which was to protect the mining operation from major flood damage while preserving the necessary continuity and quality of the existing natural drainage system in accordance with existing environmental guidelines.

The major elements of these studies were:

1. The estimation of drainage flows from natural watersheds and future disturbed watersheds in the mining area.

2. Diversion of runoff from minor creeks and natural watersheds entering the mine and waste disposal area.

3. Collection and disposal of surface runoff from precipitation falling directly on the mine site.

4. Disposal of subsurface water from pit dewatering operations or that appearing as seepage from stockpiles and waste dumps.

5. Disposal of sewage from the mine service complex.

Previous studies of the diversion of Hat and Finney creeks were adopted and the recommended canal scheme on the east flank of the mine was incorporated into overall drainage planning.

A6.2 MINE AREA DRAINAGE - (Cont'd)

A schematic flow chart showing the mine drainage system and the wastewater treatment systems is shown on Plate A6-2. The volumes of water to be treated were estimated from hydrology and seepage data; flow data are presented on Table A6-3. The individual elements of the proposed system are described in the following sub-sections. Plate A6-3 shows the overall plan of the drainage treatment systems developed. Main elements of the system would be:

(a) Diversion of Existing Creeks

Hat and Finney creeks must be diverted prior to the start of mining. Diversions are described in Appendix C. The lower reaches of Medicine Creek should not require diversion until about year 15 when waste dumping in that valley would be expected to begin.

Construction of perimeter diversion canals on the west sidehill from Houth Meadows to Finney Creek, 6.7 km (4.2 mi), and two small watershed diversions north of the Houth Meadows dump are anticipated.

(b) Drainage of Lakes

To improve stability of the west slide areas, Aleece Lake and 20 to 30 small ponds and sloughs should be drained, prior to commencement of mining.

Drainage must be collected, treated and discharged clear of the mining operation.

(c) Drainage Within the Mine Area

The pit would be surrounded by a major perimeter access road. Drainage from adjacent ditches and from conveyor ways would be directed to proposed sedimentation lagoons.

A6.2 MINE AREA DRAINAGE - (Cont'd)

Surface runoff and leachates from waste dumps, coal storage areas, conveyor ways and other surface facilities would be similarly collected for treatment.

Discharge from mine dewatering, wells and in-pit surface sumps would be pumped to these lagoons.

(d) Treatment of Drainage Flows

To satisfy regulatory requirements, drainage from the mine should meet present level A objectives mine effluent discharge prescribed by the B.C. Department of Environment (1976).

Prediction of the quality of surface water drainage from areas disturbed by mining indicate that removal of suspended sediment would be required prior to discharge to streams (Beak, 1978). Sedimentation tests on laboratory prepared samples of slurry indicated that chemical coagulants would be necessary to obtain acceptable suspended sediment levels in sedimentation lagoons within realistic detention times (B.C. Research, 1978).

Projections of the quality of leachates from coal, low-grade coal, and mine waste based on laboratory tests indicate that these effluents would be unfit for stream discharge and would require chemical treatment or disposal in a zero discharge system (Beak, 1978).

A6.3 UTILITIES

(a) Power Supply and Distribution

A network to supply power to the pit, waste dumps, and support facilities, would be developed as shown on Plate A6-4. The electrical network would include all electrical equipment required to supply power from the 60 kV buses of the proposed Hat Creek powerplant to the open pit and dump areas, and to distribute the power within these areas to the shovels, conveyors, spreaders, and the crushing and blending equipment. The network would also include supply for the various service buildings and provide the construction power required during the development phase of the mine.

Consideration was given to the load fluctuations resulting from electric shovel cycles and the start-up of large motors.

A summary of estimated load requirements is given in Table A6-2.

(b) Water Supply and Sewerage

The total estimated requirements of the mine are not large. No significant process consumption is involved. Potable water, fire protection, irrigation and dust control are the main requirements.

Estimated daily requirements and sources of supply are indicated below:

	Daily* Requirement m^3	(yd^3)	Source
Potable water and fire protection	395	(510)	Offsite construction supply
Irrigation	505	(660)	Pit rim reservoir on Hat Creek
Dust control	2000	(2600)	Mine area drainage and dewatering
	2900 m^3/day	(3700 yd^3/day)	

* Maximum average daily assumed demand at full mine development.

A6.3 UTILITIES - (Cont'd)

Sanitary sewage would be pretreated in an oxidation ditch system, and recycled to dust control use on haul roads and coal stockpiles.

TABLE A6-1

MINE DRAINAGE AND WATER SUPPLY SYSTEM FLOWS

HAT CREEK PROJECT MINING REPORT 1978

Code (As on flow chart)	Description	Watershed Area Km^2	Flow Frequency	Flow Type	Estimated Flow m3/S	Estimated Volume $m^3 \times 10^3$	Sources of Data	Assumptions and Remarks
DIVERSION DRAINS								
D1	Upper Southwest Perimeter	2.5	100R	P	1.4		1	8mm Runoff
D2	Finney Creek Canal	19.5	1000F	P	4.3		2	
D3	Hat Creek Upstream of Headworks Reservoir	248	1000F	P	27		2	
D4	Ambusten Creek and Southeast Watershed	35	1000F	P	7		2	
D5	South Medicine Creek	6	100R	P	2		1	5mm Runoff
D6	Pit Rim Pump	4.4	-	P	0.12		5	Flow limited to pump capacity
D7	North Medicine Creek	43	1000F	P	8.5		2	Excludes ash pond and reservoir 6 Km^2
L1	Canal Leakage	-	DY	M	.01-.025		5	
D8	Hat Creek Downstream of Medicine Creek	360	1000F	P	34		2	Includes Dump Area
D9	East Watershed	2	100R	P	1.2		1	8mm Runoff
D10	West Perimeter	25	1000F	P	5		2	
D11	Hat Creek Downstream of Mine	383	1000F	P	35		2	
D12	North Perimeter	1	100R	P	1		1	5mm Runoff
P1	Lower Southwest Diversion	0.9	100R	P	0.7		1	8mm Runoff
P2	Southeast Diversion	0.5	100R	P	0.5		1	8mm Runoff
P3	Watershed below Canal	3	100R	P	1.5		1	8mm Runoff
MINE SURFACE WATER COLLECTION SYSTEM						24 Hour Volumes		
S1	Disturbed Area Drainage to North Valley Sedimentation Lagoons	3-6 Disturbed +3 Reclaimed	10R 10R	- -	- -	45-90 0-15	1 1	15mm Runoff 5mm Runoff
S2	Mine Service Area Drainage	0.35	10R	-	-	5.3	1	15mm Runoff
S3	Washdown Water		DY	M	-	0.09	1	
S4	Pit Surface Water	1-6	10R	-	-	17	1	Limited to Pump Capacity
S5	Surficials Groundwater and Seepage		DY DY	M M	.011-.024 .006-.016		6 4	Pumps and Seepage Pumps Only
S6	Disturbed Area Runoff Medicine Creek	2.5 Disturbed +1.3 Reclaimed	10R 10R	- -	- -	0-37.5 0-6.5	1 1	15mm Runoff 5mm Runoff
ZERO DISCHARGE SYSTEM						Annual Volumes		
Z1	Sanitary Effluent	-	DY	M	0.0016	51	7	700 Shifts x 0.2 m^3
Z2	Coal Blending Runoff and Seepage	0.2	A	M	-	2-20	7	10-100mm Annual Yield
Z3	Low-Grade Coal Runoff and Seepage	0.33	A	M	-	3-30	7	10-100mm Annual Yield
Z4	Embankment Seepage Houth Meadows	-	A	M	-	100-550	6	
L2	Seepage lost to Evaporation	-	A	M		2-20	7	1-10 ha x 200mm Loss
Z5	Holding Pond Inflow	-	A	M	-	154-631	7	
Z6	Coal and Bedrock Groundwater	-	A	M	-	9-25	4, 7	5% of Groundwater Flows
Z7	Dust Control Use	-	A	M	-	250-370	7	
L3	Holding Pond Loss	2	A	M	-	4	7	200mm Loss
Z8	Medicine Creek Embankment Seepage	-	A	M	-	55-365	6, 7	
L4	Seepage Lost to Evaporation	-	A	M	-	2-20	7	1-10 ha x 200mm Loss
WATER SUPPLY						Annual Volumes		
H1	Supply to Mine Service Area	-	DY	M	0.0041	101	7	700 Shifts + Garden + Washdown
H2	Supply to Revegetation Nursery	-	A	M	-	75	7	10 Ha.

KEYS TO SYMBOLS IN TABLE:

- 100R - 100 year recurrence interval rainstorm flood after snowmelt
- 1000F - 1000 year recurrence interval freshet flood during snowmelt
- 10R - 10 year recurrence interval rainstorm flood after snowmelt
- DY - DAILY
- A - ANNUAL
- P - PEAK Discharge
- M - MEAN Discharge

SOURCES OF DATA:

1. CMJV Rainstorm Nomograph
2. CMJV Freshet Nomograph
3. Hat Creek Flow Records
4. Golder Associates Geotechnical Report 1977
5. B.C. Hydro H.E.D.D. "Diversion of Hat and Finney Creeks" 1978
6. Beak Consultants "Hydrology Drainage and Water Use-Impact Report" 1978
7. CMJV Estimate

NOTE:

These data are based on Preliminary Mine Planning Data, Hydrological and Hydrogeological Studies. Surface water flows from small watersheds and seepage flows are estimates based on several arbitrary assumptions as to runoff infiltration factors and hydraulic conductivities therefore they should be upgraded when further site specific data becomes available.

Where a range of flow is shown this identifies the variability of flow in terms of the assumptions made.

Areas used correspond to the estimated maximum effective area of natural watersheds, disturbed areas or mine facilities to be drained.

TABLE A6-2

TOTAL MINE ELECTRICAL LOAD DURING PEAK YEARS

	Installed Load	Typical Load	Annual Average Load
Conveyor Load	53,500	31,500	15,600
Maintenance Complex	2,760	1,681	1,320
Mine Dry Building	841	427	363
Rubber Repair Building	359	165	125
Mine Service Building	805	383	278
Administration Building	644	349	178
	58,909 kW	34,505 kW	21,618 kW
In-pit Loads			
8 - Shovel Sub-stations (1 spare)	12,000	5,900	2,268
Pumping and Miscellaneous	455	180	126
	12,455 kW	6,080 kW	2,394 kW
Total	71,364 kW	40,585 kW	24,012 kW

cominco-monenco joint venture

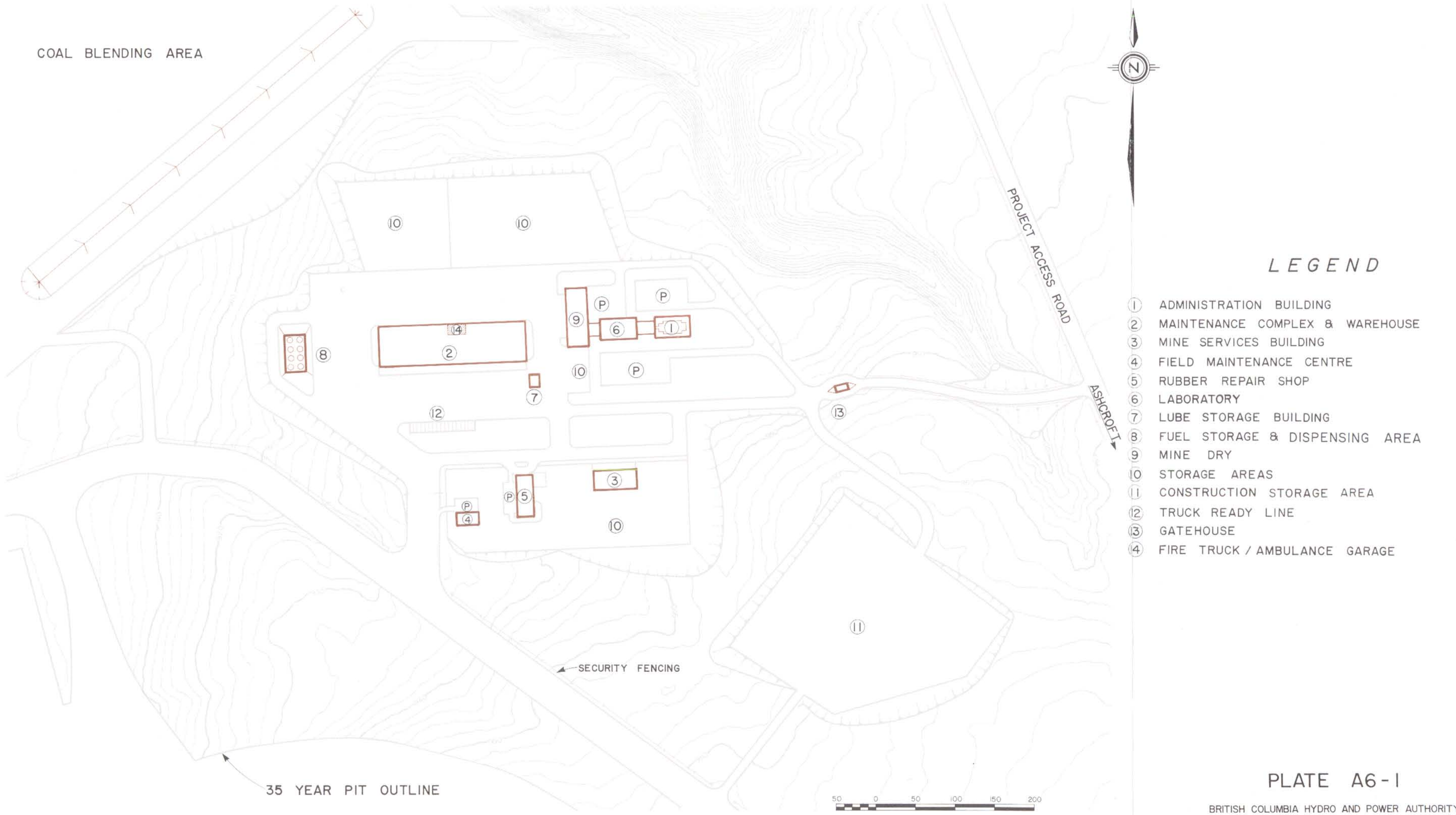


PLATE A6-1

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

MINE SERVICE AREA
GENERAL ARRANGEMENT

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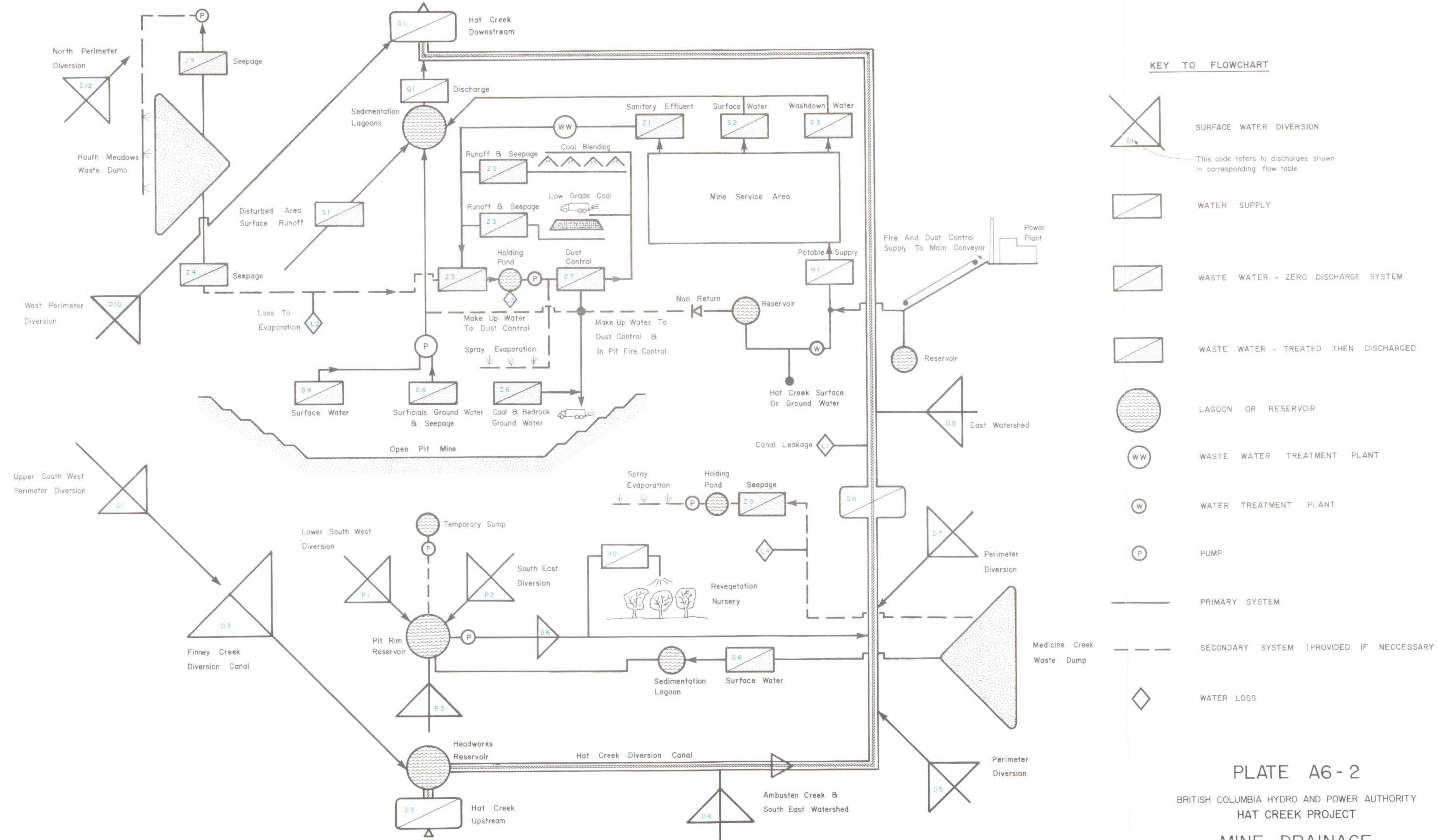
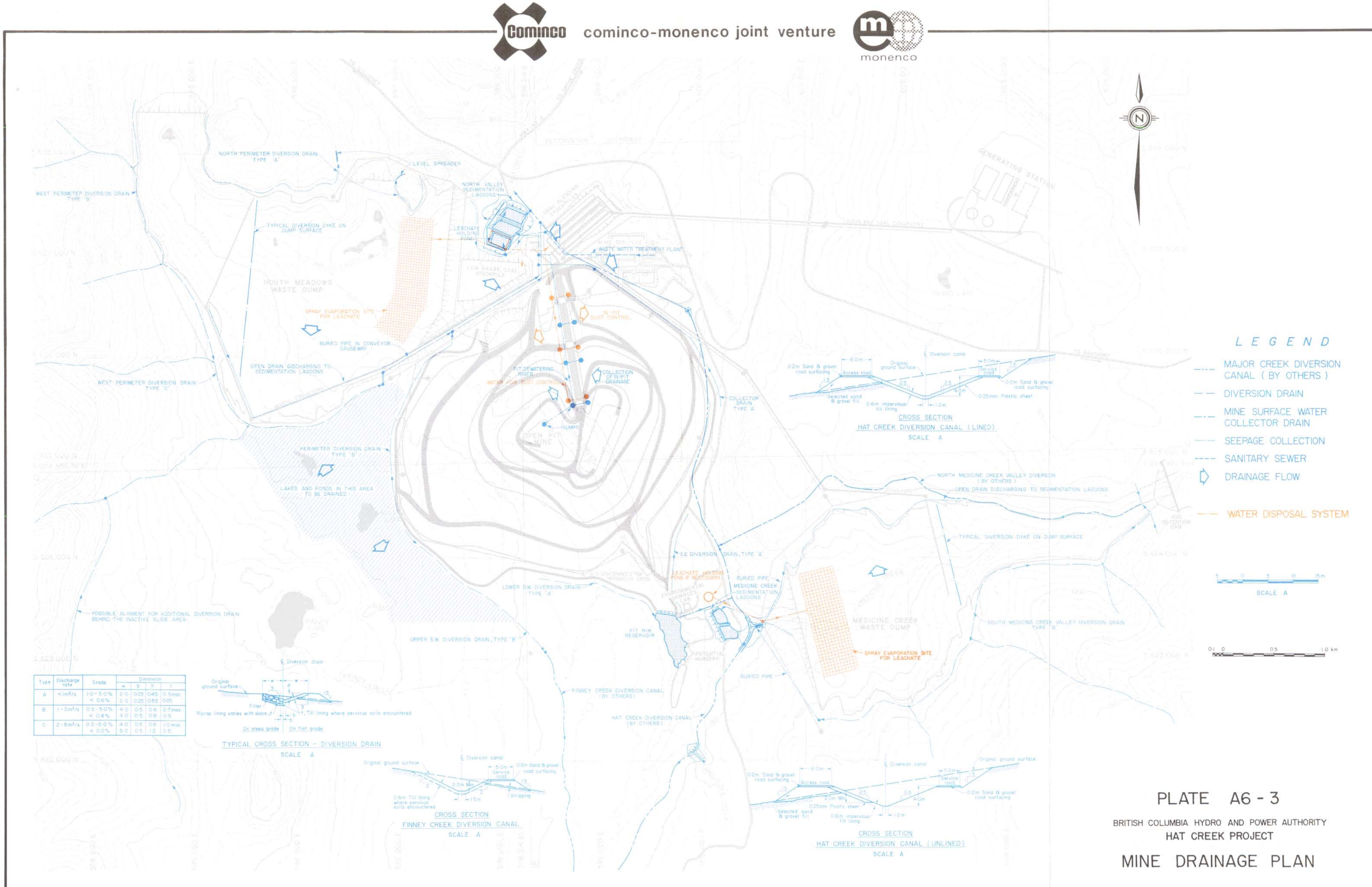


PLATE A6-2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY

HAT CREEK PROJECT

MINE DRAINAGE & WATER SUPPLY FLOWCHART



Type	Discharge rate	Grade	Dimension			
			w	b	h	t
A	< 1m³/s	1.0 – 5.0 %	2.0	0.25	0.45	0.5 max
		< 0.6 %	2.0	0.25	0.55	0.25
B	1 – 2m³/s	0.5 – 5.0 %	4.0	0.5	0.6	0.7 max
		< 0.4 %	4.0	0.5	0.8	0.5
C	2 – 5m³/s	0.2 – 5.0 %	4.0	0.5	0.8	1.0 max
		< 0.2 %	5.0	0.5	1.2	0.6

PLATE A6 - 3

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

MINE DRAINAGE PLAN

cominco-monenco joint venture

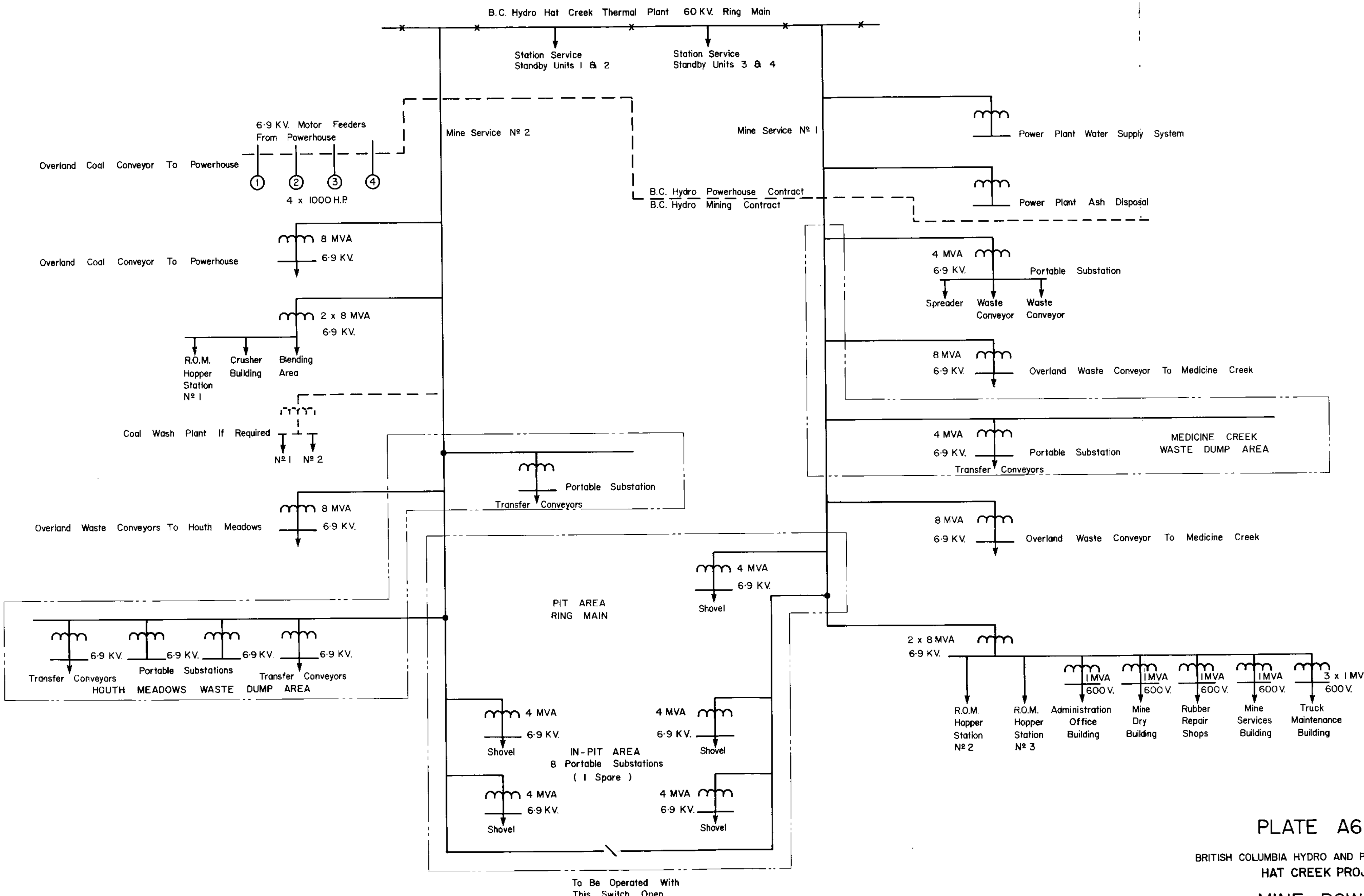


PLATE A6-4

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

MINE POWER
DISTRIBUTION NETWORK
SINGLE LINE DIAGRAM

SECTION A7.0 - RECLAMATION AND ENVIRONMENTAL PROTECTION

The project area is situated within the Hat Creek drainage basin. Medicine, Finney, Ambusten and Houth creeks generally drain northwards into Hat Creek, which also flows north and then east to the Bonaparte River, thence joining the Thompson River system just north of Ashcroft. The water bodies of significance in the general project area are Aleece and Finney lakes.

The regional climate is classified as continental and is typified by long cold winters and short, warm summers. Semi-arid conditions prevail; precipitation is estimated to be only 317 mm (12.5 in) annually, of which approximately one-half falls as snow. Winds are associated with the mountain/valley topography and are channelled predominantly upslope from the north to the south and southeast.

The objective of the reclamation and environmental protection plan would be to protect land, water and air during the construction and operational phases of mining, and on completion to re-establish, where practical, land uses of similar type and productivity as existed prior to mining. During the construction and operation phases of the project, much of the emphasis would be on control of drainage and seepage in order to minimize the impact to the downstream aquatic environment. However, it is important to ensure that any practical revegetation of disturbed areas is undertaken on a continuing basis. Control of dust and noise would also be carried out during the mine development and operation.

Over the long-term, the objective would be to establish a self-sustaining vegetative cover consistent with specified land uses once the mining operation in each area is completed. Rangeland, both native and improved, wildlife habitat and forestry constitute the present major land use in the mine area and are proposed as the end land use following mining. Drainage control sources would be stabilized such that operation of mine collection and treatment facilities would be no longer necessary.

A7.1 DISTURBANCES

Detailed estimates of the land areas disturbed by various mining activities at the end of 35 years of mining are presented in Table A7-1.

A7.2 ENVIRONMENTAL LOSSES

The reclamation and environmental plan is based in part on identification of potential environmental losses due to the proposed mine plan. The distribution of major environmental losses among land disturbances and major land uses (i.e. agriculture, forestry, wildlife habitat, and wetland habitat) is summarized in Table A7-2.

Three major reclamation and environmental protection priorities were identified: development of a safe pit abandonment scheme, effective revegetation of waste dumps and disturbed land areas, and drainage control during and after mining.

A7.3 EVALUATION OF SOILS, OVERBURDEN AND WASTE

Based on available data, soils, overburden and waste materials were arranged in order of preference for use as reclamation material (Table A7-3). The following recommendations are made with respect to selecting material for various reclamation purposes:

A7.3 EVALUATION OF SOILS, OVERBURDEN AND WASTE - (Cont'd)

1. Surface soils are the preferred reclamation material and should be used to cover final waste dumps and disturbed lands.

2. Seam waste, non-seam waste and slide debris are designated unsuitable for reclamation and should be buried.

3. Non-sodic glacial till, glaciofluvial and alluvial gravels are acceptable surface reclamation materials and are considered suitable for covering sodic waste. Glacial till is preferred to gravels because of its improved moisture holding capacity.

4. Non-sodic overburden with satisfactory water retention properties should be placed as a buffer material or layer between sodic waste and surface soils.

5. Low-grade coal and non-seam waste are considered unsuitable and would probably require incorporation of chemical amendments (gypsum, $CaCl_2$) and leaching if used as surface reclamation material.

6. Coaly waste and carbonaceous shale are strongly acid and would probably require incorporation of lime if used as surface reclamation material.

A7.4 WATER QUALITY

Estimates of leachate quantities and composition from stockpile and dump areas were developed through laboratory and field tests. Field studies suggest that pH, dissolved solids and sulphate concentrations would exceed present Pollution Control Board (PCB) level A objectives. Laboratory studies, based on a rate of release or probable "worst case" test indicate that arsenic, chromium, copper, iron, BOD

A7.4 WATER QUALITY - (Cont'd)

mercury, dissolved and suspended solids may be elevated when compared to present PCB level A objectives. Simulated samples of runoff water from dominant surface materials in disturbed areas were examined for sedimentation behaviour and it was found that chemical treatment followed by settling ponds may be required.

A7.5 DUST

Two high volume samplers monitored ambient air suspended particulate concentrations in the mine area during the 1977 bulk sample programme.

Dust problems developed in and near trench A during dry windy weather but were confined mainly to the trench. Carbonaceous shale and surface soil were particularly susceptible to dusting. Dust was controlled on access roads and trenches by water truck. Dust from dormant coal piles did not appear to be a problem. In all cases suspended particulate concentrations were less than either the 24-hour or annual present PCB level A objective of 150 and 60 mg/m^3 (geometric mean value).

A7.6 NOISE

Noise levels from typical excavating and blasting activity during the bulk sample programme were measured at varying distances from the trench.

The conclusion reached is that background noise levels outside the immediate pit area will not be significantly affected.

A7.7 SPONTANEOUS COMBUSTION

Some testwork was carried out during the 1977 bulk sample programme on coal, coaly waste and low-grade coal piles.

These products at Hat Creek are susceptible to spontaneous combustion under some conditions and precautionary measures such as crushing, packing and sealing are contained in the operating plan.

A7.8 PROPOSED RECLAMATION AND PROTECTION PLAN

Land areas which would be reclaimed throughout the mine area by the end of pre-production, year 15, year 35 and year 45 are summarized in Table A7-4. Reclamation would be carried out progressively and concurrently with mining as soon as any particular activity was completed. Disturbed areas left inactive for a number of years would be temporarily reclaimed.

Consideration was given to creating a lake in the pit void after mining operations are shut down; however, due to instability of the surrounding ground materials and the anticipated poor quality of pit water this proved undesirable. The open pit, at maximum safe mining depth would still contain a significant coal resource; filling up the pit with water (or alternatively with waste material) could adversely affect the economic viability of future extraction. Reclamation of the open pit would therefore comprise: regrading and revegetating the upper benches, seeding the remainder of the open pit, maintaining diversion ditches and canals to prevent undue water entry, and fencing the entire pit perimeter including adjoining areas of unstable materials to ensure the safety of the public.

In addition, open pit slopes could be expected to be subject to slides after the cessation of mining (Golder, 1978), the nature and extent of which cannot be predicted accurately with present data.

A7.8 PROPOSED RECLAMATION AND PROTECTION PLAN - (Cont'd)

Reclamation and environmental protection of the open pit, as described by CMJV, is based on minimizing water entry into the pit by maintaining ditches and diversions on a permanent basis.

Local surface soils are considered to be the most suitable materials available for reclamation in order to achieve desired end land use and productivity. These soils, varying in depth from 150 to 450 mm (6 to 18 in) should, where practical, be conserved from all disturbed areas and reused in 150 to 300 mm (6 to 12 in) layers.

On areas identified as having serious sodic problems, such as waste dumps and low-grade coal stockpiles, a sufficient depth of non-sodic overburden should be applied as "buffer material" to form a plant root zone between surface soils and sodic wastes. Based on limited data, a buffer depth of 1 m has been chosen for planning; on-site research during mining may suggest an increase or decrease in depth of buffer material. Mine planning is presently allowing for excavation and transportation of 10 MBCM of non-sodic glacial till and alluvial sands and gravels, some of which would be used in construction of waste dump embankments, and the remainder stockpiled on or near waste dumps for later use in reclamation.

Productivity of revegetated land is dependent on the quality and depth of surface reclamation material. Increasing depth would improve moisture and nutrient storage for revegetation. Current plans are to cover non-sodic waste and overburden with 150 to 300 mm (6 to 12 in) depth of surface soil.

Commercially available range grasses and legumes are planned to be seeded initially to stabilize non-vegetated areas and would be fertilized and managed until vegetation is self-sustaining. The determination of whether vegetation is self-sustaining at a given point in time would be based on change in vegetation cover, species composition, biomass, nutrient content, and seed yield and viability.

Although the diversity of present native vegetation and soils cannot be duplicated exactly, native plant species should eventually invade the perimeter of revegetated areas and over the long-term initiate succession similar to that on adjacent lands.

Selected areas for wildlife habitat would be planted with native trees, shrubs and forbs. Ongoing studies of the food, cover, nesting, watering, breeding and migratory habits of local fauna should provide the basis for establishment of wildlife habitat.

Wetland and riparian habitat cover a relatively small area 190 ha (470 ac), but are particularly important for waterfowl and wildlife use. Where possible, these areas would be conserved.

The perimeters of Hat and Finney creek diversions, interceptor ditches, reservoirs and water treatment lagoons would be revegetated as soon as possible following construction. An attempt at rehabilitating and re-establishing riparian habitat would be made based on conservation and reuse of fertile alluvial soils and the propagation and replanting of dominant, characteristic native plant species.

Replacement of about 995 ha (2460 ac) of grazing land and wildlife habitat on waste dump surfaces would be attempted. Assuming the water entry to an abandoned open pit is minimized, it is estimated that the water level should not reach an outflow level.

Reclamation of the open pit could eventually provide wildlife habitat. Reclamation efforts adjacent to reservoirs, lagoons, ditches

A7.8 PROPOSED RECLAMATION AND PROTECTION PLAN - (Cont'd)

and diversions would provide about 50 ha (125 ac) of riparian habitat. Clearwater reservoirs would be retained after mining to aid in regulation of downstream flow, flood control, and provide additional water for agriculture and fisheries. Reclamation of stockpiles, service yards, and transportation corridors (approximately 200 ha, 495 ac), would be devoted to forest land, wildlife habitat, and open range depending on specific location and elevation. Approximately 80 ha (200 ac) of land could not be reclaimed. This comprises those portions of roads 34 ha, (85 ac), lagoons 8.5 ha, (21 ac), reservoirs 9 ha, (22 ac) ditches 20.5 ha, (50 ac) and diversion canals 7.5 ha, (20 ac) remaining after mining and on which no reclamation would be carried out (i.e. water, active road surfaces, etc.).

A7.9 COSTS

The estimated capital and operating costs of reclamation and pollution control amount to $48.1 million. These estimates include the capital cost of buildings, field equipment and pickups, laboratory, greenhouses, seed and plant stock. Operating costs include staff, maintenance of greenhouses and storage buildings, stripping and stockpiling of surface soils, surface regrading, placement of buffer material and growth media, revegetation and subsequent maintenance.

Reclamation costs span a 6-year preproduction period, 35 years production and a 10-year post-production period. These charges include costs for activities associated with environmental protection which include sampling, testing and laboratory analysis involved in effluent and air quality control, together with soils analysis and testing required for the revegetation programme.

TABLE A7-1

Land Area of Disturbance (hectares)
(cumulative)
Hat Creek Project Mining Report 1978

Disturbance		End of Pre-Production	Year 15	Year 35
Open Pit		105	506	606
Waste Dumps				
Houth Meadows		165	445	610
Medicine Creek		*	*	385
	sub-total	165	445	995
Stockpiles				
Low grade coal		*	17.2	17.2
Coal		26.4	26.4	26.4
Topsoil		4.3	9.1	13.6
	sub-total	30.7	52.7	57.2
Service Yards		107	107	107
Roads				
Pit perimeter		22.5	39.2	47.3
Main access		3.0	3.0	3.0
	sub-total	25.5	42.2	50.3
Conveyor Corridors				
Thermal Plant		14.0	14.0	14.0
Medicine Creek		*	6.0	6.0
	sub-total	14.0	20.0	20.0
Water Treatment Lagoons				
Main		9.0	9.0	9.0
Medicine Creek		*	2.0	2.0
	sub-total	9.0	11.0	11.0
Clearwater Reservoirs				
Headworks (upper)		6.1	6.1	6.1
Pit rim (lower)		8.8	8.8	8.8
	sub-total	14.9	14.9	14.9
Ditches		14.0	15.8	27.0
Stream Diversions				
Hat Creek		33.6	33.6	33.6
Finney Creek		8.9	8.9	8.9
	sub-total	42.5	42.5	42.5
	GRAND TOTAL	528	1257	1931

* not constructed

TABLE A7-2

Environmental Losses (hectares)

Hat Creek Project Mining Report 1978

Losses	Open Pit	WASTE DUMPS		Service Yards	Stock-piles	Roads	CONVEYORS		DIVERSIONS		Reser-voirs	Lagoons	Ditches	Total
		Houth Meadows	Medicine Creek				Thermal Plant	Medicine Creek	Hat Creek	Finney Creek				
Agriculture														
A.L.R.	590	215	293	107	53	50	7	6	34	9	15	11	18	1408
Class 3	174	61	0	77	50	17	7	2	32	0	6	0	2	428
Class 5	126	93	0	0	0	2	0	0	0	8	9	9	1	248
Class G-2	306	344	134	0	4	30	6	0	0	1	0	0	11	836
Class G-3	0	112	246	0	0	0	1	4	0	0	0	0	10	373
Class G-4	0	0	5	30	3	1	0	0	2	0	0	2	3	46
Total	606	610	385	107	57	50	14	6	34	9	15	11	27	1931
Forestry														
Open Range	423	188	125	0	17	3	4	2	3	9	15	11	5	805
Non-Productive	0	12	0	0	0	0	0	0	0	0	0	0	0	12
Poor	183	382	260	107	40	47	10	4	31	0	0	0	22	1086
Medium	0	26	0	0	0	0	0	0	0	0	0	0	0	26
Good	0	2	0	0	0	0	0	0	0	0	0	0	0	2
Total	606	610	385	107	57	50	14	6	34	9	15	11	27	1931
Wildlife Habitat														
Douglas fir-Pinegrass	60	310	131	14	40	20	4	3	12	0	0	0	17	611
Sagebrush Grass-land	363	157	0	0	10	2	0	0	6	0	4	4	2	548
Ponderosa pine-Douglas fir-Grassland	143	116	133	50	4	27	5	2	16	1	0	2	6	505
Mid-elevation Grassland	0	0	104	0	0	0	5	1	0	0	0	0	1	111
Low-elevation Grassland	2	0	0	6	0	1	0	0	0	8	6	0	1	24
Riparian	38	0	0	0	3	0	0	0	0	0	5	5	0	51
Aspen	0	19	17	0	0	0	0	0	0	0	0	0	0	36
Bog	0	6	0	0	0	0	0	0	0	0	0	0	0	6
Rock	0	2	0	37	0	0	0	0	0	0	0	0	0	39
Total	606	610	385	107	57	50	14	6	34	9	15	11	27	1931
Heritage Sites														
* Surficial Sites	158	72	36	0	0	0	0	0	12	0	0	0	0	278
Estimated Site Area	7	9	0.6	0	0	0	0	0	0.5	0	0	0	0	17.1
Wetland Habitat														
Riparian Zone	107	0	0	0	3	0	0	0	0	0	5	5	0	120
Ephemeral Wetland Zone	0	65	0	0	0	0	0	0	0	0	0	0	0	65
* Ephemeral Ponds	2	2	0	0	0	0	0	0	0	4	0	0	0	8
* Intermittent Ponds	0	0	0	0	0	0	0	0	0	1	0	0	0	1
* Semi-Permanent Ponds	7	1	0	0	0	0	0	0	0	2	0	0	0	10
* Permanent Pond (Edge Vegetation)	0	7	0	0	0	0	0	0	0	1	0	0	0	8
* Permanent Pond (No Edge Vegetation)	2	1	0	0	0	0	0	0	0	2	0	0	0	5
Watercourses														
** Hat Creek	5	0	0	0										
** Finney Creek	3.2	0	0	0										
** Medicine Creek	0	0	5	0										
** Ephemeral Creeks (Houth)	0	2.5	0	0										
Lakes														
Aleece Lake	15													

* number of sites or ponds
** channel length in kilometres

Sources: Calculated from maps obtained from CBRC (1978) Agriculture, Reid-Collins (1978) Forestry, Tera (1978) Wildland and Wetland Habitat, Pokotylo and Beirne (1978) Heritage Resources

TABLE A7-3

Suitability of Soil, and Waste for Reclamation
Hat Creek Project Mining Report 1978

Order of Preference	Reclamation Material	Plant Growth Limiting Factors	Susceptibility to Erosion	Susceptibility to Dusting	Susceptibility to Leaching
1	Surface soils	low levels of nitrogen and phosphorous	moderate	high	moderate
2	Glacial till	low levels organic matter, N, and P; depressions may be saline	moderate	moderate	low
3	Glaciofluvial gravels	low levels of O.M., N, P, and possibly K; soil moisture deficiency	low	low	high
4	Alluvial gravels	low levels of O.M., N, P, and possibly K; soil moisture deficiency	low	low	high
5	Baked clay	slightly saline; coarse texture; low O.M., N, and P content	low	low	high
6	Slide debris	moderately saline; elevated B levels; strongly saline; low O.M., N, and P	moderate	moderate	low
7	Bentonitic clay	strongly saline; fine texture; low O.M., N, and P content	high	low	low
8	Coaly waste	low pH, slightly saline; elevated B; low O.M., N, and P content; dark colour resulting in increased surface temperature	high	low	low
9	Carbonaceous shale	low pH, slightly saline; elevated B; low O.M., N, and P content; dark colour resulting in increased surface temperature	high	high	low
10	Low-grade coal	highly sodic; low O.M., N, and P content	moderate	moderate	low
11	Gritstone	slightly sodic; slightly saline; low O.M., N, and P content	moderate	moderate	low
12	Waste rock	highly sodic; low O.M., N, and P content	moderate	moderate	low

Note: Materials 1-4 - suitable.

Materials 8 and 9 require addition of lime for use as surface reclamation materials.

Materials 5, 6, 7, 10, 11, and 12 require addition of chemical and organic amendments and leaching for use as surface reclamation materials.

O.M. - refers to organic matter

Note: Materials were rated for susceptibility to erosion, dusting, and leaching based on texture.

Susceptibility to rill erosion and dusting increases with finer texture.

Susceptibility to leaching increases with coarser texture.

A surface crust tends to form with increased sodium content which reduces dusting and leaching; but increases runoff.

Sources: Acres (1978) and B.C. Research (1975)

TABLE A7-4

Estimated Area of Reclamation (hectares)
(cumulative)
Hat Creek Project Mining Report 1978

Location	End of Pre-production	Years 15	Years 35	Years 45
Open Pit - top 3 berms	0.0	0.0	0.0	115
lower berms				491
				606
Waste Dumps				
Houth Meadows	2.0	38.0	380.0	610
Medicine Creek	*	*	212.0	385
Sub-total	2.0	38.0	592.0	995
Stockpiles				
Low grade coal	*	8.6	17.2	17.2
Coal	0.0	0.0	0.0	26.4
Topsoil	4.3	9.1	13.6	13.6
Sub-total	4.3	17.7	30.8	57.2
Service Yards	6.0	6.0	6.0	106.8
Roads				
Pit perimeter	5.0	10.0	15.0	15.0
Main access	1.0	1.0	1.0	1.0
Sub-total	6.0	11.0	16.0	16.0
Conveyor Corridors				
Thermal Plant	7.0	7.0	7.0	14.0
Medicine Creek	*	3.0	3.0	6.0
Sub-total	7.0	10.0	10.0	20.0
Water Treatment Lagoons				
Main	2.0	2.0	2.0	2.0
Medicine Creek	*	0.5	0.5	0.5
Sub-total	2.0	2.5	2.5	2.5
Clearwater Reservoirs				
Headworks (upper)	2.0	2.0	2.0	2.0
Pit rim (lower)	4.0	4.0	4.0	4.0
Sub-total	6.0	6.0	6.0	6.0
Ditches	3.0	3.5	6.5	6.5
Stream Diversions				
Hat Creek	27.0	27.0	27.0	27.0
Finney Creek	8.0	8.0	8.0	8.0
Sub-total	35.0	35.0	35.0	35.0
GRAND TOTAL	71.3	129.7	704.8	1851

* not constructed

SECTION A8.0 - COST ESTIMATES AND FINANCIAL ANALYSIS

The major objective of this appendix is to develop an estimated cost of delivering datum coal to the powerplant, as one component of the cost of producing electrical energy at Hat Creek.

The capital and operating expenditures required for the shovel/truck/conveyor system described in this appendix were estimated in accordance with the following criteria:

1. Use October 1977 Canadian dollars throughout.

2. Develop estimated annual expenditures for the pre-production period (years -6 to -1 inclusive), 35-year production period and 10 years of post-production reclamation activity.

3. Specific exclusions are B.C. Hydro corporate overhead, land purchase or lease costs, mineral rights purchase or lease costs, housing and other infrastructure costs.

A8.1 ESTIMATING CRITERIA

The basis on which estimating was carried out is listed below. These estimates were developed utilizing a variety of information sources.

(a) Capital

1. Major equipment costs based on manufacturers' budget estimates.

2. Equipment service lives estimated from the experience of operating mines.

A8.1 ESTIMATING CRITERIA - (Cont'd)

3. U.S./Canadian exchange rate $1.08 used, where applicable.

4. Labour rates and fringes taken from B.C. Construction Agreements in effect at September 1977.

5. Loaded rates were developed on the basis of 37.5-hour week plus an allowance of 20 percent for random overtime.

6. Buildings estimated using industry rates.

(b) Operating

1. Staff salaries taken from published B.C. Mining Industry survey.

2. Hourly wages and benefits from September 1977, B.C. Mining Agreements, with appropriate allowances for absenteeism, shift work, overtime, etc.

3. Equipment productivities, availabilities, materials and supply rates, mainly from the experience of operating mines.

4. Mine operating schedule - continuous, 365 days/year.

5. Hours per shift - 8.

(c) Summary

The total estimated capital and operating expenditures over the life of the project are summarized according to major cost centres in Table A8-1.

The cash flows for the major cost centres are provided in Tables A5-4, A8-2 and A8-3.

A8.1 ESTIMATING CRITERIA - (Cont'd)

Detailed cost estimates were generated for the shovel/truck/conveyor system. Using the basic estimating data from this system, "order-of-magnitude" costs were developed for the combined system. A comparison of total expenditures for the project life in 1977 Canadian dollars is shown in Table A8-4.

Table A8-5 shows the annual operating and capital costs and the cash flow of operating plus capital cost for the life of the project. Annual operating costs per GJ and per tonne delivered to the powerplant are also provided.

A8.2 MINE OPERATING COSTS

The direct operating costs of the mine which are shown in detail in Table A8-2, were distributed into the following cost centres:

1. Overburden - includes cost of drilling, blasting, loading, dozing and hauling to dump pocket of overburden materials.

2. Waste - includes cost of loading, dozing and hauling to dump pocket of waste materials.

3. Conveying waste - includes cost of conveying and dump handling of waste materials.

4. Coal - includes cost of drilling, blasting, loading, dozing and hauling to dump pocket of coal and low-grade coal plus the cost of hauling, spreading and compacting of low-grade coal in the stockpile area.

5. Coal conveying, stacking and blending - includes cost of conveying coal and low-grade coal plus costs of dump pockets, crushing and sampling plants and stacking and blending facilities.

A8.2 MINE OPERATING COSTS - (Cont'd)

6. Pit maintenance - includes the cost of pit drainage in-put electrical maintenance, snow control, road construction and maintenance and trailing cable handling.

7. Dumps - includes the cost of the 32 t (35 ton) trucks, front-end loaders, dozers and graders etc. used in the construction and maintenance of the dumps including conveyor causeways and relocating conveyors.

8. Mine service vehicles and equipment - includes the costs of operating service vehicles and equipment such as cranes, service trucks, forklifts, ambulance and fire truck.

9. Maintenance electrical services - includes the cost of all out-of-pit electrical maintenance.

10. Fuel stations - include the cost of operating the main fuel storage station and the in-pit fuel station together with the cost of operating the fuel and lube trucks.

11. Shovel supplies - includes the cost of shovel teeth and adaptors.

12. Water, sewers and drainage - includes the cost of maintaining the area drainage ditch system and for operating and maintaining the water treatment plant, water distribution system, sewage disposal system and treatment lagoons.

The costs were developed using the production schedule (Plate A4-13), equipment productivities (see Section A5.0), utilization (Plate A5-1), M&S rates (Table A8-6), hourly operating labour rates (Table A8-7), salaries for mine staff (Table A8-8) and the mine operating parameters (Table A8-9). For example, in a typical year, the following costs for the drilling and blasting, loading, dozing and hauling of coal and low-grade coal to the dump pockets plus trucking, spreading and compacting of low-grade coal would be developed in the following way:

Drilling and Blasting

Auger Drill

Total volume for blasting	=	3.213 MBCM
Volume per hole (7.5 m x 7.5 m x 15 m deep)	=	840 BCM
No. of holes	=	3823
Scheduled hours (auger drill)	=	4016
Effective utilization (auger drill))	=	60%
Operating hours	=	2731
Operating labour rate/hour	=	\$13.40
M&S rate/hour	=	\$40.00
Explosives cost including detonators etc.	=	\$24.00/45 kg
Developed cost per hole	=	\$91.00

TOTAL COST DRILLING AND BLASTING

-	Operating Labour	= (\$13.40/hour) x (4016 hours)	=	\$ 54,000
-	M&S	= \$40.00 x 2731	=	\$110,000
-	Blast cost	= \$91.00 x 3823	=	\$348,000
		TOTAL		\$512,000

Loading

Shovel (16.8 m^3, 22 Yd^3)

Total coal to be mined	=	7.23 MBCM
Capacity/operating hour	=	827 BCM
Scheduled hours/year	=	8760
Effective utilization	=	68%
Operating hours/year	=	6000

A8.2 <u>MINE OPERATING COSTS</u> - (Cont'd)

Shovel operator I hourly rate	=	\$13.95
Shovel operator II hourly rate	=	\$13.05
Shovel M&S rate/hour	=	\$82.00

Unit cost of coal loaded comprises:

1. Operating labour = $\frac{(\$13.95 + \$13.05)/\text{hour}}{827 \text{ BCM/hour}} \times \frac{8760 \text{ hours}}{6000 \text{ hours}}$ = \$0.04767/BCM

2. M&S (based on operating hours) = $\frac{\$82/\text{hour}}{827 \text{ BCM}}$ = \$0.09915/BCM

Total loading cost for coal = \$0.14682/BCM

$\frac{\$0.1468}{\text{BCM}}$ = 7.23 MBCM = \$1,062,000

<u>Front End Loader</u> (11.5 m^3 bucket)

Total coal to be mined	360,000 BCM
Capacity per operating hour	450 BCM
Scheduled hours/year	1460
Effective utilization	55%
Operating hours/year	803
Loader operator hourly rate	\$13.40
M&S rate/hour	\$60.00

1. Op. Labour = $\frac{\$13.40}{450} \times \frac{1460}{803}$ = \$0.05414/BCM

2. M&S = $\frac{\$60.00}{450}$ = \$0.133/BCM

Total Front End Loading Cost = \$0.18747/BCM

= \$0.1875 x 360,000 = \$68,000

A8.2 MINE OPERATING COSTS - (Cont'd)

Dozing

Scheduled hours	=	1460
Effective utilization	=	57%
Operating hours	=	832
M&S rate/hour	=	$33.50

TOTAL COST FOR DOZING IN COAL

1.	Operating Labour = $13.40 x 1460 hours	=	$20,000
2.	M&S cost = $33.50 x 832	=	$28,000
	TOTAL	=	$48,000

Hauling

Total coal to be trucked	=	7.59 MBCM
Truck hours required	=	22,137
(developed from haul study based on 60 minute hours)		
Truck effective utilization	=	57%
Truck scheduled hours	=	38,837
Operator rate/hour	=	$13.05
M&S rate/hour	=	$52.00

TOTAL COST FOR HAULING COAL

1.	Operating labour = 38,837 hrs x $13.05/hr	=	$ 508,000
2.	M&S = 22,137 x $52.00/hr x $\frac{60}{50}$ (adjusting to 50 minute hour)	=	$1,380,000
	TOTAL	=	$1,888,000

HAULING COST FOR COAL	=	$0.249/BCM
	=	$0.386/t

A8.2 MINE OPERATING COSTS - (Cont'd)

The costs of the low-grade coal handling by the support equipment to reclaim by front-end loader, truck, spread and compact were similarly developed. The total cost amounted to $733,000.

		Total	Unit Cost $/t
TOTAL OPERATING COSTS OF COAL AND LOW GRADE COAL			
Drilling and blasting	=	$ 512,000	0.07
Loading - shovel	=	$1,062,000	0.14
- front end loader	=	$ 68,000	0.01
Dozing	=	$ 48,000	0.01
Hauling	=	$1,888,000	0.25
Low-grade coal handling	=	$ 733,000	0.09
TOTAL		$4,311,000	0.57

A8.3 FINANCIAL ANALYSIS

(a) General Approach

The capital and operating cash flows discussed in the previous sub-section were the basic inputs for the financial evaluation. These cash flows are schedules of annual capital and operating cash requirements for the mining and transportation of Hat Creek coal to the powerplant including expenditures for reclamation and environmental protection and royalties.

The objective of this analysis is to determine the price in 1978 dollars of Hat Creek coal delivered to the powerplant. At this price the mining venture must yield a rate of return equal to B.C. Hydro's cost of capital. The price of Hat Creek coal delivered to the powerplant could then be compared with any alternative fuel with an equivalent energy content.

A8.3 FINANCIAL ANALYSIS - (Cont'd)

(b) Parameters

The financial parameters used in this analysis are as follows:

1. Base date for economic calculations is October 1978. The capital and operating cash flow in 1978 dollars were derived by applying an inflation rate of 7.75 percent to the October 1977 costs reported by CMJV.

2. Inflation rates:

   The following composite inflation rates based on weighting of 35 percent labour and 65 percent material were used in the analysis:

	Fiscal Year	Rate %
	1978/1979	7.75
	1979/1980	7.50
	1980/1981	7.25
	1981/1982	6.25
	1982/1983	5.75
after	1982/1983	5.75

3. Plant operating date:

   It is assumed that the first unit commences operation in the fiscal year 1986/1987.

4. Debt: Equity ratio:

   The financial structure is 100 percent debt.

A8.3 FINANCIAL ANALYSIS - (Cont'd)

5. Rate of return:

   A rate of return of 10 percent is required to cover interest only. No operating profit assumed.

6. Income tax:

   No income tax is paid by B.C. Hydro.

7. Provincial royalty:

   Assumed to be $.738/t ($.75/long ton).

   New government regulations have recently been announced which will reduce these royalty payments to 3 1/2 percent of the mine head value which is equivalent to about $.25/t ($.26/long ton). The effect of this change will be incorporated into the mine costs at a later date.

8. Corporate overhead:

   Corporate overhead has been calculated at 5 percent of the total capital cost incurred.

(c) Analysis

Utilizing a financial computer model the present values of the cash outflows and inflows associated with the project were equated. The cash outflows being the capital and operating cash requirements by year for the mining operations, royalty payments, transportation of coal to the powerplant and reclamation and environmental protection. The cash inflows were determined from the schedule of coal to be supplied to the powerplant and the price of the delivered coal.

A8.3 FINANCIAL ANALYSIS - (Cont'd)

This analysis incorporated the inflation rates stipulated above, a discount rate of 10 percent and a time horizon of 45 years since reclamation activities continue for a 10-year period after mining ceases.

(d) Conclusion

Based on the above financial parameters the price of Hat Creek coal delivered to the powerplant is \$0.70/GJ (\$0.74/MBtu) in 1978 dollars, which is equivalent to \$8.14/t (\$7.40/ton) of coal with an average heat value of 17 MJ/kg (7327 Btu/lb), dry basis. If the cost of power for the mining operations is excluded the price of coal is reduced to \$0.679/GJ (\$0.716/MBtu).

In the evaluation of the project, financial parameters could be varied to reflect different corporate policies. As an example, it may be decided that coal prices should be increased by only 3 percent per annum irrespective of the given inflation factors. Such a policy would result in a selling price of coal in 1978 dollars in the order of \$1.29/GJ (\$1.36/MBtu) or \$15.00/t (\$13.64/ton).

TABLE A8-1

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

($000's October 1977)

CMJV Hat Creek Project Mining Report 1978

Cost Centre	Amount	Unit Cost/ tonne of coal Delivered ($)	Unit Cost/ GJ(MBtu) ($)	
CAPITAL COSTS				
Engineering and Construction	45,831	0.131	0.010	(0.011)
Mine Property Development	79,379	0.227	0.018	(0.019)
Buildings and Structures	15,581	0.045	0.003	(0.004)
Pit Services	2,308	0.007	0.001	(0.001)
Mining Equipment	178,146	0.510	0.041	(0.042)
Coal Conveying, Crushing and Blending Equipment	45,034	0.129	0.010	(0.011)
Waste Disposal Equipment	81,312	0.232	0.018	(0.019)
Reclamation and Environmental Protection	1,879	0.005	0.001	(0.001)
Contingency	59,509	0.170	0.013	(0.014)
TOTAL...	508,979	1.456	0.115	(0.122)
OPERATING COSTS				
Direct Mining				
Mining Waste Above Bedrock	198,583	0.568	0.045	(0.047)
Mining Bedrock Waste	118,808	0.340	0.027	(0.028)
Conveying Waste	71,768	0.205	0.016	(0.017)
Mining Coal	140,248	0.401	0.031	(0.033)
Conveying Coal	29,475	0.084	0.006	(0.007)
Coal Stockpiling and Blending	30,613	0.088	0.006	(0.007)
Pit Maintenance and Roads	197,483	0.565	0.045	(0.047)
Operation of Mobile Dump Equipment	62,275	0.178	0.014	(0.015)
Operation of Mine Service Vehicles	58,559	0.168	0.013	(0.014)
Maintenance of Electrical Services	46,056	0.132	0.010	(0.011)
Operation of Fueling Stations	32,470	0.093	0.007	(0.008)
Reclamation and Environmental Protection	46,267	0.132	0.010	(0.011)
General Mine Expense	157,092	0.449	0.035	(0.037)
Local Overheads	284,229	0.813	0.064	(0.067)
Power	86,784	0.248	0.019	(0.020)
Royalties	255,130	0.730	0.057	(0.060)
Contingency	261,774	0.750	0.059	(0.062)
TOTAL	2,077,614	5.944	0.465	(0.491)

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SCHEDULE OF OPERATING COSTS

ACCOUNT 100 - DIRECT MINING
ACCOUNT 200 - RECLAMATION AND ENVIRONMENTAL PROTECTION
ACCOUNT 300 - GENERAL MINE EXPENSE

TABLE A8-2

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

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TABLE A8-2

ACCOUNT CODE	DESCRIPTION	PRE-PRODUCTION PERIOD						PRODUCTION PERIOD											
		-6	-5	-4	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12
100	DIRECT MINING																		
101	Mining Waste Above Bedrock			34	6697	6833	5244	7367	7505	6689	6058	5920	6595	6879	6879	6879	6578	5567	5557
102	Mining Bedrock Waste					745	1089	1515	1483	1633	1818	2524	2585	2395	2395	2395	2621	3336	3336
103	Conveying Waste						1203	1961	1617	1566	2070	1934	1759	1744	2566	2058	2058	2046	2006
104	Mining Coal						411	1310	2163	3164	4346	5346	5089	4398	4398	4398	4502	5109	5119
105	Conveying Coal						392	539	602	686	825	899	899	899	899	899	899	899	899
106	Coal Stockpiling and Blending						454	642	712	805	875	875	875	875	875	875	875	875	875
107	Pit Maintenance		1928	838	575	2677	3288	4397	4537	4757	5092	5172	5067	5013	5007	5142	6045	6231	6357
108	Operation of Mobile Equipment on Dumps					2083	1311	1474	919	694	803	839	876	876	876	876	876	949	949
109	Operation of Mine Service Vehicles				474	1087	1348	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590
110	Maintenance of Electrical Services						696	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296
111	Operation of Fueling Stations					485	485	900	900	900	900	900	900	900	900	900	900	900	900
	TOTAL ACCOUNT 100		1928	872	7746	13910	15921	22991	23324	23780	25673	27295	27531	26865	27681	27308	28240	28798	28884
200	RECAMATION AND ENVIRONMENTAL PROTECTION																		
201	Staff Salaries		71	212	212	212	212	276	276	276	276	277	337	337	336	336	336	338	337
202	Vehicles and Buildings		41	82	104	124	132	132	132	132	132	132	132	132	132	133	145	145	145
203	Earthworks			701	652	695	257	190	186	252	182	197	197	288	197	199	290	199	277
204	Nursery Operation				4	3	3	20	20	20	20	20	20	20	20	20	20	20	20
205	Revegetation			11	11	11	11	4	5	5	5	5	5	5	5	5	5	5	5
	TOTAL ACCOUNT 200		112	1006	983	1045	615	622	619	685	615	631	691	782	690	693	796	707	784
300	GENERAL MINE EXPENSE																		
301	Mine Supervision					662	951	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191
302	Mine Engineering					297	560	634	634	634	634	634	634	634	634	634	634	634	634
303	Mine Geology					64	240	425	425	425	425	425	425	425	425	425	425	425	425
304	Mine Communications					83	206	299	299	299	299	299	299	299	299	299	299	299	299
305	Mine Transportation					108	161	189	189	189	189	189	189	189	189	189	189	189	189
306	Mine Training					475	493	492	492	492	492	492	492	492	492	492	492	492	492
307	Close-Spaced In-Pit Drilling						198	198	198	198	198	198	198	198	198	198	198	198	198
308	Safety					47	47	47	47	47	47	47	47	47	47	47	47	47	47
309	Coal Quality Control						428	428	428	428	428	428	428	428	428	428	428	428	428
310	Light Vehicle Operation					232	395	424	424	424	424	424	424	424	424	424	424	424	424
	TOTAL ACCOUNT 300					1968	3679	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD											
		13	14	15	16	17	18	19	20	21	22	23	24
100	DIRECT MINING												
101	Mining Waste Above Bedrock	5557	5557	5567	6607	6654	6654	6654	6653	6484	6215	5839	5839
102	Mining Bedrock Waste	3336	3336	3346	4249	4536	4536	4534	4534	4431	4761	5035	5035
103	Conveying Waste	1797	2689	2009	2375	2481	2041	3302	2343	2343	2343	2343	2343
104	Mining Coal	5101	5101	5111	4553	4130	4130	4130	4130	3690	3835	3876	3876
105	Conveying Coal	899	899	899	899	899	899	899	959	1003	1003	1003	1003
106	Coal Stockpiling and Blending	875	875	875	875	875	875	875	875	875	875	875	875
107	Pit Maintenance	6326	6420	5908	5922	5791	5787	5787	5789	5890	6043	6036	6036
108	Operation of Mobile Equipment on Dumps	949	1097	1353	9228	7675	1544	1544	1544	1500	1500	1500	1500
109	Operation of Mine Service Vehicles	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590
110	Maintenance of Electrical Services	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296
111	Operation of Fueling Stations	900	900	900	900	900	900	900	900	900	900	900	900
	TOTAL ACCOUNT 100	28626	29760	28854	38494	36827	30252	31511	30613	30002	30361	30293	30293
200	RECAMATION AND ENVIRONMENTAL PROTECTION												
201	Staff Salaries	337	337	337	336	336	336	336	336	374	374	374	374
202	Vehicles and Buildings	145	145	145	145	145	145	145	145	145	145	145	145
203	Earthworks	199	199	362	486	486	486	486	486	486	664	764	764
204	Nursery Operation	20	20	32	32	32	32	32	32	32	32	32	32
205	Revegetation	5	5	5	20	20	20	20	20	20	20	20	20
	TOTAL ACCOUNT 200	706	706	881	1019	1019	1019	1019	1019	1057	1235	1335	1335
300	GENERAL MINE EXPENSE												
301	Mine Supervision	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191
302	Mine Engineering	634	634	634	634	634	634	634	634	634	634	634	634
303	Mine Geology	425	425	425	425	425	425	425	425	425	425	425	425
304	Mine Communications	299	299	299	299	299	299	299	299	299	299	299	299
305	Mine Transportation	189	189	189	189	189	189	189	189	189	189	189	189
306	Mine Training	492	492	492	492	492	492	492	492	492	492	492	492
307	Close-Spaced In-Pit Drilling	198	198	198	198	198	198	198	198	198	198	198	198
308	Safety	47	47	47	47	47	47	47	47	47	47	47	47
309	Coal Quality Control	428	428	428	428	428	428	428	428	428	428	428	428
310	Light Vehicle Operation	424	424	424	424	424	424	424	424	424	424	424	424
	TOTAL ACCOUNT 300	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD											POST PRODUCTION PERIOD										
		25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41	42	43	44	45	TOTAL
100	DIRECT MINING																						
101	Mining Waste Above Bedrock	5843	3497	2142	2008	2026	2026	1896	1898	1898	1894	1894											198583
102	Mining Bedrock Waste	5035	3146	3220	3239	3240	3235	3234	3230	3230	3230	3230											118808
103	Conveying Waste	2860	1156	1141	1590	1135	2011	1177	1177	1177	1152	1137											68710
104	Mining Coal	3876	3358	3513	3513	3508	3498	3498	3517	3517	3517	3517											140248
105	Conveying Coal	1003	1003	1003	1003	1003	1003	1003	1003	1003	1003	1003											32533
106	Coal Stockpiling and Blending	875	875	875	875	875	875	875	875	875	875	875											30613
107	Pit Maintenance	6038	6131	4710	4682	4682	4672	4542	4542	4542	4542	4542											197483
108	Operation of Mobile Equipment on Dumps	1500	1500	1500	1500	1500	1500	1500	1588	1588	1588	1588											62275
109	Operation of Mine Service Vehicles	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590	1590											58559
110	Maintenance of Electrical Services	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296	1296											46056
111	Operation of Fueling Stations	900	900	900	900	900	900	900	900	900	900	900											32470
	TOTAL ACCOUNT 100	30816	24452	21890	22196	21755	22606	21599	21616	21616	21587	21572											986338
200	RECAMATION AND ENVIRONMENTAL PROTECTION																						
201	Staff Salaries	374	374	374	374	374	374	374	374	374	374	374	374	374	374	256	256	121	121	67	36	36	14973
202	Vehicles and Buildings	145	145	145	145	145	145	145	145	145	145	145	145	145	145	90	90	48	48	41	20	20	6234
203	Earthworks	675	642	642	642	642	735	670	643	661	684	714	1272	1272	937	654	4						22316
204	Nursery Operation	32	32	32	32	32	32	32	32	32	32	32	32	32	32								1058
205	Revegetation	20	20	20	20	20	20	20	20	20	20	20	146	146	146	146	146	146	146	146			1686
	TOTAL ACCOUNT 200	1246	1213	1213	1213	1213	1306	1241	1214	1232	1255	1285	1969	1969	1634	1146	496	315	315	254	56	56	46267
300	GENERAL MINE EXPENSE																						
301	Mine Supervision	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191	1191											43298
302	Mine Engineering	634	634	634	634	634	634	634	634	634	634	634											23047
303	Mine Geology	425	425	425	425	425	425	425	425	425	425	425											15179
304	Mine Communications	299	299	299	299	299	299	299	299	299	299	299											10754
305	Mine Transportation	189	189	189	189	189	189	189	189	189	189	189											6884
306	Mine Training	492	492	492	492	492	492	492	492	492	492	492											18188
307	Close-Spaced In-Pit Drilling	198	198	198	198	198	198	198	198	198	198	198											7128
308	Safety	47	47	47	47	47	47	47	47	47	47	47											1739
309	Coal Quality Control	428	428	428	428	428	428	428	428	428	428	428											15408
310	Light Vehicle Operation	424	424	424	424	424	424	424	424	424	424	424											15467
	TOTAL ACCOUNT 300	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327											157092

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SCHEDULE OF OPERATING COSTS
SUMMARY CASH FLOW
TABLE A8 - 3

BRITISH COLUMBIA HYDRO AND POWER AUTHORITY
HAT CREEK PROJECT

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TABLE A8 - 3

ACCOUNT CODE	DESCRIPTION	PRE-PRODUCTION PERIOD -6	-5	-4	-3	-2	-1	PRODUCTION PERIOD 1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
100	DIRECT MINING		1928	872	7746	13910	15921	22991	23324	23780	25673	27295	27531	26865	27681	27308	28240	28798	28884	28626	29760	28854	38494	36827
200	RECLAMATION AND ENVIRONMENTAL PROTECTION		112	1006	983	1045	615	622	619	685	615	631	691	782	690	693	796	707	784	706	706	881	1019	1019
300	GENERAL MINE EXPENSE					1968	3679	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327
400	LOCAL OVERHEADS				412	6146	7585	7185	7269	7304	7330	7330	7395	7407	7547	7663	7663	7663	7663	7663	7663	7713	7763	7775
500	POWER					339	1627	2051	2142	2193	2356	2368	2636	2636	2636	2636	2636	2302	2516	2516	2646	2646	2648	2648
600	ROYALTIES						752	2248	3964	5986	7782	8249	8264	8293	8293	8293	8293	8322	8322	8322	8322	8322	7972	7796
700	CONTINGENCY		362	333	1623	4097	4937	6238	6312	6411	6739	7030	7094	6994	7148	7103	7270	7386	7399	7339	7540	7419	9165	8871
	TOTAL OPERATING COSTS		2402	2211	10764	27505	35116	45662	47957	50686	54822	57230	57938	57304	58322	58023	59225	59505	59895	59499	60964	60162	71388	69263

ACCOUNT CODE	DESCRIPTION	PRODUCTION PERIOD 18	19	20	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35
100	DIRECT MINING	30252	31511	30613	30002	30361	30293	30293	30816	24452	21890	22196	21755	22606	21599	21616	21616	21587	21572
200	RECLAMATION AND ENVIRONMENTAL PROTECTION	1019	1019	1019	1057	1235	1335	1335	1246	1213	1213	1213	1213	1306	1241	1214	1232	1255	1285
300	GENERAL MINE EXPENSE	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327	4327
400	LOCAL OVERHEADS	7775	7801	7801	7815	7815	7815	7931	7931	7931	7931	7943	7943	7943	7943	7943	7943	7943	7943
500	POWER	2648	2648	2643	2787	2787	2787	2787	2787	2257	2099	2047	2047	2047	2047	2047	2047	2047	2047
600	ROYALTIES	7796	7796	7796	7796	7599	7599	7599	7599	6490	6585	6585	6585	6585	6585	6585	6585	6585	6585
700	CONTINGENCY	7703	7931	7772	7673	7768	7774	7794	7871	6735	6280	6337	6258	6426	6236	6234	6237	6237	6239
	TOTAL OPERATING COSTS	61520	63033	61971	61457	61892	61930	62066	62577	53405	50325	50648	50128	51240	49978	49966	49987	49981	49998

ACCOUNT CODE	DESCRIPTION	POST PRODUCTION PERIOD 36	37	38	39	40	41	42	43	44	45	TOTAL
100	DIRECT MINING											986338
200	RECLAMATION AND ENVIRONMENTAL PROTECTION	1969	1969	1634	1146	496	315	315	254	56	56	46267
300	GENERAL MINE EXPENSE											157092
400	LOCAL OVERHEADS											284229
500	POWER											86784
600	ROYALTIES											255130
700	CONTINGENCY	350	350	290	204	88	56	56	45	10	10	261774
	TOTAL OPERATING COSTS	2319	2319	1924	1350	584	371	371	299	66	66	2077614

CANADIAN $ 000'S OCTOBER 1977

TABLE A8-4

Summary Comparison of
Shovel/Truck, Conveyor System and
Combined Bucket Wheel/Shovel/Truck/Conveyor System

Hat Creek Project Mining Report 1978

	Combined Bucket Wheel Excavator/ /Shovel/Truck/ /Conveyor ($ 000's)	Shovel/Truck Conveyor ($ 000's)
Total Operating Cost	$1,428,082	$1,441,209
Total Capital Cost	447,856	449,470
Total Power Cost	95,631	86,784
Total Royalty Cost	255,130	255,130
Total Contingency	424,886	309,814
	$2,651,585	$2,542,407

TABLE A8-5

Summary of Annual Costs

Canadian $ October 1977

Hat Creek Project Mining Report 1978

Year	COAL tonnes x 10^6	COAL MJ x 10^9	$000 Annual Operating Cost	$ Annual Operating Cost/tonne	$ Annual Operating Cost/ GJ	$000 Annual Capital Costs	$000 Total Annual Capital + Operating Cost	$000 Total Cumulative Operating + Capital Cost
-6						2,350	2,350	2,350
-5			2,402			6,057	8,459	10,809
-4			2,211			15,398	17,609	28,418
-3			10,764			65,473	76,237	104,655
-2			27,505			89,197	116,702	221,257
-1	1.03	13	35,116	34.09	2.701	48,060	83,176	304,533
1	3.08	39	45,662	14.82	1.171	12,231	57,893	362,426
2	5.43	69	47,957	8.83	0.695	13,235	61,192	423,618
3	8.20	106	50,686	6.18	0.478	5.989	56,675	480,293
4	10.66	136	54,822	5.14	0.403	9,946	64,768	545,061
5	11.30	144	57,230	5.06	0.397	16,968	74,198	619,259
6	11.32	144	57,938	5.12	0.402	3,884	61,822	681,081
7	11.36	145	57,304	5.04	0.395	6,826	64,130	745,211
8	11.36	145	58,322	5.13	0.402	9,303	67,625	812,836
9	11.36	145	58,023	5.11	0.400	12,958	70,981	883,817
10	11.36	145	59,225	5.21	0.408	7,464	66,689	950,506
11	11.40	145	59,505	5.22	0.410	9,034	68,539	1,019,045
12	11.40	145	59,895	5.25	0.413	2,625	62,520	1,081,565
13	11.40	145	59,499	5.21	0.410	14,766	74,265	1,155,830
14	11.40	145	60,964	5.34	0.420	7,740	68,704	1,224,534
15	11.40	145	60,162	5.27	0.415	7,150	67,312	1,291,846
16	10.92	139	71,388	6.53	0.514	9,315	80,703	1,372,549
17	10.68	135	69,263	6.48	0.513	5,078	74,341	1,446,890
18	10.68	135	61,520	5.76	0.456	12,159	73,679	1,520,569
19	10.68	135	63,033	5.90	0.467	21,399	84,432	1,605,001
20	10.68	135	61,971	5.80	0.459	22,926	84,897	1,689,898
21	10.68	135	61,457	5.75	0.455	2,505	63,962	1,753,860
22	10.41	135	61,892	5.94	0.458	2,938	64,830	1,818,690
23	10.41	135	61,930	5.95	0.459	9,173	71,103	1,889,793
24	10.41	135	62,066	5.96	0.460	4,502	66,568	1,956,361
25	10.40	135	62,577	6.01	0.464	5,394	67,971	2,024,332
26	8.89	115	53,405	6.00	0.464	5,037	58,442	2,082,774
27	9.02	115	50,325	5.58	0.438	6,787	57,112	2,139,886
28	9.02	115	50,648	5.62	0.440	5,020	55,668	2,195,554
29	9.02	115	50,128	5.56	0.436	11,371	61,499	2,257,053
30	9.02	115	51,240	5.68	0.446	3,801	55,041	2,312,094
31	9.02	115	49,978	5.54	0.435	7,461	57,439	2,369,533
32	9.02	115	49,966	5.54	0.434	4,486	54,452	2,423,985
33	9.02	115	49,987	5.54	0.435	685	50,672	2,474,657
34	9.02	115	49,981	5.54	0.435	1,232	51,213	2,525,870
35	9.02	115	49,998	5.54	0.435	693	50,691	2,576,561
36			2,319			98	2,417	2,578,978
37			2,319			29	2,348	2,581,326
38			1,924			24	1,948	2,583,274
39			1,350			98	1,448	2,584,722
40			584			46	630	2,585,352
41			371			10	381	2,585,733
42			371			34	405	2,586,138
43			299			2	301	2,586,439
44			66			21	87	2,586,526
45			66			1	67	2,586,593
TOTAL	349.49	4460.0	2,077,614	5.94	0.466	508,979	2,586,593	

TABLE A8-6

EQUIPMENT MAINTENANCE AND SERVICE RATES
AND
MAINTENANCE LABOUR RATES (1977 DOLLARS)

HAT CREEK PROJECT MINING REPORT 1978

Item	Equipment M & S ($/hr)	Maintenance Labour ($/hr)
Drills		
Auger - truck mounted	24.00	16.00
Air-trac c/w compressor	21.00	14.00
Shovels (rope)		
16.8 m^3	53.30	28.70
F.E. Loader		
5 m^3	24.00	16.00
7.6 m^3	27.00	19.50
11.5 m^3	33.00	27.00
Haulage Truck		
32 tonnes	13.20	8.80
109 tonnes w/coal box	33.80	18.20
136 tonnes w/rock box	40.30	21.70
Scraper		
24 m^3 (tandem)	30.25	24.75
Dozer (track)		
CAT D6	9.75	5.25
CAT D8	17.20	9.30
CAT D9	20.50	11.00
CAT D9 w/ripper	21.75	11.75

(Continued)

TABLE A8-6 (Continued)

Item	Equipment M & S ($/hour)	Maintenance Labour ($/hour)
Welders (portable)		
600 A diesel	2.80	0.70
600 A electric	2.40	0.60
Miscellaneous Equipment		
Backhoe (1 m^3)	6.50	3.50
Compressor (17 m^3/min)	7.50	2.50
Compressor (30 m^3/min)	10.50	4.50
Steam Cleaner	6.00	4.00
Lighting plant (3 kW)	6.50	3.50
Gradall	18.75	6.25
50 kW generator	6.00	4.00
Water wagon (45.5 kL)	19.50	10.50
Crushing plant	36.00	24.00
Calcium chloride spreader	2.40	0.60
Trucks (miscellaneous)	$/month	$/month
5-tonne service	1,400	600
3-tonne flatdeck (c/w 2-tonne crane)	900	600
Tire truck	2,450	1,050
Line truck	2,275	1,225
Lube truck	1,960	1,540
Fuel truck (13.6 kL)	2,610	1,890
Dump truck (10-tonne)	2,275	1,225
Sanding truck (10-tonne)	2,275	1,225
Blasting truck	3,150	1,350
Fire truck	325	175
Ambulance	350	150
Personnel bus (24 passenger)	1,050	450
Pick-up (1-tonne)	480	320
Pick-up (3/4-tonne)	480	320
Lo-boy and tractor	600	300
Hi-boy trailer	600	300

TABLE A8-7

Labour Rates for Mine Operations Staff
(October 1977 Dollars)

Hat Creek Project Mining Report 1978

	Hourly Rate
MINING *	
Bucket wheel operator	14.90
Shovel operator Belt wagon operator Bucket wheel oiler Stacker operator	13.95
Rotary drillers Crane operator - 27 tonne + Dozer operator I Grader operator I F E L I Bucket wheel helper	13.40
Percussion driller Dozer operator Grader operator F E L operator Certified blaster Haul truck operator Shovel operator II	13.05
Conveyor operator Mobile crane operator - 27 tonne Diamond driller Backhoe operator F E L - 4 cubic yard	12.40
Personnel driver Dump truck driver Hyab truck driver	12.05
Flat deck truck driver Shovel learner	11.70
Equipment learner Blaster helper	11.35
Labourer	11.00

(Continued)

* hourly rates include:
- trade base rate
- shift differential......$ 0.45/hr
- 8 hr/week overtime
  (48 hr/week)......22% of base rate
- payroll burden ......33% of base rate

TABLE A8-7 (Continued)

	Hourly rate
MAINTENANCE AND SHOPS*	
Certified journeyman -Electrician -Machinist -H.D. mechanic -Pipefitter -Gas mechanic -Welder	13.95
Boiler maker welder - certified Tradesman - uncertified I Carpenter Radio repair technician	13.95
Tire repairman I Tradesman - uncertified II Boiler maker welder - uncertified Carpenter II Painter Automotive mechanic	13.00
Tradesman - uncertified Tire repairman II	12.65
Utilityman - labourer Lube serviceman Service truck driver	11.95
Trades helper	11.25
Labourer	10.90

(Continued)

*hourly rates include:
- trade base rate
- shift differential
- overtime allowance
- payroll burden

TABLE A8-7 (Continued)

	Hourly Rate
SERVICE LABOUR **	
Receiver/Shipper	11.25
Warehouseman I	10.60
Fire Department helper	10.30
Warehouseman II	10.00
Tool Crib Attendant	10.00
Fork Lift operator	10.00
Labourer/Janitor	9.40

** Above rates include payroll burden 33% plus 4 hours overtime per week (44 hr/week). No shift work.

TABLE A8-8

Annual Salaries - Mine Staff
(1977 Dollars)

Hat Creek Project Mining Report 1978

	Base Rate Per Annum	Payroll Burden 26%	Rate per Annum
Mine Manager	43,000	11,200	54,200
Assistant Mine Manager	37,000	9,600	46,600
Superintendent - Mine	35,000	9,100	44,100
Superintendent - Maintenance	35,000	9,100	44,100
Superintendent - Mine Engineering	35,000	9,100	44,100
Senior Geologist	27,000	7,000	34,000
Geologists	21,000	5,500	26,500
Mine Engineer	25,000	6,500	31,500
Junior Engineer	19,000	4,900	23,900
General Foreman	28,000	7,300	35,300
Foreman	24,000	6,000	30,000
Chief Accountant	28,000	7,300	35,300
Purchasing Agent	24,000	6,200	30,200
Surveyor	19,500	5,100	24,600
Rodmen	15,500	4,000	19,500
Secretaries	15,000	3,900	18,900
Clerks	12,000	3,100	15,100
Typists	12,000	3,100	15,100
Technicians	17,500	4,500	22,000

TABLE A8-9

MINE OPERATING PARAMETERS

Operating and Maintenance Schedule*	
Operating and maintenance days per year	- 365 days/year
Mine production - operating shifts/day (1 crew on swing shift)	- 3 shifts/day
Field maintenance - operating shifts/day (1 crew on swing shift)	- 3 shifts/day
Shop maintenance - operating shifts/day (1 crew on swing shift)	- 3 shifts/day
General Service (5 days/week, 52 weeks/year, no swing shift)	- 1 shift/day
Operating and maintenance hours/shift	- 8

* Operating and maintenance shifts shown are applicable to peak production years and should vary accordingly as total mine production or coal demand decreases.

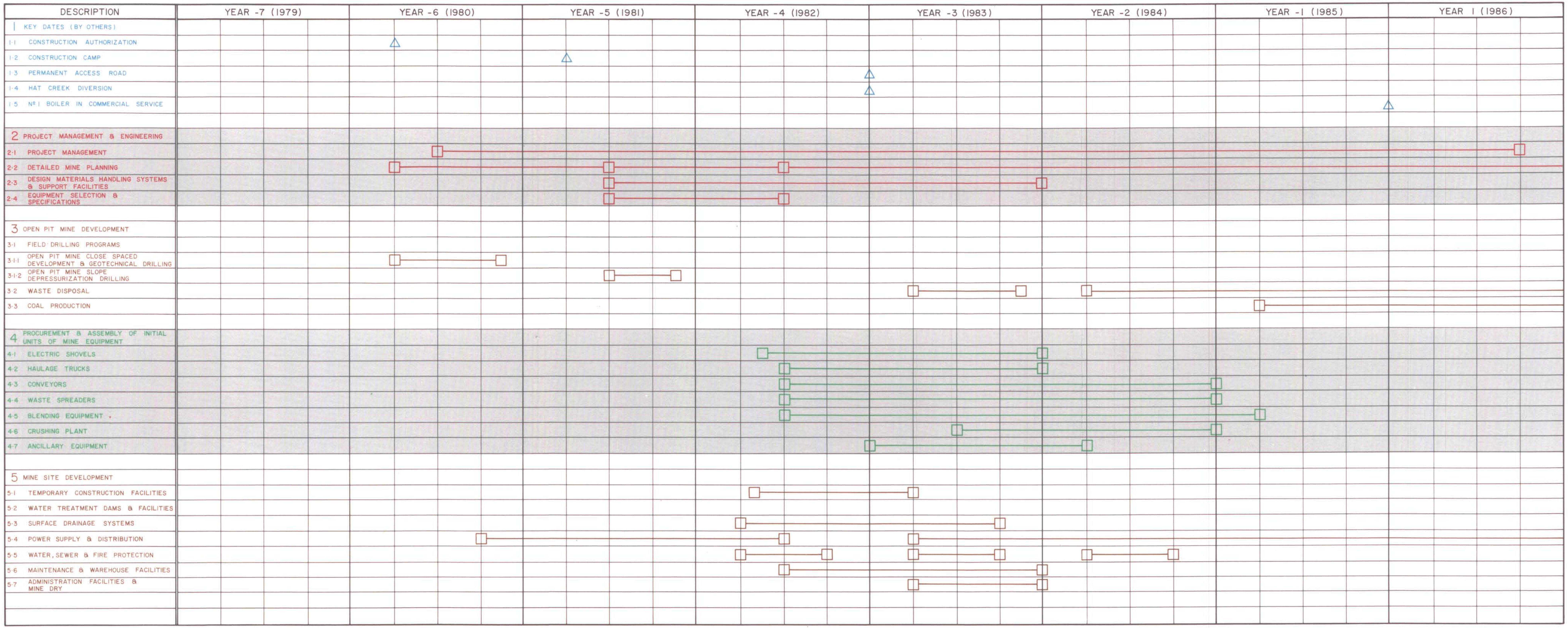
SECTION A9.0 - CONSTRUCTION SCHEDULE

A schedule of activities from the date of construction authorization in year -6 (April 1, 1980) through year 1 of production (1986) is shown on Fig. A9-1.

cominco-monenco joint venture

PLATE A9-1

CONSTRUCTION SCHEDULE



SECTION A10.0 - IMPACT OF POSSIBLE OPERATIONAL CHANGES

The following is a list of possible changes in project operation for which impacts to the mine design and operation have been considered:

1. Plant requires 20 percent more coal.

2. Plant requires 10 percent higher quality coal.

3. Plant requires 10 percent less undesirable fuel constituents.

4. Excavation of 20 percent of coal and/or waste requires blasting.

5. Mines inspector orders 20 percent reduction in airborne dust levels. Fugitive Dust Report

6. 20 percent longer than planned periods of MCS operation required.

7. Design changes to installing two 300 MW units initially.

8. First power is needed by 1984.

9. Only the Flucogas project goes ahead.

A10.1 PLANT REQUIRES 20 PERCENT MORE COAL

(a) Short-term - Less Than 6 Months Duration

It should be possible to meet this requirement without any major problems by utilizing spare capacity built in to handle peak requirements. If the goal of storing 6 months coal for full load conditions at all times is achieved, a slight shift in emphasis from waste to coal removal would be sufficient to meet the extra coal requirements. Coal represents 35 percent of the total volume of material to be moved. If waste stripping were to fall behind schedule as a result of producing more coal it would be possible to contract out part of the waste removal.

Conveying equipment should be able to handle the extra tonnage because it is sized to meet peak hourly requirements. There may be a very slight reduction in productivity of the truck and shovel system due to increased waiting time at the conveyor loading stations. A higher level of equipment utilization could result in some loss of selectivity in meeting coal quality. The variation in quality would not be a major deviation from the datum fuel.

(b) Long-term - Continuous

A continuous requirement for 20 percent more coal would require an additional 16.8 m^3 (22 yd^3) shovel and four or five extra trucks. The conveying system should be adequate.

During the lead time required to obtain the additional equipment it may be necessary to contract out some waste removal.

Adequate reserves are available to maintain higher production levels throughout the plant life although it could be at a lower average coal quality. The long-term impact would be to increase the cost of coal because the enlarged pit would have a higher waste to coal ratio, although this may be offset in part by the economies of scale.

A10.2 PLANT REQUIRES 10 PERCENT HIGHER QUALITY COAL

(a) Short-term or Intermittent

Short-term or intermittent improvement in the quality of coal could be supplied by high-grading, however this would lower the average quality of the remaining coal. Some improvement, possibly 5 percent, could be gained through selective mining techniques.

Raising the cut-off grade would also provide coal of higher quality. This action would increase the quality of low-grade coal to be removed and stockpiled or dumped as waste and would reduce the level of resource utilization (unless an alternate use was available for low-grade coal). This approach would increase the cost of the fuel.

(b) Long-term - Continuous

Raising the cut-off grade significantly would be unacceptable unless alternate uses were available.

The most likely practical alternative would be coal washing, which would require the design and construction of a plant, would increase the cost of coal and lower the effective utilization of the resource.

A10.3 PLANT REQUIRES 10 PERCENT LESS UNDESIRABLE FUEL CONSTITUENTS

There are too many variables to define specific plans for handling problems associated with undesirable fuel constitutents. For each case the specific problem must be identified and corrective action initiated. There are some general approaches which would be followed to help solve the problem.

A10.3 PLANT REQUIRES 10 PERCENT LESS UNDESIRABLE FUEL CONSTITUENTS - (Cont'd)

(a) Coal Quality Parameters

A thorough knowledge of the spatial distribution of coal quality parameters would provide the basis for adjusting operating plans to avoid an excess of an undesirable constituent.

(b) Selective Mining

Rejection of low-grade coal or waste bands by selective mining would remove or reduce some undesirable constitutents.

(c) Coal Beneficiation

Coal washing or some other form of beneficiation would reduce the amount of some undesirable constitutents.

A10.4 EXCAVATION OF 20 PERCENT OF COAL AND/OR WASTE REQUIRES BLASTING

Current plans provide for blasting 50 percent of the coal and 10 percent of the waste. Should it be necessary to increase these quantities some additional equipment would be required.

The cost of mining would not be significantly affected by additional blasting costs: the order of 1 to 2 percent of mining cost. Part of this increase would be offset by equipment productivity improvement.

A10.5 MINES INSPECTOR ORDERS 20 PERCENT REDUCTION IN AIRBORNE DUST LEVELS

Dust control measures planned include extensive use of water sprays or possible chemical coating agents and the revegetation of disturbed areas on a progressive basis. Possible further reduction measures can only be evaluated after an assessment of actual operating conditions.

A10.6 20 PERCENT LONGER THAN PLANNED PERIODS OF MCS OPERATION REQUIRED

The anticipated level of MCS operation, which would require the exclusive use of "D" zone coal, is an order of magnitude smaller than the percentage of D zone coal that would be mined over 35 years. An increase of 20 percent in the level of MCS operation would have an insignificant impact on the mining operation. There would be a marginal reduction in the average quality of coal supplied to the powerplant at other times.

A10.7 DESIGN CHANGED TO INSTALLING TWO 300 MW UNITS INITIALLY

A modified mine plan and schedule would be required should a decision be made to install two 300 MW units initially at the powerplant.

The phasing of installation of some equipment and construction of facilities at the mine may be required.

A10.8 FIRST POWER IS NEEDED BY 1984

The mine could be in production in less than the indicated lead time for boilers and turbine generators. The Hat Creek diversion could be a critical item and should be completed 3 years in advance of the first unit in-service date. The time consumed in the licencing process is probably the major limiting factor in meeting an accelerated production schedule.

A10.9 ONLY THE FLUCOGAS PROJECT GOES AHEAD

A small-scale mining operation should be developed probably on a contract basis with the Contractor to supply the necessary facilities and equipment.

SECTION A11.0 - FUTURE WORK

A11.1 GENERAL

A certain amount of additional work is recommended for completion prior to the submission of application for project licences. This work described in Sub-section A11.2 would be essentially a refinement of some aspects of the recently completed studies. Other tasks which should be completed before final design commences are noted in Sub-section A11.3.

A11.2 PRE-LICENCING WORK

(a) Update Plans to Reflect 1978 Drilling

Additional drilling programmes are in progress and scheduled for completion in September 1978. These programmes are designed to confirm geological structure, coal quality, pit slope stability and groundwater data.

The geological plans and sections would be revised to reflect the improved interpretation which would be incorporated into the computer model of the deposit to provide the basis for more detailed mine planning.

(b) Refine and Confirm Boiler Fuel Specification

The Boiler Fuel Specification is the most critical item of powerplant design data. Further laboratory testing and detailed analysis of the test data is planned to establish a high level of confidence in the final specification.

It is planned to submit the results of this work to a recognized specialist consultant for review.

A11.2 PRE-LICENSING WORK - (Cont'd)

(c) Refine Mining Plans

Due to lack of time the degree of detail that could be incorporated into the planning process has been limited. The recent drilling programme has led to a more thorough understanding of the coal deposit and a much higher degree of confidence in the geological interpretation which will make more detailed consideration of mining schemes practical.

It is recommended that:

1. The basic shovel/truck/conveyor scheme be refined in an effort to improve the economics.

2. The bucket wheel excavator/conveyor mining system which requires much more detailed planning than was previously possible, be re-examined.

3. The practicability be established, in detail, of sequencing the mine excavation and disposal arrangements.

4. The present assumptions on dilution, selective mining and run-of-mine coal quality variation be tested.

Should re-examination of the bucket wheel excavation system conclude that it is the most economic practical mining method, some shift in the location of certain project facilities would be required. Possible changes would include:

1. Relocation of the main mine access to the southeast corner of the deposit.

A11.2 PRE-LICENSING WORK - (Cont'd)

2. A shift in the primary waste disposal area from Houth Meadows to Medicine Creek.

3. Relocation of the blending area and overland conveyor route.

The impact of these changes would be favourable from an environmental viewpoint.

(d) Stockpiling and Blending

The current project plan should be re-examined to optimize the stockpiling and coal blending facilities between the mine and the powerplant. Potential exists for a reduction in capital and operating costs.

(e) Waste/Ash Disposal Schemes

There are some alternatives to the basic waste and ash disposal schemes which warrant closer examination. These alternatives, which appear to offer both economic and environmental advantages, require close integration of the mine and powerplant designs.

(f) Land Reclamation Testwork

It is essential that the present reclamation testwork be continued and extended through the licencing, final design and construction periods.

(g) Examine Operating Options

There are a number of scenarios available for operating the mine. These need to be examined carefully considering their impact on labour relations, economics and management control.

A11.3 FINAL DESIGN PREPARATORY WORK

The final design preperatory work would include:

1. Detailed planning of final design phase.

2. Closely spaced drilling in initial mining area.

3. Coal crushing tests.

4. Coal beneficiation testwork: Should coal beneficiation become an essential part of the project, further testing would be required to establish design parameters before the final design was started.

SECTION A12.0 - LIST OF MINING REPORTS AND STUDIES

1 J.F. McIntyre, Dolmage Campbell. Hat Creek Project - Interim Report on Coal Analysis No. 1 Openpit Deposit. June 1975.

2 Dolmage Campbell. Hat Creek Coal Deposits - Proposed No. 1 Openpit - Statistical Tables of Proximate Analysis Data. July 1975.

3 P.T. McCullough. Memorandum on Hat Creek Exploration Program. July 1975.

4 PD-NCB. Interim Report on Geological and Geotechnical Exploration at Hat Creek. November 1975.

5 PD-NCB/Wright/Golder. Preliminary Report on Hat Creek Openpit No. 1. March 1976.

6 PD-NCB/Wright/Golder. Preliminary Report on Hat Creek Openpit No. 2. March 1976.

7 Dr. A.J. Sinclair. Inter and Intra Laboratory Reproducibility - 1976 Hat Creek Coal Analyses. May 1976.

8 Birtley. Results of Washability and Plant Washing of Samples from A, B and C - the Hat Creek Deposit. June 1976.

9 PD-NCB/Wright/Golder. Combined Pit Operation Study for 5000 MW Powerplant. January 1977.

10 PD-NCB/Wright/Golder. Revised Report on Hat Creek Openpit No. 1. March 1977.

11 PD-NCB/Wright/Golder. Hat Creek Geotechnical Study. March 1977.

12 Hill, Carleton University, Ottawa. The Mineralogy of Coal from Hat Creek, British Columbia - Phase 1: A Report on Three Samples A.B.C. March 1977.

13 Mandi and Brown. Petrographic Studies of Hat Creek Coals. April 1977.

14 Papic, Warren and Woodley (BCHPA). Hat Creek Coal Utilization. April 1977.

15 Dolmage Campbell. Exploration Report - No. 1 Hat Creek Coal Development. June 1977.

16 Stone and Webster. Hat Creek Coal Utilization Study. October 1977.

17 Dolmage Campbell. Petrographic and Geologic Features of Oxidized (Burnt) Rocks - Hat Creek Coal Deposits. November 1977.

18 UBC Dept. of Metallurgy. Mineral Matter Content and Gross Properties of Hat Creek Coal. March 1977.

19 Dr. A.J. Sinclair. Evaluation of Analytical Data from Test Holes 76-135 and 76-136 - Hat Creek No. 1 Coal Deposit. March 1977.

20 Simon-Carves. Washability Testwork of 1977 Bulk Samples. February 1978.

21 B.C. Hydro/Canmet. Pilot-scale Preparation Studies with Hat Creek Coal. April 1978.

22 Mintec, Inc. Minability Study Hat Creek Project. April 1978.

23 CMJV - Hat Creek Project, Mining Feasibility Report, Volume 1, Summary. July 1978.

24 CMJV - Hat Creek Project, Mining Feasibility Report, Volume II, Geology and Coal Quality. July 1978.

25 CMJV - Hat Creek Project, Mining Feasibility Report, Volume III, Mine Planning. July 1978.

26 CMJV - Hat Creek Project, Mining Feasibility Report, Volume IV, Mine Support Facilities. July 1978.

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SECTION A13.0 - GLOSSARY

ac	-	acre
ASTM	-	American Society for Testing and Materials
BCM	-	bank cubic metres
BOD	-	Biochemical Oxygen Demand
Btu	-	British Thermal Units
CCRL	-	Canadian Combustion Research Laboratories
cm	-	centimetre
cm^3	-	cubic centimetre
CMJV	-	Cominco-Monenco Joint Venture
f.o.b.	-	free on board
fpm	-	feet per minute
g	-	gram
h	-	hour
ha	-	hectare
hp	-	horsepower
IDT	-	Initial Deformation Temperature
IREM-MERI	-	Institut de Recherche en Exploration Minérale - Mineral Exploration Research Institute
J	-	Joule
kg	-	kilogram
kV	-	kilovolt
L	-	litre
lb	-	pound
m	-	metre
m^3	-	cubic metre
MCS	-	Meteorological Control System
mg	-	milligram
mm	-	millimetre
min	-	minute

M&S	-	Maintenance and Service
M	-	Mega or million
N	-	Newton
NAMCO	-	North American Mining Consultants
PCB	-	Pollution Control Board
PD-NCB	-	Powel-Duffryn National Coal Board Consultants Ltd.
ROM	-	Run-of-mine
rpm	-	revolutions per minute
s	-	second
t	-	tonne