PD-NCB Consultants Limited

In association with

Wright Engineers Ltd & Golder Associates

Report No. 2

Preliminary Report on Hat Creek Openpit No.1

Volume I

to

British Columbia Hydro and Power Authority

London

March 1976

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CHAPTER I

INTRODUCTION

1. This report deals with a conceptual mine, "Openpit No 1", situated in Area 1 at the northern end of the Hat Creek valley. The extraction of the pit is considered in two phases; down to the 2,400-ft level, called the 600-ft pit and, subsequently, down to the 1,500-ft level, called the 1,500-ft pit.

TERMS OF REFERENCE

2. Report No 1 dated November 1975, "Interim Report on Geological and Geotechnical Exploration at Hat Creek" (Appendix "A") included the full terms of reference for the study but was confined to items 3(a) and 3(b).

3. The main conclusions of Report No 1 were:-

(i) Geological exploration should concentrate on the northern end of Area 2 to bring it up to the same level as Area 1. Area 1 looks the most promising at this stage.

(ii) The depth of geological exploration and geotechnical investigation should be limited to a tentative pit depth of 600 ft, with the exception of an occasional deeper hole to explore major features.

(iii) Geotechnical problems will be severe due to surface mud flows and the nature of the rocks in the mine, particularly the mudstones.

(iv) Coal quality data require to be processed in blocks so as to be able to select a mining sequence and to develop production schedules.

PROGRESS TO DATE

4. A series of meetings took place in Vancouver in December 1975 and it was decided to continue geological drilling at the northern end of Area 2 (as recommended in (i) above).

5. This means, of course, that no additional geological information has been received on Area 1 since the preparation of Report No 1. Also, very little additional geotechnical data will be available.

6. Chapter II summarises the geological and geotechnical position to date but basically it is unchanged since Report No 1 was prepared.

7. A visit was made with British Columbia Hydro and Power Authority (BCH) personnel to Centralia Mine, Washington, USA, to examine pit conditions etc in a geological setting which has similarities to Hat Creek. The difficulties caused by the mudstones in the mine and in the washery were very evident. This mine is now successfully using a bucket wheel excavator and conveyor system in the superficial deposits, having failed disastrously in the deeper, harder rocks.

8. The documentary information received since 19th November, 1975 is listed in Appendix "B".

BASIC DATA

9. Table I lists the basic data used in this Report. Some of the data have been provided by Dolmage Campbell and Associates (DCA) but the remainder is assumed. Quite clearly these assumptions will need verification as soon as it is possible to do so.

CHAPTER II

GEOLOGICAL AND GEOTECHNICAL ASSESSMENT

INTRODUCTION

1. Since Report No 1 was prepared, the borehole logs for Area 1 have been re-examined in order to check the interpretative cross-sections of DCA and BCH prior to preparing additional sections and plans for conceptual-mine-design purposes. No further drilling information has been available for this study but the location and inclination of the more recent boreholes have been checked and, accordingly, some data have been re-plotted. The latest geophysical interpretations have been examined but have little relevance to the current phase of the work.

2. The structural interpretation upon which the present mine-planning study has been based is shown on Plates 1 to 14 and a legend of the symbols and abbreviations used appears on Plate 15. These plans and sections are based on topographical maps compiled to an approximate scale of 1 in to 2,000 ft by McEllanney Surveying and Engineering Ltd from an aerial survey dated 13th May, 1975. Additional interpretative data were also supplied by DCA and BCH.

3. No new information has been forthcoming on the composition of either coal, overburden or parting materials in this area, although boreholes in Area 2 may have some bearing on the geology of the south-eastern side of the mine. No additional information relating to the groundwater of Area 1 has been received.

STRUCTURE

Faulting

4. Four main faults traverse the northern section of Area 1, apparently converging towards the north. Some of these faults may continue well to the south of Area 1, notwithstanding a possible element of E-W cross-faulting. The northern faults from east to west are as follows:-

(i) Mag Fault - a normal fault down-throwing to the east and trending NW-SE

(ii) Fault "H" - a normal fault down-throwing to the west and trending NNW-SSE

(iii) Fault "S" - a normal fault down-throwing to the east and trending N-S

(iv) Dry Lake Fault - a normal fault down-throwing to the east and trending NE-SW

Two faults which occur towards the south of Area 1 are:-

(v) Trig Fault - a normal fault with steep hade down-throwing to the south-east and trending NE-SW

(vi) Finney Fault - assumed to be a normal fault, down-throwing to the south and trending ENE-WSW

If the throw directions on the Trig and Mag faults are correct, there is also the possibility of a further, more easterly fault down-throwing to the west. However, the plan and section data on the magnitude and direction of the Mag, Trig, Finney and Dry Lake faults are uncertain.

5. The effect of the northerly faulting is to form a graben, or trough, along the valley to which the coal is confined but within which the Mag fault and "H" fault produce an ancillary horst. The major coal-bearing area lies between the Mag fault and fault "S". Owing to the shortcomings of the strata correlation there is still no precise information on the throw of the faults but, using the roof and, in places, the floor of the thickest coal units as indicators, there appear to be some marked changes in throw along the faults.

6. The fault positions are shown on the plans of total overburden isopachs and top of coal contours. At the northern end of Area 1 these faults converge north of the small outcrop areas near Hat Creek, coal-bearing strata being apparently absent north of this point. To the south of Area 1, the geology is highly conjectural. The Finney fault has been assumed to have the attitude shown on Plate 1 (as in DCA Report). On this basis, near-surface coal would be absent to the south. Its position is, however, uncertain as is also the direction of its displacement. The location of this fault has a significant effect on the coal reserves and the economic pit limit. Two boreholes south of this fault position show both thinner and thicker coal units, similar to those occurring further north. This could indicate that there is either:-

(i) A reversal in direction of throw along Finney fault, with a northerly down-throw in the west and a southerly down-throw in the east or

(ii) the Finney fault trends parallel to the Dry Lake fault and lies further south in the SW corner of Area 1. In this case, changes in strata orientation would also be indicated.

7. In view of the absence of sound correlation of the coal and large gaps in the drilling hereabouts, no assessment has been made of coal south of the conjectural fault position, in either the 600-ft or 1,500-ft pits. Clearly more exploration is required in this area.

Folding

8. The strata between the faults is inclined almost everywhere.

(i) The structure of the eastern horst is unknown but current drilling will more closely define the extent of the coal.

(ii) In the southern half of the larger fault block west of the horst (ie between faults "H" and "S"), there is a gentle syncline with its axis running north-south and plunging towards the south. Dips on the flanks are up to 30^{o}, with drill core evidence showing localised dips in excess of this value. Some boreholes also show marked variations in strata dip from top to bottom. The coal in this area exceeds 1,000 ft in thickness and there is as yet no clear evidence of fault-induced thickening by repetition. Whilst some of these dips may reflect the presence of non-diastrophic structures, such as slumping and channel formation, the possible occurrence of numerous small faults with throws of less than 50 ft cannot be eliminated. In that case, slices of strata between these faults would be inclined at a variety of angles. The cause(s) of dip inflexions in individual boreholes requires further investigation. The central section of this area has coal incropping beneath superficial deposits. To the north-east of this fault block there is some evidence of down-warping to the west near fault "S".

(iii) The western flank of the mine comprises a coal bed 400 to 500-ft thick, outcropping to the west and dipping in an ESE direction at 30 to 45^{o}. This coal is somewhat thinner and could represent the lowest stratigraphic horizon of the main deposit against which it abuts along fault "S", the upper coals deteriorating rapidly to the west.

9. The east-west sections, Plates 5 to 10, illustrate the above structures, including the disposition of faults and possible folding within the blocks. Clearly a considerable area of conjecture is present owing to the large gaps in the drilling and projections on to section lines. The north-south section similarly shows the interpretation of the overall structure as plunging towards the south with increasing overburden cover.

10. Folding and inclination of strata is indicated on the plan of top of coal contours. It now seems quite likely that the coal in Area 1 is all part of a single sequence of deposition with a quite continuous lower bed and a thick central section in the centre of the syncline or basin. Plate 7 shows this quite clearly.

MATERIALS

Overburden

11. This has been conventionally divided into superficial materials and waste, the consolidated and/or conformable strata above the coal. The superficial materials are typically drift deposits, ie glacial moraines and till debris and alluvium. Much of this material has been subject to periglacial or recent disturbance and large mudslides (see Plate 16) derived from both till and in-situ strata are prominent along the valley. The eastern side of Hat Creek in Area 1 apparently has fairly thick granular deposits, glacial outwash and alluvial gravels. The several types of engineering soils present in the superficial deposits have not been separately identified on the sections. In general, the superficial deposits are thicker on the western and southern flanks of the coal area than along the central and northern valley. Plans showing isopachytes of superficial deposits and contours of the base of superficial deposits are given on Plates 11 and 12. No marked channels or significant local thickenings are present.

12. The Consultants' current observations on the claystones do not differ from those made earlier in the Report No 1. As can be seen from the plans and section, the thickness of overlying claystones increases appreciably to the south. Near the surface, volcanic material has been found in recent drilling in Area 2 so it is possible that such materials, which may include bentonitic ashes, are present but undetected in Area 1. The isopachytes of total overburden are shown on Plate 13 and differ from the isopachytes of superficials outside the coal incrop areas.

Coal

13. No significant correlations have been made other than between occasional adjacent boreholes. Some of the partings can be projected locally but undetected structures or lateral sedimentary variations may be present, which may limit the continuity of these features. No detailed attempt has been made in the sections or plans to show quality variations in the coal. However, in general the quality (notably ash content) deteriorates from the bottom upwards, from north to south and in the upper section of the thicker coal to both east and west. The coal incrop and outcrop areas and top of coal contours are shown on Plate 14

GEOTECHNICAL IMPLICATIONS OF THE GEOLOGY

14. The implications regarding geological structure and materials remain very much as included in the earlier report. In summary, these findings were:-

(i) There is a moderate balance between upper and lower coals within a 600-ft pit.

(ii) The thickness of partings and the frequency of dips in excess of 30^{o} are such that the segregation of waste during excavation will present problems.

(iii) A variety of potential failure modes is possible owing to the apparently low shear strength of some of the superficial deposits, as well as the claystones and the geometric configurations of structure and pit slope.

(iv) Large gaps were present, both within and beyond the immediate coal areas, which require detailed prospecting.

(v) It is not yet possible to fully assess the diggability of overburden nor the behaviour of the several rock and soil types regarding site trafficability.

15. The geological interpretations since the first report have enabled the potential coal area to be extended. This in turn has been increased by the area covered by the 15 to 16^{o} slopes (see Chapter III) and it is suggested that an additional 13,000 ft of drilling should be considered to explore this extension. Much of this additional drilling will be of geotechnical concern and related to slope stability, diggability, etc, although close attention should be given to a possible extension of the coal area to the south-west corner of the proposed mine.

16. Much additional drilling will be required, both to depth within the coal areas and at the perimeter of the mine, if it is decided to excavate the mine to 1,500-ft depth. The tens of thousands of feet of drilling required for that development can of course be undertaken over many years duration, assuming it is possible to organise the waste dumps and lagoons from the 600-ft pit, so that they do not sterilise this potential extension.

GEOTECHNICAL

17. As a result of geotechnical studies carried out by Golder Associates Ltd (GA) in 1975, the following conclusions were presented in Report No 1:-

(i) There is no strong preference for mining Areas 1 or 2 on the basis of geotechnical considerations.

(ii) The nature of the coal deposit and the surrounding rock mass, together with anticipated difficulties in achieving effective drainage, leads to the recommendation that 25^{o} be regarded as the maximum overall slope angle, to be used in preliminary mine planning and that a minimum of 15^{o} may emerge from more detailed studies.

(iii) Since it is considered unlikely that economic recovery of this coal deposit could be achieved without inducing some slope failures, it is recommended that the layout of the pit and access ramp be designed to accommodate such slides. The development of both Areas 1 and 2 should also be considered as a possible means of achieving a secure supply in the event of the disruption of production as a result of slope failure in one pit.

(iv) The presence of significant areas of potential mudslide material, in both Areas 1 and 2, must be taken into account in designing the upper slopes of the pit and also in considering the problems of waste disposal, site road design and plant locations.

(v) Because of the unusual severity of the geotechnical and groundwater problems which are likely to be encountered in mining the Hat Creek coal deposit, it is recommended that provision be made for a detailed study of these problems during the design stages and for on-going geotechnical work during the mining operation.

18. It should be emphasised that the geotechnical problems highlighted above are not an insurmountable barrier to the effective exploitation of the Hat Creek deposit. Experience in similarly difficult mining sites has shown that an early recognition of slope stability problems and the evolution of a mine design which makes provision for these problems, can lead to an economically acceptable mining operation.

19. In view of the fact that a relatively comprehensive set of conclusions could be drawn on the basis of the geotechnical studies completed in November 1975, only a very limited amount of additional work has been carried out since that time. None of the conclusions presented in Report No 1 has been modified as a result of this work.

20. It should also be emphasised that the geotechnical investigations must relate closely to the findings and on-going investigations of the mine geology.

CHAPTER III

MINE PLANNING

GENERAL

1. Valid mine planning clearly depends upon the validity of the geological and geotechnical data, particularly as regards the following:-

(i) Structure

(ii) Quality of the economic mineral

(iii) Physical and chemical properties of all the rocks encountered in and around the mine

(iv) Groundwater quantities and pressures.

Structure

2. Although considerable information is available concerning the main structural features of the Area 1 deposit (see Plates 1 to 14), the stratigraphy of the coal itself is not yet determined, particularly the position, continuity and orientation of the intercalated sterile material. The recommended in-fill drilling programme should provide information on this but in the meantime, the coal can only be regarded as a mass, utilising DCA's assessment of the intercalated waste. The mass of coal and intercalates is referred to as in-situ coal. Within it, only some of the main structural planes of weakness, which have a marked influence on slope stability, have been located to date. More localised discontinuities still remain to be determined and their influence on stability, etc, assessed. This will be a major feature of the recommended geological and geotechnical exploration programme.

Coal Quality

3. Coal quality aspects are dealt with in Chapter V. In brief, some coal-quality data have been obtained under the DCA programme and an overall average quality calculated. It now remains to process these data in terms of mining blocks (see Chapter I, para 3 (iv)). This information is not available for this study and, in any case, it should properly form the basis of a much more sophisticated study involving pit optimisation.

Coal Production

4. The power station coal consumption is based on a calorific value of 6,000 Btu/lb and ash content of 28% (Ref DCA's Report, January, 1975). The in-situ coal is estimated by DCA to contain 22% waste and they have assumed that all

this waste can be mined separately in the pit. However, until more is known about the disposition of the waste in the coal it is considered necessary to allow for some dilution of the coal and it has been assumed that only 15% out of the 22% could be removed, leaving 7% dilution. The moisture content of the coal is not known accurately but 20% is assumed in each case (Ref DCA). The rom coal quality would then be approximately 5,500 Btu/lb and 32% ash. The possibility of up-grading the coal by coal preparation is discussed in Chapter V.

5. In terms of coal production quantities, adjustment in the tonnages has been made for this reduced quality as follows:-

	Annual Production	Calorific Value	Ash including Waste
	tons	Btu/lb	%
Power station fuel	12,000,000	6,000	28
Rom coal	13,100,000	5,500	32
In-situ coal	15,400,000	4,670	39

Physical and Chemical Properties

6. Apart from the chemical characteristics of the coal (proximate, ultimate and ash analyses) and a small number of tests on the rocks overlying the coal (by Klohn Leonoff Consultants Ltd), very little information is available. The mechanical properties of all the rocks are clearly needed for slope design, diggability, etc. These data will be obtained during the geotechnical exploration programme which has been recommended.

Groundwater

7. Little is known about the presence and effect of groundwater in the valley and a hydrological study has been recommended (see Report No 1).

TYPE OF MINE

8. At the conceptual stage, it is necessary to consider all feasible mining methods, both surface and underground. The Area 1 deposit is an obvious candidate for surface mining but, owing to the great depth and narrow width of the deposit, it is conceivable that underground mining might, at some stage, become more economic. Therefore, a preliminary study has been made of possible underground mining methods and this is appended (Appendix "C").

9. It is evident that a very cheap underground mining system would have to be adopted because of the low value of the mineral. The cheapest method of mining a massive deposit is, of course, block caving but sub-level caving seems more practical. The main drawbacks of underground mining are clearly as follows:-

(i) possibility of gas emission

(ii) tendency to spontaneous combustion

(iii) weak rocks, hence major support problems

(iv) presence of slaking mudstones

(v) unknown effects of groundwater

(vi) dilution by mudstones which could not be separated during mining

(vii) reduced quality and increased quantity of rom production due to (vi).

Until these problems are elucidated, underground mining cannot be accepted as feasible. The configuration of the deposit and the nature of the coal would also seem to rule out underground hydraulic mining, insofar as the technology has been developed to date.

10. The economics of underground mining, irrespective of method, also appear very unfavourable, both as regards capital and production costs. It seems unlikely that the capital cost would be less than $300 million ($20 per annual ton of capacity) and the production cost less than $10 per ton. The remainder of this report therefore deals with a conceptual surface mine.

Factors Controlling the Design of a Surface Mine

11. Report No 1 discusses the geotechnical factors which appear to control the design of a surface mine and these are summarised below:-

(i) Slope stability problems will be severe - overall slope angle limited to 25^{o} or less.

(ii) Occasional slides are likely to occur whatever slope angles are adopted and so the mine must be designed to accommodate these slides without loss of production.

(iii) All access ramps must be subjected to stability investigation.

(iv) Superficial mudslides must be guarded against as regards the mine itself.

(v) The waste mudstone and slide material must be disposed of in safe areas and waste dump stability must be ensured.

(vi) Provision must be made to deal with groundwater, particularly as regards its effect on slope stability.

(vii) A considerable amount of in-pit waste is present as partings in the coal and around the periphery.

12. Certain conclusions have been drawn from consideration of these control factors, these being as follows:-

(i) The shape of the deposit and the instability of the waste mudstones preclude back-filling in the mine whilst mining operations are in progress.

(ii) The shape of the deposit and the inherent instability of the slopes favour a roughly circular pit with low slope angles. A concave face is fundamentally more stable than a straight or convex face. "Noses" or buttresses are to be avoided at all costs.

(iii) Rapid advance in depth would increase the number of benches in coal and hence improve the instantaneous stripping ratio, particularly as the upper benches advance up the hillsides.

(iv) The slaking mudstones in the pit would adversely affect rubber-tyred vehicles, particularly on ramps and therefore the use of these has been confined as far as possible to level haul, whilst recognising that all haul roads will probably require surfacing.

(v) Because of the depth of the pit, the large output and the mudstone problem, conveyors have been selected for the main haul out of the pit, both for coal and in-pit waste.

(vi) The main pit access will be a straight incline aimed more-or-less at the centre of the coal mass where it will not be endangered by slides.

(vii) Owing to the paucity of geological and geotechnical information pending the completion of in-fill drilling and geotechnical investigations, it has been decided to limit the depth of the first mine to a nominal 600 ft.

(viii) Selective mining on the benches will be required to eliminate as much as possible of the mudstone partings from the coal.

(ix) Blasting, if required, must be carried out in such a way that slopes are not weakened further.

(x) The probable circular shape of the pit and the hardness of the coal make bucket-wheel excavator/ conveyor systems unattractive. (It is possible however, that they could be used on upper benches in till if boulders and concretions are not present in significant quantities). Further work on the superficial deposits will show whether such a system could be applied.

13. Following these guidelines, a conceptual pit has been developed and important design features are discussed below (see Plate 17).

Pit Slopes

14. As mentioned in para 11 (i), the maximum overall pit slope angle is limited to 25°, although it is recognised that in places the safe-slope angle will be less than this. Therefore, for permanent slopes (which would be more carefully designed than working slopes), eg side slopes at the entrance to the incline, this angle has been adopted. However, for working slopes an angle of 15° 57' has been adopted, recognising that the final pit slopes could be steepened to whatever angle is safe for the short period involved, bearing in mind that stability is time-dependent.

15. This slope is based on the section shown on Plate 18 and incorporates the following features:-

(i) Bench width sufficient to allow 150-ton trucks to pass in safety.

(ii) With regard to bench height, geotechnical considerations have indicated a maximum of 50 ft, depending upon local discontinuities and groundwater conditions. 40 ft has been selected, this height matching the excavators selected (see Chapter IV).

(iii) Bench slope angle has been selected at ½ in 1 (64°). This needs confirmation as it depends on local conditions, blasting practice, etc but it is a typical value which seems reasonable under the circumstances.

Main Incline

16. The angle of slope of this incline has been carefully considered. The advantages of a short, steep incline are, generally, minimum excavation and minimum coal sterilised under the lower part of the incline. In this particular case, a steep angle of 1 in 3½ (16°) would be acceptable to belt conveyors but then a separate access road would be required for mobile equipment, maintenance vehicles, etc. The overall transport distance would, however, not be reduced by a steep incline because the coal still has to be transported an average distance measured from the centre of gravity of the deposit to the power station (in fact the transport distance would be slightly increased).

17. The excavation outside the pit required for the incline has been minimised by locating it in the centre of the valley and the amount of coal sterilised is reduced because the footwall dips in the same direction as the incline. By starting the incline outside the deposit, the amount of coal underlying it has been further reduced. Actually, even this coal would not be permanently sterilised as at the end of the operation the incline could be cut back and the coal recovered.

18. Since the incline is to be suitable for large trucks which may have to leave the pit fully loaded on occasions, the maximum gradient should not exceed 1 in 10. However, manufacturers recommend 1 in 15 (3^{o} 49') for large trucks and this is advantageous on a long, low-gear haul. Also, it is more suitable for walking heavy equipment. 1 in 20 (2^{o} 52') was considered as it would have been required if large, bucket-wheel excavators had been selected, which is not, however, the case. Therefore, 1 in 15 has been adopted. The width of the incline (200 ft) has been selected to accommodate three conveyors with an access road on both sides, cable and pipe track, etc (see Plate 19).

Depth Limitation

19. The reasons for limiting the depth of the pit to 600 ft were given in Report No 1 but, in fact, the elevation of the floor of the pit has been selected arbitrarily as 2,400 ft (assuming the average elevation of the surface is 3,000 ft). However, this pit will have a vertical slope height of 1,150 ft on the south-western side. There are very few pits in this type of strata as deep as this, so formidable stability problems are anticipated.

20. In Report No 1 it was roughly estimated that the in-situ coal reserves available would be in excess of 300 million tons - sufficient for about 22 years of power station operation. More detailed calculations based on the completed geological interpretation given in this Report have revealed that about 450 million tons (in-situ) (proved probable and possible) will be available at a reasonable stripping ratio in the 600-ft pit - sufficient for 30 years of power station operation. It is anticipated that sufficient additional coal for the full 35 years' life of the project would be available from the following sources:-

(i) Improved mining technology.

(ii) Lateral extension of the coal deposit - further drilling in the eastern area has already shown promising results.

(iii) Extension in depth below 600 ft following detailed geotechnical investigation.

21. During the course of mining the pit, ample opportunity will be afforded to observe the behaviour of the slopes, to back-analyse slope failures and to determine the pattern of discontinuities, so that a valid basis would be available for deepening the pit long before any decision had to be taken. The formidable problems of predicting slope behaviour from

borehole data, particularly at great depths, make it impractical to design a deeper pit at this stage. The 600-ft pit is complex enough and, indeed, exposures of the various rocks at the earliest date will be necessary to supplement exploration data collected from boreholes and surface exposures.

22. However, as requested, a pit has been postulated down to the 1,500-ft elevation, ie a 1,500-ft pit, which will result in a maximum vertical height of slope of 2,500 ft. This pit would contain mineable in-situ coal reserves of some 910 million tons, ie 91% of the estimated total reserves in the deposit of some 1,000 million tons. Access to the deeper levels could be obtained by extending the main incline to about 2,200-ft elevation, after which either a subsidiary conveyor incline, turned back in the opposite direction to the main incline, or trucking ramps would be required. Other methods of recovery of the deeper coal could also be considered if conveyors or trucks could not be used.

Pit Design

23. As mentioned in Report No 1, this design has been done manually on a practical basis. It is, of course, normal procedure nowadays to optimise the design of large openpit mines, eg copper, with the assistance of a computer. This should be done for Hat Creek, using suitably modified programs, when the existing data have been reclassified into mineable blocks. Nevertheless, it is confidently considered that a satisfactory design has been produced and that a computer-based, optimised design at this stage would not be greatly different, particularly as so much essential data are missing. A computer program which could take into account the important geotechnical controls would be very complex indeed.

24. The steps taken in establishing the design shown on Plate 17 are as follows:-

(i) Determine the centre of gravity of the in-situ coal reserves above the 2,400-ft elevation as this represents the best "target area" for the coal-transport system of a pit to this depth.

(ii) Roughly equalise the waste excavation on the east and west sides of the pit. This involves moving the target area somewhat in a north-easterly direction, ie away from the south-west wall. The distance which can be moved in this direction is limited by the cut-off of the coal by the eastern boundary fault (the Mag fault).

(iii) Draw a conical-shaped pit centred on the target area and bottoming at the 2,400-ft elevation with overall slopes of 15° 57'. These are all working slopes, except at the point of entry of the incline.

(iv) Widen out the pit in stages (8) until most of the coal above 2,400-ft elevation has been included, ie in a series of frustums. Actually, these frustums depart from the circular and become elongated in the southerly direction to conform to the shape of the coal deposit. Plate 17 shows the surface intercepts of these frustums and also the shape of the floor of the mine. This floor would, of course, be almost entirely coal and, if a wide, flat floor such as that shown in the final stage could be reached, the implication is that the pit could be deepened further. However, if the slopes are becoming very unstable, a flat floor of such width could not, in fact, be created.

(v) Locate the bottom of the incline at the point of the cone mentioned in (iii) at the 2,400-ft level. Locate the top of the incline in the valley bottom at 2,750-ft elevation. Hence the incline ascends 350 ft and its plan length is 5,250 ft. It is almost in a straight line with the power station site at the north end of the valley.

(vi) In order to develop the mine to the stage reached in (iii), determine the minimum coal-face length which can sustain initial production and be increased over a 2-year period to sustain full production, and incorporate this with the upper part of the permanent incline (1,000-ft long). At this point a conveyor and feeder station can be installed. Actual coal production during this stage has to be limited because it must be stockpiled and 1 million tons has been selected as the limit. Whilst the creation of a substantial stockpile in the pre-production stage is necessary, it is also most important to develop the mine so that production can commence on time and at the desired rate and continue uninterrupted according to the production schedule.

(vii) Measure the instantaneous stripping ratio at each stage of development. This is expressed as bank yd^3 of waste/ton of in-situ coal. This ratio enables the volumes of waste for a given output of coal to be calculated.

The instantaneous stripping ratio is the ratio of the volume of waste (yd^3) to that of coal (in tons) for one cut off each bench being worked at that time, ie the last increment in the stage. This reduces to the ratio of areas on the sloping sides of the pit. This has been approximated by measuring the lengths of waste and coal respectively at 100-ft level intervals and summating. Table II gives these figures, segregated into each main category of waste rock, based on both in-situ and rom quality coal. In the latter case, the same assumptions regarding segregated waste are made as stated above. The effect is to increase the waste volumes and decrease the coal tonnages, thereby increasing the stripping ratios. (The in-situ values are required for pit planning and the rom values for economic calculations.) The cumulative volumes of waste and tonnages of coal, and the overall stripping ratios, are also given. The sum total of all the volumes is, of course, the total volume of the pit up to the stage in question, that of the coal being the mineable reserves. Plate 20 shows the cumulative volumes of waste plotted against the cumulative tonnage of in-situ coal mined out.

(viii) In order to obtain schedules of production by years, Plate 21 has been constructed showing coal production per year based on three 750-MW generators and Table III shows the cumulative effect to the end year. The waste production on a yearly basis can also be obtained by reading off from Plate 22 against the cumulative coal production (in-situ) at the year end. All the production schedules are shown in Table III by years according to the proposed stages of mine development indicated for in-situ and rom coal and all types of waste. The yearly stripping ratio (relative to rom coal) is also indicated in Table III.

Development Programme

25. Plate 23 shows the construction schedule for the 3 x 750-MW power station and the development schedule for Openpit No 1 which is required to meet the coal demand of the power station.

ENVIRONMENTAL ASPECTS

26. Due to the conical shape of the proposed pit and the low slope angles, waste material cannot be dumped within the excavation until the mining of No 1 deposit is complete. This poses considerable problems in terms of availability of dumping space and land rehabilitation. Should further pits be developed, however, these could be so planned that the worked-out No 1 pit is used as a repository for some of the overburden and waste removed.

CHAPTER IV

MINING OPERATIONS

INTRODUCTION

1. Hat Creek is a massive deposit, shallowly overlain with superficials and pit waste, and extensive in depth but limited in area. This is unusual for a coal deposit and is more usual for a massive metalliferous ore deposit. Because of the shape of the pit, conventional large-scale equipment for overburden removal in opencast coal mines, such as draglines or bucket-wheel excavators, is not recommended. Mobility is an essential requirement of machines employed in conditions where the stability of the strata is suspect and the equipment which has been suggested provides the necessary degree of flexibility in operation and is readily available.

2. The various operations described below are based on the production schedules detailed in Table III. The equipment required each year to achieve the objectives set by the schedules is shown in Table IV. The equipment has been selected from practical considerations and has not been optimised.

DEVELOPMENT

3. The quantity of coal exposed at the outcrop will enable sufficient coal to be mined to permit geotechnical examination and its physical characteristics to be thoroughly examined before the full output is required.

4. Initial development in stages 1 and 2 will take the form of an extending wedge, deepening to form the conveyor incline (Plate 17). At the beginning of stage 3, a cone will be formed and extended in depth until, by the beginning of stage 4, the apex of the cone will reach the 2,400-ft elevation, which is approximately 600 ft below the surface level. Stages 4, 5, 6, 7 and 8 will extend the base of the cone to form a flat-bottomed frustum at the 600-ft depth. By the completion of stage 8, the majority of coal to that level will have been removed and future supplies will have to be taken from the coal below the 2,400-ft elevation, ie down to the 1,500-ft elevation if a safe pit can be developed to that depth. Benches will be 120-ft wide and 40-ft high.

DIVERSION OF HAT CREEK

5. This river would be dammed and the water pumped into a canal made outside the final limits of the planned pit. This is described in more detail elsewhere but the work on the river must be completed before a pit of any depth exists. Plate 16 shows the location of those works outside the 600-ft pit surface intercept (stage 8). The surface intercept of the 1,500-ft pit (stage 9) is also shown and this includes the diversion dam. Therefore, either the dam would have to be re-located or an early decision taken to locate it further upstream.

SUPERFICIALS

6. The vast majority of superficials are at elevations above the top of the conveyor incline. They consist of glacial till (sand, gravel, clay and boulders), claystones and some volcanics which will be removed by scraper operation without pre-blasting.

7. Further geological information will enable various types of superficial materials to be located accurately. When this has been done, the mining operation must be planned to ensure that the stable and useful granular materials are initially delivered to the thermal plant site and used as fill. They would also be used to form dams to control the claystone waste-dump areas in the western valley and, probably, a permeable base also to promote drainage.

8. Large boulders occurring in the superficials (rocks larger than three feet in general dimension would cause difficulty and delay in the scraper operation) must be moved aside by bulldozer, to be loaded separately by shovel into trucks for delivery as foundation and building stone for retaining walls and stabilisation purposes.

9. Other superficials will be loaded on to the waste disposal conveyor via a ground hopper from scrapers.

10. The benches will be extended outwards from the centre of the pit. The extension will follow detailed phasing so that the movement outwards is regular and bench formation in accordance with the planned shape. The full volume of superficials will be removed over the life of the pit and the quantities to be removed each year are given in Table III and illustrated on Plate 21. The volume increases gradually from 6 million to 18 million bank yd^3 per year over the first 30 years of operation and then sharply to 27 million bank yd^3 over the next four years, due to the lateral extension of the pit up the sides of the valley.

PIT WASTE

11. In accordance with the schedule of operations, simultaneous removal of superficials and pit waste will be carried out throughout the life of the pit. The pit waste material will be delivered to the incline conveyor by 150-ton trucks loaded by 15-yd^3 shovels.

12. The quantity of pit waste to be removed each year (Table III) increases from 3 million bank yd^3 per year at the beginning of the operation to 33 million bank yd^3 per year after 30 years. It then increases sharply over the next four years to 60 million bank yd^3 per year.

COAL AND SEGREGATED WASTE

13. Removal of the coal will be in three phases:-

(i) Initial extraction of a small tonnage for sampling and testing.

(ii) Extraction of about one million tons of rom coal for stockpile purposes. This tonnage was selected because it:-

(a) represents approximately one month's power station requirement at full output or three months' stock for the first year of operation after start up,

(b) cannot be increased due to lack of storage space and until the behaviour of the coal in the stockpile is known.

(iii) Commencement of deliveries to the thermal power station over three equal increments spread over two years as each generator comes on load. Full production starting in April 1985 is between 13 and 14 million tons of rom coal per year.

MINIMUM FACE LENGTH

14. In order to maintain the required production, a minimum length of exposed coal face is required. Calculations for the Hat Creek deposit show that for the initial sampling work a length of under 100 ft is adequate. This increases to 200 ft for stockpile production in 1983-84 and to 600 ft for full production. All planning and scheduling has been based on these figures, increased by a margin of 50%, ie the minimum face length required for full production is taken as 900 ft. At the same time, an adequate area of coal in advance of the face has to be cleared of superficials and waste and development has been scheduled to enable one month's coal production in advance to be exposed throughout the working of the deposit.

15. The coal will be loaded into the 150-ton trucks by 15-yd^3 shovels and delivered to the incline conveyor feeder hoppers. The in-situ coal is said to contain an average of 22% waste material and it is assumed that 15% can be removed by selective loading at the face. Separation will be decided visually by the machine operators and it is expected that they will become proficient at selection after about two months of working in the pit. 100% segregation cannot be achieved in any case, and a better estimate cannot be made until more is known about the disposition of these waste intercalations.

16. The quantities of coal and segregated waste to be removed each year (Table III) require careful consideration. These materials will be sent out of the pit on separate conveyors and directed to stockpile or waste dump.

Potential for Increasing Mine Output

17. Openpit No 1 is designed for an output of 13,100,000 short tons per annum of rom coal but after the first few years of operation there would be sufficient face length to sustain much higher outputs, eg 30,000,000 short tons per annum or more. Clearly, this rate of output would reduce the life of the mine. If demand for this quantity of coal should develop there are a number of benefits to be derived from concentrating this output at the one mine rather than opening a second mine, ie:-

(i) Deferred expenditure in opening a second mine.

(ii) Reduced production cost due to increased volume.

(iii) Earlier availability of the worked-out mine for back-filling (which may be critical in a valley where spoil disposal is such a problem).

(iv) The possibility of complete reclamation of Openpit No 1.

18. The increased output would involve:-

(i) Increased transport capacity, particularly conveyors.

(ii) More rapid face advance which would reduce time-dependent slope failure.

(iii) More blasting, probably requiring larger equipment.

(iv) More and/or larger loading equipment.

19. Near the end of the life, output would, of course, have to be reduced but there would be ample time to develop a second mine to replace the lost production.

BLASTING

19. It has been assumed that all the coal and some of the pit waste will require blasting but that it will not be necessary for the superficials. Initially, all drilling will be by crawler rigs operated by diesel-driven air compressors. It has been assumed that 4-in diameter holes will be drilled the full 40-ft bench depth at 10-ft centres. The use of a small drill is unusual for large deposits but this method has been selected instead of large-hole blasting because:-

(i) the stability of faces will be easier to control

(ii) overall pit vibration will be reduced by the smaller quantities blasted at one time

(iii) separate blasting of coal and segregated waste is more practicable

(iv) movement and flexibility of the drills are improved

(v) fragmentation of the blasted material is more satisfactory

(vi) delays due to blasting will be minimised.

20. The disadvantages are:-

(i) an increased labour force is required

(ii) the cost is estimated to be 4¢ per ton of rom coal higher.

21. At a later stage, it may be decided to select larger drills but, apart from pit stability, this will depend partly on the quantities of pit waste requiring blasting.

22. For all estimates regarding blasting the powder factors used are 0.4 lb/ton for waste and 0.3 lb/ton for coal, although it is believed that these factors are on the high side.

23. The choice of explosives will depend on the wetness of the material to be blasted but, on the basis of 70% dry holes and 30% holes that are too wet to allow ammonium nitrate explosives to be used, the overall explosives cost would be 11¢ per yd^3, including blasting accessories. Whenever possible ANFO blasting agent would be used.

TRANSPORT

24. Plate 24 shows the mean haulage distance for the removal of the three types of material over the life of the project.

25. The superficials will be moved by scraper and during the first six years of operation would deliver material directly to:-

(i) the valley area north of the mine, ie the proposed plant location; 12 million bank yd^3 would be required to fill this area to an elevation of 2,800 ft.

(ii) the valley area to the west of the proposed plant area.

MUD FLOWS

26. In the final pit area there are two areas of colluvial soil, one on the east and one on the west side. These are either active or inactive mud flows, and are shown on Plate 16. The east mud flow contains an estimated 29 million bank yd^3 of material, 12 million bank yd^3 within the final pit area. The west mud flow contains an estimated 22 million bank yd^3, 13 million bank yd^3 within the final pit area. A depth of 50 ft is assumed in obtaining these volumes although this has not been confirmed. The volumes of material within the final area boundary are included in the production schedule given in Table III.

27. Inactive mud flows can be excavated in a scheduled manner with the glacial till, etc. An active mud flow poses a more difficult, practical problem and the following points are relevant:-

(i) Drains must be made at the start of the operations at right-angles to the direction of flow. These should be at 200-ft intervals. The most important drain will be that at the highest elevation near to the Aleece Lake. If these drains are effective, the rate of movement of the mud flow should be greatly reduced. Due regard must be had to the ground water hydrology, as determined in the geotechnical investigations, and recharge areas.

(ii) Any area of active mud flow near to the incline must be excavated as soon as possible and the dump contained by the retaining wall mentioned in para 8. If possible the main slip surface should not be under cut.

(iii) The mud flow must be excavated so the the angle of the flow is as low as possible. Without moving to the top of the mud flow this can be only a temporary expedient but should ensure that any rapid movement can be absorbed by the in-situ dump base.

(iv) Any active mud flow must be cleared within 100-ft of a working area. If the rate of movement were excessive it would be necessary to build a temporary retaining wall of large rocks and stable material and then excavate from behind this wall.

DRAINAGE AND PUMPING

28. There are likely to be four sources of water in the working area:-

(i) drainage from the surrounding hills and particularly from the active mud flow and waste dumps

(ii) seepage through the Hat Creek diversion dam

(iii) Seepage from the surrounding strata

(iv) natural precipitation.

The effects of (i) can be reduced by strategically-placed drains leading the water into the diversion reservoir. (ii) and (iii) can be reduced but ultimately some of this water will reach the pit. (iv) will affect (i), (ii) and (iii) but, having acted as described above to reduce the effect of any rain that falls in the surrounding area, any water falling directly over the open pit will require to be pumped.

29. Little meteorological information is available for the area but observations taken at Lehmann Ranch in the Upper Hat Creek valley give an average precipitation of 11.9 in per year. Using this figure, the quantity of water to be dealt with from this source, when the pit is as its maximum diameter, would be an average of 950 Imperial gallons/minute over the year, making no allowance for evaporation or seepage.

30. When the pit is down to the 2,400-ft elevation it may be necessary to pump the water in two lifts.

MOBILE MINING EQUIPMENT

31. Table IV details the mobile mining equipment required during the first 34 years of the pit operation. The actual working period for any machine for a year is taken as 5,000 hours. This is calculated on the following basis:-

Total hours x managerial efficiency x operational efficiency x machine availability

ie 8,400 x 90% x 88% x 75% = 5,000 hours.

32. Plates 25 to 28 inclusive show the hours of work for each item of equipment and the machinery required over the first 34 years. The capital and replacement costs for all this equipment are summarised by stages and years in Table V. Under the conditions to be expected at Hat Creek, replacement of the equipment is considered to be necessary as shown below:-

(i) Every ten years - shovels
pumps

(ii) Every four years - water trucks
diesel trucks
maintenance vehicles
pick-up trucks
explosives trucks

(iii) Every two years - bulldozers
wheeldozers
trucks
graders
compactors
drill and compressors
scrapers

33. In general these lives are in line with manufacturers' recommendations. However, the off-highway trucks are rated at 10,000 to 20,000 operating hours. The lower figure has been adopted because of the poor trafficability expected and the short haul. If the higher figure does not result in excessive maintenance, the cost saving would be about 10¢/ton. If the lives of the other equipment could similarly be increased by 50% a further saving of 10¢/ton could be realised.

34. A schedule of typical equipment is shown in Table VI. (The use of manufacturers' names is for illustration only and is not to be regarded as an endorsement of their products). The schedule of equipment includes allowances for miscellaneous work. This particularly includes surfacing and repair work on roads on the benches and incline. Graders and trucks are required for this and the number of truck hours allowed for miscellaneous work for each year of stage 1 was approximately 2,000 hours, for stages 2 and 3, 8,000 hours/year and subsequently 10,000 hours/year.

EQUIPMENT COSTS

35. Table V shows the cost of pit equipment and includes the initial purchase and replacement prices at 1975 values. Table VII shows the allocation of equipment costs over activity and type of machinery.

COAL REMOVAL

Transport Out of Pit

36. The ultimate pit-bottom elevation will be at 2,400 ft and the inclined access road will have an overall length of 5,250 ft with a slope of 1 in 15 against the load.

37. The amount of coal to be transported will reach a maximum of about 40,000 tons/day with peaks on the transport system of 2,500 tons/hour. To transport this amount of coal out of the mine at such a rate will demand a reliable and economic system, unaffected by climatic conditions and sub-zero temperatures. Alternative methods of transport which have been considered are as follows:-

(i) pneumatic pipeline

(ii) dump truck

(iii) belt conveyor.

Transport by Pneumatic Pipeline

38. When transporting small tonnages over medium and short distances, the pneumatic method has proved quite efficient and economical but over the envisaged distances, lifts and tonnages, the capital and operating costs become prohibitive. As time progresses and further experience is gained, undoubtedly the tonnages which can be handled pneumatically in single systems will increase. At the present time, however, this method cannot be considered economically feasible for the Hat Creek project.

Transport by Dump Truck

39. Over reasonably short haul distances on the level or gradients in favour of the load, transportation of mineral by dump truck is an acceptable and economic method and most flexible in operation.

40. However, over long distances, with large tonnage rates and with gradients against the load, the cost of transport per ton of coal rises appreciably. Long low-gear trucking up inclines is environmentally unfavourable due to dust, fumes and noise.

41. Comparing the economics of dump truck against belt conveyor transport of coal out of the pit, and using the aforementioned rates of production, haul distances and gradients against the load as a basis for calculation, the results were firmly in favour of belt conveyor installations.

42. Resulting from the cost analysis made between the two systems, the conveyor belt system offered a cost advantage of 35¢/ton of coal transported. Furthermore the presence of slaking claystones in the pit will make haul-roads expensive and difficult to maintain. Therefore dump trucks have not been considered for the out-of-pit coal transport system.

Transport by Conveyor Belt

43. Apart from the economic advantage, a belt conveyor system is far more suited to long hauls with gradients against the load, as will be the case at Hat Creek. For this reason, the whole of the out-of-pit coal transport has been based on belt conveyors working in conjunction with in-pit feeder/breaker/conveyor transfer stations (Plate 29).

44. The ultimate pit will have three of these in-pit transfer stations and three main incline conveyors which, together, will be used for coal and/or waste transport out of the pit to the main interchange station.

In-Pit Transfer Stations

45. The in-pit transfer station (Plate 29) will comprise:-

(i) A ground hopper, 20 ft x 20 ft, sited on a working bench and with a static grizzly having 2 ft x 2 ft square apertures. The undersize coal from the grizzly will pass through to a breaker-type feeder and any oversize will be reduced using a hydraulic pick-type breaker. Any boulders encountered will be separated and trucked out of the pit and used for road building and spoil dump stabilisation.

(ii) A breaker-type feeder, 78-in wide, capable of accepting any material passing from the rear dump trucks via the grizzly, and then reducing it to 15 in to 0. The coal will then be fed to a 60-in cross conveyor.

(iii) A 60-in cross conveyor with a peak capacity of 2,500 tons/hour and approximately 300-ft long, will transport the 15 in to 0 size coal to one of the three main incline accelerating conveyors, via a system of change-over chutes.

(iv) Three accelerating conveyors with a capacity of 2,500 tons/hour which will deliver the 15 in to 0 size coal on to one of the main incline conveyors.

Main In-Pit Incline Conveyors

46. There will be three main incline conveyors suitable for the transportation of the rom coal and waste out of the pit. These will be located on the main incline in the arrangement indicated on Plate 19.

47. For the ultimate pit, the three conveyors will be approximately 6,000-ft long, 60-in wide and each with a capacity of 2,500 tons/hour.

48. Coal will be collected by one, two or all of these conveyors from the in-pit breaker/feeder/conveyor transfer stations and transported to the surface interchange station.

49. Three conveyors have been proposed for the project to allow for one to be out of production for extension, maintenance, etc, and also to take care of peaks caused by slides, etc which would necessitate stepped-up, out-of-pit transport.

50. The third conveyor has been taken into account in the cost analysis of trucks v conveyors, ie $1\frac{1}{2}$ conveyors have been allotted to the transport of coal.

SPONTANEOUS COMBUSTION AND FIRE PREVENTION

51. Hat Creek coal has not been exposed to the atmosphere in significant amounts and therefore its propensity to spontaneous combustion both in the mine and in stockpiles needs to be investigated.

52. In the mine, the behaviour of exposed coal faces and blasted coal will need to be observed and, if necessary, fire prevention measures adopted. Blasting should be restricted to the quantity of coal required for immediate loading. If heating develops in the solid coal it would be dug out or blanketed with inert material.

53. Stockpiles present the greater fire hazard but well-recognised methods are available to mitigate this, ie size control, compaction, etc. In any event systematic temperature measurement is recommended so that any heating can be detected early and dealt with. At all times access must be maintained around the stockpile.

54. The actual cost of fire prevention is not substantial; the watch-words are vigilance and prompt action.

CHAPTER V

SURFACE PLANT AND COAL PREPARATION

INTRODUCTION

1. Aspects related to the mine surface plant and coal handling and preparation are discussed in this chapter. The uncertain nature of the coal within the deposit and its probable variation in quality necessitates careful consideration of coal handling arrangements and preparation.

2. Irrespective of the particular equipment used, it will be necessary to develop a mine planning and scheduling system whereby the quality of the rom product can be controlled in line with demand or at least forecast. In this respect, from the mining economics aspect, it is advisable to arrange the planning so that the removal pattern of coal and waste is optimised. Policy with regard to coal quality cannot, of course, be finally decided without serious consideration of the users' quality requirements and the degree and type of preparation required.

STOCKPILING AND RECOVERY

3. During the initial opening up of the pit, and prior to the commissioning of the first 750-MW generator in 1982, a stockpile of coal will need to be established.

4. The amount of coal mined up to the time of start-up of the first generator unit will be in the region of 700,000 tons. This quantity will be stockpiled and consolidated to prevent spontaneous combustion occurring within the piles.

5. The equipment that will be required to stockpile the initial coal production will be part of the final installation (Plate 30), with an additional temporary ground hopper, grizzly and conveyor to feed the secondary crusher. This temporary ground hopper installation will receive coal from rear-dump trucks and transport it to the stockpile via the static screen, crusher, cross conveyor, stocking conveyor and boom stacker.

6. To effect consolidation of the stockpile and to achieve some degree of blending, the stockpiles will be laid down in layers and consolidated by a combination of bulldozer and compactor.

7. This method of stocking will continue until the commissioning of the first 750-MW generating unit, at which time the second stage of the stockpiling equipment and the first stage of the coal recovery equipment will be brought into operation.

8. The temporary ground hopper and grizzly installation will be eliminated and coal will then be transported from the mine to the secondary crushers via the main incline conveyors "f" and "g" (Plate 30). The final stocking and recovery arrangement will include three cross conveyors from the crusher station feeding to the three stacking conveyors and boom stackers, and two recovery conveyors with two bucket-wheel reclaimers. The recovery conveyors will transport the $1\frac{1}{4}$ in to 0 coal from the stockpile via the bucket-wheel excavators to the power plant silo-feed conveyors.

9. For extreme emergency conditions, a ground hopper and conveyor system will be provided to transport coal from the stockpile to the silo-feed conveyors. The stockpile-consolidating bulldozer will be used for moving the coal from the pile to the ground-hopper grizzly.

10. All of the coal stocking and recovery conveyors from the interchange station to the silo-feed conveyors will be 60-in wide and capable of transporting a maximum of 2,500 tons/hour each. The secondary crushers will be rated at 2,000 tons/hour and capable of reducing the rom coal from 15 in maximum to $1\frac{1}{4}$ in to 0.

11. Plate 31 indicates the proposed coal stockpile and conveyor grid layout for the scheduled 2,000-MW power plant.

COAL PREPARATION

12. This important topic is the subject of a separate study.

COAL QUALITY

13. The existing borehole data provide general information against which coal quality can be estimated in a global sense, ie for the entire deposit as presently delineated. The drilling of the in-fill boreholes previously recommended, however, will enable much more reliable estimates of quality and quantity to be made against which the quality of rom coal can be controlled in a reasonably consistent way. Some preliminary considerations are given below.

14. The first requirement is a means of expressing the quality parameters in terms of mining blocks. This is to facilitate detailed mine planning in order to predict output in terms of coal quality and the types and quantity of waste. To this end, the whole deposit should be divided into a three-dimensional matrix, such that each horizontal layer of blocks corresponds to a mining level in the proposed openpit. The block height is then equivalent to the bench height and the extraction process can be directly related therefore to the quality of product sent to the blending stockpile.

15. These requirements lead to the provision of a spatially indexed "mineral inventory". This lists the blocks together with their coal quality and/or waste content. In order to assign the relevant values to the blocks, it will be necessary to align the sampling data from existing and future boreholes with the proposed mining levels.

Methods of Assigning Values to Blocks

16. The means of evaluation that have been considered include conventional statistical analysis, trend surface and moving average techniques, "geostatistical" methods and simple interpolation procedures.

17. In the case of statistical analysis, unless at least one hole passes through each block, all that can be deduced is an estimate of the mean value for each horizon of blocks, together with a confidence interval around this value. It is not possible to give separate estimates for individual blocks within a mining level.

18. For the moving average, trend surface and geostatistical techniques to have much practical value, a closer hole spacing would be necessary than is available or immediately envisaged for the project.

19. The interpolation methods use relatively simple means of attributing a weighted value to an unknown block from the known values at neighbouring sample points. The system usually found most effective in practice uses the inverse of the squared distance from these sampling points as the weighting factor. Although this form of weighting is somewhat arbitrary, studies have shown the technique to be as efficient as geostatistical procedures in predicting block values. Bearing this in mind, and considering its relative simplicity, this last method is recommended for obtaining initial estimates of block values.

20. In the present case, the sample points are represented in the horizontal plane by drill holes. Since the sample values within each hole are given over length rather than at points, simple weighted averages over lengths corresponding to the block height must be obtained. These averages are then treated as point values in the horizontal plane through the centre of the blocks at that level. Barren holes near the boundaries of the deposit will also be included in the scheme.

21. A computational scheme is being considered for the method. The routine derived from this scheme will be largely automatic in the sense that the only input required is the borehole data and the major pit parameters. The block coordinates and weighted values are then obtained by the computer.

Quality Control Methods

22. The object is to provide means of predicting and monitoring the quality of coal delivered to the power station. At this stage, it is only possible to use the available data for broad predictive measures. However, in the immediate pre-production and early production phases, additional sampling holes and blast-hole sampling can be used to forecast and control the output processes with greater precision.

23. Assuming that the process "generating" the sample values can be demonstrated as being under "statistical control", the relative frequencies of quality variates in the sampling distribution can be used as probabilities in predicting output (from mine and stockpile) in terms of these variates. In addition, control limits can be established to provide warning if the production/blending system is tending to go out of control. A variety of control charts, each with special characteristics, can be constructed for this purpose.

24. The usual control chart methods assume that the form of the sampling distribution, or distributions, is known. This can be determined from the sampling data by standard methods. However, even if nothing is known about the form of the distribution, procedures are available to predict limits on the variability.

25. With respect to the present data, and those accruing from the proposed drilling, the above procedures must be considered as providing somewhat gross measures when applied to short-term output. On a day-to-day basis, and especially in the early stages, production for blending will be from spatially limited areas. The only narrow predictive guide will be that given by the block values generated using the inverse-distance-squared method. Therefore, these values must provide the initial estimates of expected quality of output for short-term mining sequences, with the statistical measures giving broad limits for the quality variability. The information obtained by these methods will enable a production-scheduling program/blending-requirement program balance to be attained.

MINE POWER SUPPLY

26. It is envisaged that the power supply for the mine will be taken initially from an overhead line following the route of the Lillooet - Cache Creek road, at a point near the junction with the Hat Creek road. However, after start-up of the first power station generator, the supply would be taken from the power station.

27. A conceptual distribution system is shown on Plate 32. This basically consists of a duplicate feeder cable system down the main incline, with bench cables teed off as required, and a surface substation to supply all surface loads. Owing to the proximity of the power station, fault capacity should be carefully considered and reactors may well be necessary to reduce it to acceptable levels in the openpit. Because of

the need to maintain continuous production, the whole system should be engineered to the highest standards. The conveyors would, of course, be fitted with sequence control and routing system, the control station being located near the top of the incline. It should also be fully monitored and protected. Telephone and radio communications from all parts of the mine with the control station would be necessary.

28. Table VIII gives an estimate of the energy consumption per year, the maximum demand and the cost at 10 mills/kWh. An overall diversity factor of 0.75 has been applied to the maximum demand total to obtain the system maximum demand. Table IX is a summary of the electrical energy costs.

SCHEDULE OF FIXED INSTALLATIONS

29. Table X is the schedule of equipment (fixed installations) and the capital costs are given by years as the installations develop. Initial spares, provisionally estimated at 10%, have been included. It has been assumed (pessimistically) that the equipment would be replaced every 10 years. Depreciation has also been calculated at 10% of the capital cost.

CHAPTER VI

WASTE AND ASH DISPOSAL

MATERIALS AND QUANTITY

1. There are four types of waste to be dumped in areas near the No 1 deposit. These are:-

(i) Superficials

(ii) Pit waste (claystone)

(iii) Segregated waste separated visually from the coal (claystone, shales etc)

(iv) Ash from the power station (conditioned to 15% moisture content).

2. The quantities produced at each stage of the pit life are shown in Table XI. Specifically 1,267 million yd^3 of space are required up to the end of the 600-ft pit and 2,447 million yd^3 are needed to the end of the 1,500-ft pit. These quantities assume swell factors as shown in Table I which require verification.

DUMPS

3. Examination of the topography of the valley has shown four main areas suitable for dumps, in addition to the valley immediately at the head of the conveyor incline.

4. In calculating the space available the following criteria have been used:-

(i) the surface of the dumps should not be above 4,000-ft elevation

(ii) the maximum slope of the dump face should not exceed 24°

(iii) using a 25-ft wide bench and a 50-ft face, the overall slope would be 20°

(iv) all dumps would be founded on a layer of granular material to permit drainage

(v) each face would have a toe drain to lead water from that section to the main drain

(vi) where the waste materials are of low strength so that slope failure is a hazard (eg mud-slide material, weathered claystones) a retaining dam would be constructed of stable granular material, boulders, etc. However, considerable attention must be paid to the detailed location and design of these massive structures.

5. In positioning the dumps the following have been considered:-

(i) the slope angle of the dumps must be safe and permit restoration easily

(ii) the dumps must be as near to No 1 deposit as possible

(iii) the dumps must not intrude into possible areas required for the working of No 1 or No 2 deposits down to 1,500-ft elevation

(iv) dump heights must be minimised

(v) dumps placed over streams or main drainage areas would be designed so that water flow could continue

(vi) if possible, useful superficial material should be stacked separately.

6. The angle of slopes of all the waste dumps must, of course, be determined by further geotechnical investigation of the materials involved and the final shapes suitably contoured. In general, the better material will be used to confine the poor material. Drainage of water into the dumps from above and out of the dumps at the toes must be carefully controlled. Also, due consideration will need to be given to the encouragement of re-vegetation by top-soiling, etc. Further study in co-operation with the environmental consultant is in progress and this co-operation should continue throughout mining operations until the land is restored to the required standard.

DISPOSAL AREAS

7. In line with these criteria, Table XII shows the total dumping space available to elevations up to 4,000 ft, Plate 33 shows areas required to dump material from the 600-ft pit and Plate 34 shows those required for material from the 1,500-ft pit.

600-ft Pit

8. In addition to filling the valley at the top of the incline, it is planned to dump at the north end of the deposit. Dump No 1 can be used for superficial material from the east side of the deposit delivered directly by scrapers. The areas near the incline can also be designed to provide a stocking space for about 12 million tons of low-grade coal. The area north of the incline would hold 1 million tons of rom coal for the power station.

9. The volumes available, as shown on Plate 33, are:-

		10^6 yd^3
Valley dump	(to 2,800-ft elevation)	17
Dump No 1	(to 3,400-ft elevation)	132
Dump No 2	(to 3,800-ft elevation)	1,156
		1,305

(Volume required 1,267 million yd^3)

1,500-ft Pit

10. To accommodate the extra waste material produced by mining down to the 1,500-ft elevation two more dumping areas are required; these are shown on Plate 34. Using these dumps two alternative methods of dumping are available:-

(i) To dump material so that all dumps reach the same elevation at the end of the 1,500-ft pit.

In this case the dump volumes are:-

		10^6 yd^3
Valley dump	(to 2,800-ft elevation)	17
Dump No 1	(to 3,900-ft elevation)	319
Dump No 2	(to 3,900-ft elevation)	1,355
Dump No 3	(to 3,800-ft elevation)	59
Dump No 4	(to 3,900-ft elevation)	773
		2,523

(Volume required 2,447 million yd^3)

Dump No 3 may not be needed but would be convenient for direct dumping of superficials from the west side of the deposit.

(ii) To dump material so that the northern dumps No 1 and 2 are at the maximum assumed elevation of 4,000 ft, placing the minimum quantities in dumps No 3 and 4.

In this case the dump volumes are:-

		10^6 yd^3
Valley dump	(to 2,800-ft elevation)	17
Dump No 1	(to 4,000-ft elevation)	376
Dump No 2	(to 4,000-ft elevation)	1,571
Dump No 3	(to 3,700-ft elevation)	32
Dump No 4	(to 3,800-ft elevation)	529
		2,525

(Volume required 2,447 million yd^3)

Similarly, No 3 dump space may not be required.

11. If the mining of Area 2 is taken into account it is clear that alternative (ii) leaves the maximum dumping space close to that deposit

12. During stage 9 for dumping of waste at the south an additional incline would be needed on the south side of the pit.

13. Allowance has been made in the costing for machinery to compact the dumps but no account has been taken of the reduced volume of dumping space required due to compaction. It is thought likely that there would be a reduction in waste volume of between 10 and 15% on account of mechanised and self compaction. Optimisation of the mining methods may suggest a dumping pattern which modifies the above but it is considered that the changes would be minor.

METHODS OF WASTE TRANSPORT

14. The in-pit waste and segregated waste would be transported by trucks to the incline conveyor. After the incline the waste would be distributed to the dumps by conveyors and waste spreaders.

15. Superficials would be delivered directly to the dump area or to a ground hopper and from there to the waste conveyors for distribution to the dumps. Ash would be delivered to the waste disposal system by conveyor from the power station after conditioning with water. The conditioned ash would be added to one of the waste systems for distribution. As some ash would be needed for road-surfacing provision would be made for ash to be loaded into trucks directly from a hopper.

16. On the dumps the waste would be spread by crawler-mounted boom spreaders which would either place the waste in front of the conveyor (below its level) or behind the conveyor (above its level), the height of the dumps being determined by slope stability considerations and, in the case of the rear dump, by the maximum dumping height of the machine.

CHAPTER VII

INFRASTRUCTURE AND CIVIL WORKS

HAT CREEK DIVERSION

Object

1. The object of the diversion is to prevent ingress of stream flow into the pit where it would cause unacceptable damage and interference with mining operations. It is proposed that this work be done in 1977 before any stripping begins, otherwise temporary diversions will have to be made.

Data

2. Data on stream flow are available from "Historical Stream Flow Summary - British Columbia" which gives records over a period of 10 years with a maximum recorded peak flow of 517 ft^3/sec. Monthly precipitation data are available over the past 15 years. No information exists, however, on the intensity of short-duration rainfall or the flood peaks resulting therefrom.

Diversion Alternatives

3. Although the upper limits of the pit have not been clearly determined, it is probable that these will extend to an elevation of 3,350 ft on the east side and 3,550 ft on the west side at pit stage 8. A tunnel diversion has not been considered at this stage due to lack of geological data. For a surface diversion, the east side of the pit offers a preferable route due to shorter distance and lower elevation. A gravity flow diversion of the major part of the flow in Hat Creek is theoretically possible but would require a diversion point approximately 3 miles upstream of pit No 1. Because of the distance involved, the size of channel or conduit required and the difficulties which would be caused by the steep valley sides, this alternative has not been considered further at this time. Diversion further upstream, with flood irrigation as a potential benefit, may be preferable if economics permit. The whole subject will be studied further as part of the Conceptual Design Study of the Proposed Hat Creek Development.

Hydrology

4. Consideration of any form of diversion necessitates determination of the design flood and the optimisation of storage capacity versus diversion conduit capacity. The design flood must be evaluated in relation to the available hydrological data and to estimated costs of damage resulting from floods in excess of the design value. In this case, it is considered that the cost of designing for the probable maximum flood would not be justified and that a small degree of risk of overflow into the pit can be accepted, provided measures are taken to minimise the resulting damage.

5. A flood-return period of 50 years has therefore been adopted and a hypothetical flood hydrograph has been constructed, based on available data and using hydrograph procedure and a calculated peak discharge of 1,150 ft^3/sec. A base flow of 50 ft^3/sec has been added to the flood hydrograph and mass curves produced in order to assess optimum storage capacity and pump discharge rates. Prior to determination of the final design concept, it is recommended that further data be collected, including continuous stream flow and precipitation measurements.

Diversion Scheme

6. The selected scheme (Plate 16) allows for an earthfill dam with a maximum height of 55 ft and a crest elevation of 3,030 ft, located approximately 800 ft downstream from the confluence with Medicine Creek. Maximum pond surface elevation is 3,025 ft and the southern extremity 2,500 ft from the dam face. A pumping station with 5,000 installed horsepower would be located at the dam with twin 30-in diameter steel pipelines, 3,200-ft long, discharging a maximum of 100 ft^3/sec at an elevation of 3,400 ft into a lined, open channel, 11,000-ft long. The channel would follow the natural contours and discharge into the small creek which flows from Harry Lake to Hat Creek. During normal flow conditions, the water surface in the reservoir would be kept at elevation 2,990 ft and would provide approximately 850 acre-feet of reserve flood-storage capacity.

7. The dam would be provided with an emergency spillway discharging into the existing channel of Hat Creek. At the limits of the pit, the flow would enter a culvert and discharge at the bottom of the pit where it could be dealt with safely by the pit drainage system.

8. The cost of the diversion scheme and the various component items is detailed in Table XIII.

SURFACE DRAINAGE

9. A minor diversion of Finney Creek will be necessary to convey flow into the flood storage reservoir. No other creeks flow directly into the pit but it will be necessary to construct drainage ditches around the limits of the pit in order to divert surface flow to the Hat Creek channel. The area draining to these ditches is comparatively small, the costs being shown in Table XIV.

ROAD DIVERSION

10. The existing road running north-south and parallel to Hat Creek would be diverted around the eastern perimeter for a distance of 15,000 ft. The diversion (Plate 16) would have a width of 30 ft and a limiting gradient of 10%. The estimated cost is shown in Table XV.

SURFACE MINE BUILDINGS

11. The surface buildings associated directly with the mining operation would be typical of most Canadian open-pit mines. A listing and brief function of each building, the costs of which are shown in Table XVI, follows:-

(i) Maintenance and Warehouse Building

This building should accommodate the major workshops and warehouse for the mine (Plate 36). Space for repair bays would be adequate for 150-ton, off-highway trucks and tracked vehicles. Offices for maintenance staff are included in this building.

(ii) Change House

The change house (Plate 37) is designed to accommodate approximately 250 personnel per shift. This building would also contain offices for shift bosses and general foremen.

(iii) Administration Building

Staff personnel, including senior management, engineering, geology, accounting and general administration would be located here (Plate 38).

(iv) Core Sheds

Drilling and sampling of coal will require approximately one core shed per year. A standard design as presently used on the site is proposed (Plate 39).

(v) Powder Magazine

Powder storage and handling facilities are allowed for by the provision of a magazine.

ROAD CONSTRUCTION AND ROAD IMPROVEMENT

12. The existing road to the area of Pit No 1 must be improved and bridges or culverts installed where required. It is proposed that this road be widened and paved, also the mine buildings area be paved.

13. An allowance of $860,000 is made for widening and paving approximately 5 miles of road, plus yard area and installation of miscellaneous culverts.

SERVICES

14. Services such as power and water supply, sewage disposal and miscellaneous mobile equipment, are provided for in Table XVII, Schedule of Equipment - Infrastructure, where an allowance of $1,637,000 has been made.

HOUSING

15. The demand for housing will depend on the build-up of the labour force as shown in Tables XVIII and XIX.

16. Stage 1, 1977 to 1982, will consist primarily of construction work and pre-production mining of overburden, for which the maximum personnel is estimated at 126. Stage 2, 1983 to 1984, and stage 3, 1985 to 1987, will build up to the initial and full production manpower requirements and these are estimated at 260 and 628 respectively for the two stages. As the waste stripping requirements increase, additional labour is required with an eventual total labour force of about 1,400 at the completion of mining to the 2,400-ft elevation.

17. During the pre-production stage, it is assumed that contractors' employees will be housed in suitable trailer camps provided by the contracting companies. Typical camp layouts are shown on Plates 40 and 41. Once the production of coal commences, it is naturally assumed that a more stabilised labour force will develop. In order to attract and retain the right type of workman, the development of permanent housing in existing communities will be required. By stage 3, 1985 to 1987, it is expected that trailer residents will be re-located to permanent housing. Typical housing costs are contained in the schedule given in Table XVI.

SCHEDULE OF INFRASTRUCTURE

18. Table XVII lists the infrastructure requirements and the capital costs are given by years as the project develops. Allowances have been made for initial spares and for depreciation. Employee housing costs are shown below the line because, assuming no element of subsidy, this item would not be chargeable to the mine. Therefore, it is not included in the production cost of coal. However, the housing, both temporary and permanent, has to be financed. Thereafter, maintenance and capital charges are assumed to be covered by rent.

CHAPTER VIII

ECONOMICS

BASIS

1. The following steps are necessary to derive the economic results of mining the coal at Hat Creek in Openpit No 1 down to the 2,400-ft elevation:-

(i) Capital cost estimates - these arise in the foregoing chapters from the schedules of equipment for each aspect of the project. Capital charges are also derived from these figures.

(ii) Direct operating cost estimates, ie labour, energy and materials - these are derived from the production schedules of coal and waste and the schedules of equipment.

(iii) Cumulative capital expenditure and interest during construction.

(iv) Production cost in each year - capital charges and direct operating cost.

(v) Cash flow and discounted cash flow - these calculations enable one uniform selling price to be calculated (at constant prices) to give a specified DCF rate of return.

(vi) The calculations are repeated for inflated prices.

Basic Financial Data

2. Appendix "D" reproduces the basic financial data provided by BCH (letter dated 29th October, 1975). All costs have been expressed in 1975 Canadian dollars but inflated costs are also given. As regards the rate of inflation, it has been assumed that capital goods and electrical energy will have similar inflation rates to labour and materials.

3. The following additional assumptions have been made:-

(i) Import duty for mining equipment, 15%.

(ii) Local and municipal taxes not included.

(iii) Cost of land, wayleaves, compensation, etc not included.

(iv) Legal costs not included.

(v) Inflated costs have been calculated on a "revaluation of assets" basis rather than a "historic cost" basis so that depreciation and interest can be directly inflated.

Explanation of Tables

Capital Costs

4. Table V, Schedule of Mobile Mining Equipment, gives the numbers and cost of each item of equipment, scheduled according to the production requirements given in Table III. Replacements have been included when the item reaches the end of its normal life. Different stages in the development and operation of Openpit No 1 are indicated. Initial spares have been included as these are capital items, as distinct from subsequent consumable spares. The depreciation of mobile mining equipment, calculated on an average basis weighted approximately according to the life of each item of equipment, is included in Table XXII.

5. Tables X and XVII deal with fixed installations and infrastructure in the same way. It should be noted that many of the fixed installations, eg coal stockpiling and reclaiming and also ash handling, are strictly not part of the mine but could be considered to be part of the power plant. The capital cost of this equipment is about $34 million.

Direct Operating Costs

6. Table VIII, Electric Power - Consumption and Costs, and Table IX, Summary of Electrical Energy Costs, provide the basis on which the cost of electrical energy consumption of the whole project is derived, based on a rate of 10 mill/kWh.

7. Table XVIII, Labour Schedule and Payroll Costs, and Table XIX, Managerial, Technical and Administrative Staff Costs, develop the number of labour and staff and the yearly costs. Fringe benefits have been included at 21% in Table XVIII and at 25% in Table XIX. An allowance of 5% has been added to the labour payroll costs to cover overtime. Further comments on labour aspects are given in Appendix "E".

8. Table XX, Materials and Fuel Cost Summary, estimates the costs of consumable stores and spare parts, diesel oil, gasolene, lubricants, etc.

9. Table XXI, Direct Operating Cost Summary, assembles together labour, materials and energy costs and, in addition, an allowance is made for Consultants' fees in view of the serious unresolved problems which will still require solution as the pit develops.

10. In all these tables the cost per short ton of rom coal and per million Btu is shown for information.

Total Investment and Capital Charges

11. Table XXII, Depreciation Summary, assembles the depreciation estimates derived in Tables V, X and XVII, ie for mobile mining equipment, fixed installations and infrastructure. In addition, other capitalised costs have been depreciated over the life of the 2,400-ft pit.

12. Table XXIII, Capital Investment, Interest during Construction, Interest and Insurance, takes the yearly capital costs from Tables V, X and XVII and adds other cash items which have to be capitalised prior to start-up. It is tentatively assumed that, after start-up, the cash flow will be positive. These items include direct operating costs (from Table XXI), insurance and working capital. The latter is estimated in total at 3 months direct-operating costs at the full production stage.

13. The BCH corporate overhead is then added to the resulting capital cost and the figures obtained are accumulated in order to calculate interest during construction (IDC) according to Appendix "D", Plate 42. (Clearly this applies only up to date of start-up). The grand total, including IDC, is referred to as "total investment". This is estimated at about $134 million to start-up, of which $34 million, ie 25%, could be chargeable to the power plant (see para 5 above).

14. In order to calculate the yearly interest, the average investment outstanding in each year has first to be calculated. This is the difference between the cumulative total investment and the cumulative depreciation, the average being the mean of the amounts at the beginning and the end of each year.

15. Insurance is assumed to be 2% of the average investment outstanding in each year.

Production Cost (1975 Prices) - 600-ft Pit

16. Table XXIV shows the development of the production cost as it would be calculated in normal accounts, ie direct-operating costs, depreciation, interest, insurance and royalty. (Although the production cost is expressed in terms of run-of-mine coal, it is not the mine-mouth cost but the delivered cost into the power plant bunkers, allowing also for ash disposal - see para 5 and 13 above. Coal handling and ash disposal costs are about 80¢/ton rom.) Since interest at 10% on outstanding capital is included and all capital is

regarded as loan capital, the production cost is equal to the price at which the coal could be sold on a break-even basis. This figure is different in each year because the pit is not a static entity - the production of waste must vary year by year and the shape of the pit changes gradually, although coal production remains constant after the build-up stage until the run-down stage. It will be noted that the production cost during stage 8 at $9.68/ton rom is not substantially greater than that at the beginning of stage 2, ie $8.74/ton rom. However, the cost at stage 8 is considerably higher than the cost at the intervening stages. This reflects the expensive extension of the pit laterally, although it could be extended in depth at the same time with additional coal production (and reduced costs) if the stability of the pit could be assured.

Production Cost (1975 Prices) - 1,500-ft Pit

17. As mentioned in Chapter III, para 22, a pit down to the 1,500-ft elevation (1,500-ft pit) has been postulated. The instantaneous stripping ratio at the probable limit of that pit is 13.7 bank yd^3/short ton rom. Since production cost is, broadly-speaking, in linear relationship with stripping ratio (assuming no changes in the mining system), the approximate production cost at the probable limit of the 1,500-ft pit can be obtained by extrapolation. Plate 43 is a plot of the production cost for each stage against the instantaneous stripping ratio at the end of the stage and the linear relationships can be seen. Extrapolation after stage 8, the probable limit of the 600-ft pit, to a ratio of 13.7, the probable limit of the 1,500-ft pit, results in a production cost of about $14 per ton, so that the additional coal between the 600-ft pit and the 1,500-ft pit could be mined at a production cost of between $10 and $14 per ton.

Discounted Cash Flow (1975 Prices)

18. Table XXV shows the cash flow of the expenses and the calculation of a constant uniform selling price which would yield an internal rate of return of 15%. (This discount rate was specified by BC Hydro - see Appendix "D"). The cash flow includes all capital and direct-operating-cost elements and such cash expenses as insurance and royalty. The total cash flow is then discounted at 15% to obtain the present value of the expenses. By equating these to the present value of the revenue at a uniform coal price, that uniform price can be calculated, the formula being as follows:-

$$\text{Uniform Selling price} = \frac{\text{PV of expenses (\$)}}{\text{PV of coal production (tons)}} \ \$/\text{ton}$$

19. In other words, if all the coal were sold at the uniform selling price throughout the life of the project, the internal rate of return on all the investments would be 15%. Plate 44 shows this uniform selling price compared with the production cost. It will be noted that the uniform selling price calculated on this basis is $6.35/ton rom (58¢/10^6 Btu).

20. As requested, the calculation has been repeated at a discount rate of 10% and the uniform selling price recalculated (see Table XXV). This results in a price of \$5.63/ton rom (51¢/10^6 Btu). This result is also shown on Plate 44. It is directly comparable with the stream of production costs calculated in Table XXIV as in this case the discount rate (internal rate of return) can be equated to the interest rate because it has been assumed that <u>all</u> the capital is borrowed.

Confidence Limits of Estimated Selling Price

21. A pessimistic view has been taken in all areas of uncertainty in this conceptual mine design and in the economic calculations, ie the estimated production costs and selling prices can reasonably be regarded as a "maximum". It is expected that the uncertainty would be about ± 10% about the mean and therefore the "mean" and "minimum" figures can be calculated. Also, in order to effect true energy cost comparison, it is necessary to deduct coal handling and ash disposal costs estimated at 80¢/ton (see para 16) to arrive at the coal mine-mouth price. The resulting uniform selling prices and ranges for 10% and 15% discount rates are given below.

Coal Prices, \$/ton

Discount rate	10%	15%
Uniform selling price including coal handling and ash disposal	5.63	6.35
Coal handling and ash disposal costs	0.80	0.80
"Maximum" mine-mouth selling price	4.83	5.55
"Mean"	4.35	5.00
"Minimum"	3.90	4.50

Probable areas of cost savings include steeper slopes, less blasting and earlier economic cut-off (depending on availability of reserves).

Life of Openpit No 1

22. As mentioned in Chapter III, para 20, the reserves of coal available from the 600-ft pit are sufficient for 30 years of power station operation. For 35 years an additional 100 million tons would be required. If stability can be proved then a few deeper benches would easily yield this extra coal at reasonable economic cost. The effect on the uniform selling price as calculated above would be negligible because the last 5 years would be heavily discounted.

Production Cost (Inflated)

23. Table XXVI is a repetition of Table XXIV except that the various cost elements have been inflated according to the rules given in Appendix "D" and the additional assumptions which have been made (see para 3). The effect of the "revaluation of assets" basis is that the "real" value of the capital investment would be recovered rather than the "historic" value. This method clearly gives higher values than the "historic" cost basis. Royalty has not been inflated but it would very likely be adjusted from time to time by political decision. According to this calculation the production cost increases from $15.04 per short ton rom in stage 2 to $65.26 in stage 8 (years 2011 to 2013).

Discounted Cash Flow (Inflated)

24. Table XXVII is a repetition of Table XXV, inflated in the same way as Table XXVI. In this case, depreciation and interest do not appear so that the distinction between "historic" value and "real" value is irrelevant. It gives a "true" figure for 15% internal rate of return (discount rate). Another method commonly used is to inflate the discount rate, eg for a 5% per annum inflation rate the discount rate would be 20%. This is more difficult to apply when the rate of inflation is not constant.

25. It will be noted that on this basis the uniform selling price over the life of the mine becomes $13.52 (123¢/$10^6$ Btu) or just over twice the uninflated figure of $6.35 (58¢/$10^6$ Btu).

Opportunity Value of Hat Creek Coal

26. The economics of any mining operation depend on the market value of the product. This concept of market value has little meaning for a captive mine supplying a power plant and the criterion then becomes that the mine product must not be more costly than any alternative fuel for an equivalent energy content. (This simple concept ignores political factors such as the use of indigenous resources, employment, trade balance, forward and backward linkages, etc).

27. Since the mid-fifties, oil has been the most important international fossil fuel and has acted as the price leader against which others have been judged. Coal is not as convenient a fuel as oil or natural gas. It is dirty, more difficult to handle, requires expensive storage arrangements and needs special equipment to minimise atmospheric pollution. To allow for this, an arbitrary deduction of 20% compared with the value on a straight head-constant basis has been made. Thus, to be competitive with oil, the delivered cost of coal should not exceed 80% of the cost of an equivalent amount of oil.

28. The current international price of crude oil is about \$11/bbl. Heavy bunker oil, as burnt in power stations, can at times be marginally cheaper but this price relationship is likely to remain stable as most oil-fired stations can burn either crude or bunker oil.

29. This price of \$11/bbl is approximately equivalent to \$2/10^6 Btu. Thus, the maximum delivered price for coal will be equivalent to \$1.6/10^6 Btu, ie \$40/ton for standard bituminous coal or \$19/ton for 6,000 Btu/lb coal.

30. This picture is true internationally. There are, however, some modifications to this position in Canada. Firstly, the current domestic oil price is only \$8/bbl which will depress the other prices pro-rata, though it may be only a matter of time before prices rise to the international level. Secondly, \$2/10^6 Btu is well above the current Western Canadian price for natural gas, and natural gas is an even more convenient boiler fuel than oil. However, this price has recently increased and will increase further, and as supplies decrease there are likely to be restrictions imposed on its use for power generation.

31. Because of these uncertainties, opportunity values of coal and lignite have been calculated for a range of oil prices and these are shown graphically on Plate 45.

32. It must be emphasised that these are prices "as delivered" to the customer. In a country the size of Canada, coal transportation costs from mine to customer can exceed the mining costs. This is not a problem at Hat Creek where the power station will be adjacent to the mine. A price of \$6.35/ton will, therefore, be almost half the opportunity cost of coal, based on \$8/bbl of oil. This coal price is equivalent to 58¢/10^6 Btu (including about 7¢/10^6 Btu for coal and ash handling). Whilst sub-bituminous coals can be produced on the plains in Alberta at prices substantially lower than this (because of lower stripping ratios and easier mining conditions), delivery cif into the Vancouver area would increase the price to about 60¢/10^6 Btu. Typical cif prices in the Western US are over 70¢/10^6 Btu. Therefore, Hat Creek coal is competitive with other coals and oil.

Break-Even Stripping Ratio

33. The break-even stripping ratio (waste to coal) is that value of the stripping ratio at which the costs of stripping waste and loading coal exactly equal the value of the coal, the latter being either the opportunity value in comparison with alternative fuels or the total cost of production by alternative methods of mining. It is expressed by the following formula:-

$$\text{Break-even stripping ratio} = \frac{\text{value of coal/ton} - \text{coal production cost/ton}}{\text{stripping costs/yd}^3}$$

Plate 46 shows the break-even stripping ratios for various coal values against coal production cost. It will be seen that at a production cost of $6.35/ton and a coal value of $8/ton, the break-even stripping ratio is approximately 4.2 to 1. For the final cut-off ratio marginal costs should be used not total costs.

CHAPTER IX

SUMMARY AND RECOMMENDATIONS

GENERAL

1. The general geology has been determined but details are lacking at this stage and the borehole results are capable of different interpretations. This report adopts one interpretation for conceptual purposes and it is pointless at this stage to indulge in further speculation.

2. Specific geotechnical data are almost wholly absent but it can be said with confidence that slope stability will be a major problem. The working slope angle adopted may be considered over-pessimistic but, again, it is pointless to speculate. It may well be possible to adopt steeper slope angles at many locations if required but there will certainly be places where even the low angle adopted may prove too steep. The main pit access incline has been located in such a way that it should not be vulnerable to slides once the pit has opened out. However, very careful attention must be given to the design of this incline to guarantee, as far as humanly possible, its stability.

3. With regard to the selection of the mining method and equipment, a practical scheme has been adopted without, however, attempting detailed optimisation exercises which, again, would be pointless until more information is available on the characteristics of the rocks to be mined. Unless the physical environment proves to be even worse than anticipated, it could be expected that somewhat more economic combinations of equipment could be adopted. Therefore, the economic estimates can be regarded as being near the upper bound of the uncertainty band (see Chapter VIII, para 21).

DATA COLLECTION

4. It is recommended, therefore, that efforts be directed at this juncture to obtaining the necessary data rather than trying to develop further, more refined, conceptual mine designs. It is a matter of mining engineering judgement when the amount of additional data is sufficient to validate the design within acceptable technical and economic limits. It is, therefore, important that the collection of this data should be under the general control of the mining engineer (subject to specialist advice), so that efforts are directed towards the most important uncertainties and areas of interest. If this is not done, a very expensive and time-consuming exercise could develop. Although this would undoubtedly elucidate the complex geological and geotechnical setting in the Hat Creek valley, it would be of limited practical use to the mining engineer charged with the responsibility of designing and developing the mine.

Geological and Geotechnical Programmes

5. It is for the reason given above that Report No 1 recommended a two-year geological and geotechnical test programme. There is no reason arising from this study to alter that view, unless it is a question of the timing of the project as a whole. In this case, a substantial increase in the annual amount of drilling envisaged would become necessary.

6. The recommendation to limit exploration to the 600-ft pit, ie to an elevation of 2,400 ft, still stands and it is not considered feasible to design a deeper pit until substantial mining experience has been gained. Incidentally, this applies equally to the hypothetical underground mine as to an openpit.

Coal Quality

7. Report No 1 recommended that the coal quality data be processed in blocks. An outline of how this should be done is contained in Chapter V.

Pit Optimisation

8. It is recommended that a suite of computer programs be developed, based on block values of the whole range of data, starting with coal quality data and adding further data such as mechanical properties, waste rock identification, etc.

Coal Testing

9. The recommended coal testing programme would comprise:-

(i) preparation process testing

(ii) mechanical properties

(iii) crushing and grinding

(iv) weathering, degradation

(v) oxidation, spontaneous combustion

(vi) combustion characteristics

(vii) properties and behaviour of the ash

(viii) properties and behaviour of the plant waste products.

This programme would involve the procurement of bulk samples, shipment to test laboratories, test specification, control and reporting and, finally, the appraisal of the results.

Until this is done it is not possible to come to a definitive conclusion, although it must be said, prima facie, that coal preparation is an unattractive proposition.

LABOUR

10. Further investigation is required to ascertain the extent to which labour availability and salary rates for a mine in the Hat Creek area are likely to be affected by other developments in the coal industry in Western Canada.

BUDGETS

11. The budget estimates given in Report No 1, Appendix "C" have been revised in the light of the economic results derived in the course of this study and the revised budget is given in Table XXVIII. All costs are at 1975 prices and the former estimates are shown in brackets for comparison. It can be seen that the inclusion of fixed installations indicates an apparent increase in the mining equipment and services charge of some $34 million. Furthermore, the expenditures are incurred a year earlier. This is also the case for the Creek diversion and box-cut and pre-production overburden removal items. This last item is increased by some $5 million. The advancement of this capital expenditure pre-supposes that an early decision on the sequence of mining the alternative deposits in the valley can be made.

ENVIRONMENTAL CONSIDERATIONS

12. The configuration of the deposit necessitates an openpit type mine rather than a strip-mine, area mine or hilltop mine as are usually the case with surface coal mines. The high coal output required and the large volumes of waste material necessarily having to be removed will leave a large hole and result in large waste dumps. Backfilling of the hole by waste will not be possible until the Area 1 deposit is worked out or abandoned as far as surface mining is concerned. Naturally these major features will have substantial environmental impact but sympathetic consideration and thorough advance planning can mitigate these effects.

APPENDIX "A"

REPORT NO. 1 - INTERIM REPORT ON GEOLOGICAL AND GEOTECHNICAL EXPLORATION AT HAT CREEK

(November 1975)

CONTENTS

LIST OF APPENDICES

I INTRODUCTION

TERMS OF REFERENCE

1. We submitted proposals for the Hat Creek Mining Study on 16th September, 1975 and these were accepted by BCH letter dated 16th September. Work commenced immediately in accordance with the terms of reference which are reproduced for convenience in Appendix A. This interim report covers only items 3(a) and 3(b).

AFFILIATES

2. As described in the proposals, we are associated with locally-based affiliates and the allocation of responsibilities is as follows:-

Wright Engineers Ltd (WE)	- Local mining data, mine services, detailed design (later stage)
Golder Associates (GA)	- Geotechnical investigations.

Mr. G. Walton, our geological and geotechnical Associate Consultant, is responsible for geological interpretation and liaison with both GA and your geological consultants, Dolmage Campbell and Associates Ltd (DCA).

PROGRESS TO DATE

3. Dr. M.J. Barber, Managing Director, was present in Vancouver for the discussions in connection with the assignment and was able to make an immediate start obtaining data from DCA and holding preliminary meetings with them and with WE and GA. Shortly afterwards Mr. S.C. Brealey, Director in charge, and Mr. G. Walton arrived and a visit of all parties to the site accompanied by Dr. P.T. McCullough, BCH geologist, took place on 6th and 7th October. Subsequent visits have been made by GA personnel to complete the preliminary geotechnical field work.

4. The documentary information collected to date is listed in Appendix B. The assimilation, interpretation and utilisation of this large volume of data is now in progress.

5. The first requirement is of course the assessment of the geological and geotechnical situation, ie the subject of this report. This is because the further work which is known to be required is time-consuming and should therefore be started as soon as possible and also because this information is urgently required even for preliminary mine design upon which more accurate budgets can be based. Work is proceeding on the preparation of basic maps and sections for mine design purposes and is well-advanced for Area 1.

PROGRESS REPORTS

6. Monthly Progress Reports No 1 and 2 have been submitted on 21st October and 20th November respectively. Also preliminary budget estimates were given by telex on 20th October (see Appendix C). These budgets were, of course, very preliminary and adjusted figures are shown in the light of this present report.

CONCLUSIONS

7. The main conclusions reached to date are discussed below:-

(i) Geological exploration should concentrate on the northern end of Area 2 to bring it up to the same level as Area 1. Area 1 looks the most promising at this stage.

(ii) The depth of geological exploration and geotechnical investigation should be limited to a tentative pit depth of 600 ft with the exception of an occasional deeper hole to explore major features.

(iii) Geotechnical problems will be severe due to surface mud flows and the nature of the rocks in the mine, particularly the mudstones.

(iv) Coal quality data requires to be processed in blocks so as to be able to select a mining sequence and to develop production schedules.

GEOLOGICAL EXPLORATION

8. The southern end of Area 2 looks the least promising from a mining point of view, despite the great thickness of coal there, because the deposit is very narrow and the cover is great due to the fact that it is not under the valley bottom but under the steep hillside. We have therefore recommended concentrating drilling on the northern end of Area 2 so as to bring it up to the same level of exploration as Area 1. This will enable a reliable comparison to be made. However, preliminary feeling is that Area 1 will prove to be the most favourable mainly because of the thinner cover and wider deposit, and therefore an in-fill drilling programme for that area is recommended.

LIMITATION IN DEPTH TO 600 FT

9. The reasoning behind the recommendation to limit the pit depth to 600 ft maximum is as follows:-

(i) Geotechnical problems increase non-linearly with depth and it may not be feasible to go deeper within the constraints of guaranteed production, safety and economics.

(ii) In Area 1 reserves to 600 ft depth are estimated to be more than 300 million tons of in situ coal - sufficient for about 22 years of power station operation. Total in situ reserves are estimated to be around 500 million tons but to mine this additional tonnage would involve:-

(a) Increased depth - 1,200 to 1,500 ft (beyond current world experience in weak rock).

(b) Increased width - proportional to depth <u>if</u> overall slope angles can be maintained. This will in turn result in the sides of the pit climbing further up the sides of the valley, thereby increasing waste excavation and where mud flows are present probably requiring extensive stabilisation works.

(c) Increased volume - proportional to $(\text{depth})^3$ <u>if</u> overall slope angles can be maintained, eg

Depth (ft)	Volume (10^6 ft^3)
600	1,600
1,500	24,600

(assuming inverted cone)

(iii) There are possibilities other than extending the pit in depth, eg underground mining, opening another pit, in situ gasification etc. One of these may prove to be safer and more economic by the time extension is needed.

(iv) Concentrating exploration and test work at levels above 600 ft depth will be considerably quicker and cheaper in achieving the same level of confidence as for a deeper pit. Alternatively, a higher degree of confidence can be reached with the same expenditure of time and money. Much of this information will, of course, be amenable to extrapolation in depth anyway. Furthermore, actual observation of slope behaviour during mining operations will be possible and this will considerably improve the level of confidence in designing a deeper pit.

QUALITY OF COAL

10. Coal quality data are still coming in and will continue to do so whilst drilling continues. DCA have processed these data on a borehole basis, by borehole groups and by the whole deposit.

However, in order to predict rom quality and selective mining possibilities and to develop production schedules it will be necessary to re-process these data by mining blocks. For example, if a 40 ft bench height is selected the deposit would be divided into horizontal slices 40 ft thick and each slice into blocks. Then, for any mining sequence, it will be possible to predict:-

(i) Mean and range for each quality parameter of rom coal production.

(ii) Amount of intercalated waste which can be removed selectively.

This would be done year by year. Overburden stripping schedules would then be developed to enable this coal to be won. Then it will be possible to predict accurately overall mining costs on an annual basis. In the meantime approximate methods will have to be used.

ACKNOWLEDGEMENTS

11. The considerable help and assistance of BCH staff and the co-operation of DCA is acknowledged with thanks.

II GEOLOGICAL ASSESSMENT

1. This assessment has been carried out by Mr. G. Walton who has also commented in Section III on the geotechnical assessment. Attention is concentrated on the structure and rock types in the potential mining areas.

2. The bulk of the drilling which has been done to date is still in the north of the valley (Area 1) hence observations on the quality of drilling as well as geotechnics deal mainly with that area.

3. The synopsis of the regional geological setting as set out in DCA reports is agreed in general. The more detailed reports by DCA and BCH show certain differences of interpretation resulting from limited drill data. In drawing up outline plans at Enfield, the BCH structural interpretation has been favoured as it appears to give a better fit for dips and partings in boreholes. Even so, the current plans are still mostly conjectural.

STRUCTURE

Faulting

4. This is by no means fully located at all points and in some cases faults shown on sections do not agree with data from fully cored boreholes. The strata correlation is at best crude; since the base of coal is not fully ascertained, the faults can only be related to the somewhat transitional top surface of coal and known positions in cores. At this stage, therefore, major fault positions could vary by several hundred feet: similarly there appears to be little chance of ascertaining faults of small throw, say less than 50 ft.

5. The major fault trend appears to be sub-parallel with Hat Creek and to result in the basic geometry of the deposit, namely a long narrow trough. However, it is likely that the major controlling faults to the east and west down-throw the coals on either flank, ie the coal structure is a horst rather than a trough (although the structure of the valley as a whole is a graben). It is probable that deep coal, in excess of 1,500 ft, is present on either side of the main deposits. Other

faults are parallel to this main north-south trend, both within and probably outside principal coal areas.

6. Other significant fault trends run ENE to NE and produce changes in the elevation of the coal along the horst, without greatly displacing the deposit across the valley.

7. The low angle fault referred to by DCA in Area 1 may well be a slip plane related to surface instability.

8. The operational significance of these faults is:-

(i) Change in diggability at the same bench level, eg coal into claystone.

(ii) The stability of slopes may vary markedly across faults, eg on the west side of Area 1, strata dipping towards the proposed excavation will be encountered on crossing the fault. Also related changes in permeability and groundwater levels on different sides of faults may be expected.

(iii) Materials in fault zones may be several 10's of feet thick and being in a disturbed condition may themselves be unstable.

Folding

9. This is based on upper coal surface contouring allied to observed dips in cores and their use in sections. Again, in the absence of reliable marker horizons for correlation, the dips of coal and overburden materials are only approximate in the north (Area 1) and open to considerable differences of interpretation in the south (Area 2).

Area 1

10. In Area 1 it is probable that the axis of folding is sub-parallel to the main faulting (and Hat Creek), with a plunge of the fold axes to the south over much of the deposit. It is

not clear to what extent the dip of strata is altered near to the main faults, but a western syncline and an eastern anticlinal horst appear as elements in the main coal area. Whilst the strata east of the main eastern north-south fault (Mag Fault) apparently dips to the east, that on the west of the deposit does not dip away from the horst, but again towards the east.

Area 2

11. Area 2 is apparently a single anticlinal horst with a major element of dip to the west.

12. The operational significance of the inferred dips and overall structure is:-

(i) The combined effects of faulting and plunge ensure that in Area 1 with a 600 ft deep mine, a proportion of both upper and lower sections of the thick coal would be included in the take. No such balance exists in the south (Area 2), where only the upper and slightly inferior section is present to 600 ft.

(ii) Dips are frequently in excess of 30°. Since coal and parting materials will be loaded from the same bench with the same excavators, it is significant that a range of geometric configurations is likely. Irrespective of the pit shape, coal and parting may be present within the same bench height, dipping into, out of, or obliquely to the bench face. Hence, problems of segregation during excavation are possible and some bench collapses are probable if bench slopes are very steep. It is partly for this reason that more detailed exploration is needed within the coal area.

(iii) The western slope of Area 1 comprises strata dipping at 15 to 30° towards the excavation. Quite apart from the problems of superincumbent mud slides, it seems likely that careful benching will be necessary, possibly to the extent that bench faces lie parallel to the dip of the strata. Even with this solution the possibility of slides due to buckling or cross-over joints must be investigated.

(iv) The separate and thinner coal on the western side of the deposit will lie partly within the slopes of the 600 ft deep mine, therefore the proposed limits on the western side of Area 1 must, in due course, be carefully examined to investigate the final limit of working.

(v) The combined effect of dip and faulting will influence the siting of the main access incline. A truly central incline would probably lie west of the anticlinal horst running approximately NS along the position of Hat Creek, where coal outcrops can be seen. However, on the basis of structural interpretations, certain advantages may accrue if it were sited along this horst to the east of Area 1, and not in the centre of the deposit.

(a) Less coal might be temporarily or permanently sterilised beneath the incline, since the horst appears to include the lower coals, most of which would be removed, whereas the central incline would be located over upper coals and preclude the mining of the lower coals.

(b) In view of the preliminary findings of GA regarding the location of mud slides, and the identification of more stable glacial gravels, a stable easterly incline may become quite feasible. Clearly the positioning of the side slopes of the incline in various places must be examined to check the implications for security of the whole.

13. It is possible to see significant gaps in present structural knowledge of ground within the slopes. Using the range

of slope angles from 15 to 25^{o} as reasonably suggested by GA, it is notable that large areas have not been drilled to date particularly on the north-west, east and south-east areas of the proposed mine. Particular attention must be given to these areas to determine the dip of the strata, since outside the coal areas it is not possible to use the assumption that overlying claystones are conformable. Also, little is known of the levels of the interfaces between the claystones and the volcanic rocks on the west, and the claystones and gravelly superficials on the east. These interface levels must be known, since stable and economic slope angles will vary in the different soil and rock types.

14. The gaps in structural information within the main coal deposit are smaller, but just as significant, and recommendations regarding the infill drilling are made below.

MATERIALS

Overburden, Partings and other Strata

15. The observations of GA are pertinent to both stability and ease of excavation. Clearly the stored cores of claystone may be either stronger (dessicated) or more broken (slaked) than fresh material. From the description given in Section III, the compressive strength of a few of the claystones could be as low as 200 to 300 kN/m^2, ie 30 to 45 lb/in^2. (English coal measure mudstones range from 700 to 5,000 lb/in^2.) A few compressive and shear strength tests would certainly be a useful guide at this stage. This material is obviously an engineering soil and will behave as such, except where bedding separation surfaces and mylonites are present; ie rotational or curvilinear failure surfaces will develop. Benches of 50 ft could well be too high, even for short term slopes and the proposed 40 ft benches now seem more realistic. Obviously more reliable bench heights and angles will be obtained following testing: since stability and drainage are related and time dependent, this testing should consider the effects of drained, partially drained and undrained

material strengths. The strength range indicated (perhaps 30 to 500 lb/in^2) would probably be suitable for either excavation with loading shovels or BWE. Of the remaining materials, little is known at present. We are not even certain whether the moderately strong materials at the NW end of Area 1 are part of a burn zone or true volcanics. Either material is likely to be quite strong and will require blasting and excavation with shovels. Obviously much more data are required in this area and the mode of origin must be ascertained since this has a bearing on features which can be anticipated. Slopes between the proposed range of slopes 15 to 25^o would incorporate large areas of overburden of completely unknown strength, ie proportion and distribution of volcanics and claystones. A significant amount of drilling must be allowed to cover the slopes on all sides of the pit to permit a good assessment of excavation requirements, as well as stability problems.

16. The superficial materials are likely to vary greatly: the large mudflows should represent quite easy digging, but difficult stability problems. The glacial gravels and boulders would give few stability problems, unless lying above weak clays or water-logged sands or silts. In both materials loading shovels should handle material adequately. More needs to be known about the distribution of boulder sizes and the size of competent blocks within the mudslides and moraines before commenting on the use of BWE's in these materials.

Coal

17. Trends in quality are not precise and reflect the difficulty in establishing lithological correlation, even over short distances. It is usual for coal quality variations within an individual horizon to follow systematic trends and for these variations to be characteristic of a single seam or layer with other variations in the quality of adjacent seams. It is doubtful, therefore, that quality variations can be produced on a seam-by-seam basis, and therefore data must be accumulated on a bench-by-bench basis so that the effects of deepening and widening on systematic benches can be examined. This may require DCA to change

their system of sampling and presentation of data, particularly that from inclined boreholes, and it is partly for this reason that it is suggested that future infill drilling should be vertical.

DRILLING COSTS

18. In many parts of the world it is usual to undertake detailed coal exploration in relatively simple geological settings at borehole spacings down to 1/16 mile. On this basis 5,000 to 10,000 tons of coal is proved per foot of borehole in coal. Drilling costs may not directly reflect this figure, since the relationship does not take into account footage drilled in overburden. However, it is recommended that a similar approach should be used in an area such as that at Hat Creek where a considerable degree of structural complexity obtains.

Basis of Drilling

19. Since some major faults and fold axes trend N to S along the valley, the emphasis on E to W sections, and also N to S sections, appears sensible. The present distance between sections of 1,000 ft in Area 1 is too great to remove structural uncertainties regarding correlation or the location of faults. Much of the recent drilling has been inclined: some of the earlier boreholes do not have reliable analytical results and some data from recent inclined boreholes are difficult to integrate with a bench-orientated layout. Hence it is suggested:-

(i) That further EW lines of cross-section be established at 500 ft intervals from N to S.

(ii) That the spacing interval between boreholes on section lines should, if possible, be not greater than 400 ft.

(iii) That vertical, rather than inclined boreholes be employed.

(iv) That drilling should extend to the depth indicated by preliminary geotechnical and mining considerations, namely 600 ft and should be extended to include ground within the provisional slopes.

Choice of Future Drilling

20. The present drilling, although of greater density in the north (Area 1) appears to indicate that this area is preferable to the deposits in the south (Area 2 north and 2 south). The basis for this assumption is:-

(i) A better balance of upper and lower thick coal in Area 1.

(ii) A large coal tonnage with limited overburden cover in Area 1 if mined to a fixed depth of 600 ft. The cover in Area 2 south is particularly great, since it lies beneath a drift terrace rather than along the Creek bottom.

Possible tonnages in each of the three areas:-

Area	In situ tonnage of coal
1	316,000,000
2 north	248,000,000
2 south	163,000,000

Note: tonnage based on 250 ft overburden cut-off and 600 ft depth at toe of slope.

(iii) A more circular geometry in the north which favours an access incline in the centre of the pit. In a setting where even after full geotechnical studies some instability is likely; a stable side wall incline, almost essential in the narrower southern deposits, may be difficult to guarantee.

21. Notwithstanding the above remarks, the estimates have been made on the assumption that both outline and infill drilling is completed in at least one part of Area 2. The future drilling programme is, therefore, divided into the following:-

(i) Drilling required to bring Area 2 (north) up to approximately the level of knowledge of Area 1 at present.

(ii) Further drilling required to infill Area 1 both within the main coal area and the slopes.

(iii) Further drilling required to infill Area 2 (north) both within the main coal area and the slopes.

(iv) The likely cost of undertaking full geotechnical logging of these boreholes, together with additional detailed geophysical logging for structure (seisviewer).

22. The cost of the above is as follows:-

(i) Drilling on E-W section lines at 1,000 ft intervals allowing three boreholes of 1,000 ft on 8 section lines, the approximate footage with contingency allowance is 25,000 ft. Cost: $750,000.

(ii) Further infill to 400 ft spacing on existing section lines, 36 additional 600 ft deep boreholes with contingency allowance is 25,000 ft. Infill of the N-S interval between E-W sections of 500 ft spacing requires 48 additional 600 ft deep boreholes with contingency allowance, 36,000 ft drilling. Infill of slope areas with 4 boreholes from toe to crest within 1,000 ft length of slope toe, is 4 x 15 boreholes of 300 ft, ie 18,000 ft. Total footage 78,000 ft at total cost of $2,040,000.

(iii) Further infill drilling on each section line 2 x 10 boreholes of 600 ft with allowances, 15,000 ft of drilling. Infill drilling at 500 ft N-S intervals in E-W section lines 10 x 5 boreholes of 600 ft depth with allowance is 36,000 ft of drilling. Infill of slope areas with allowance as in (ii) above, 4 x 20 boreholes of 300 ft, ie 24,000 ft. Total footage 74,000 ft at total cost of $1,890,000.

(iv) Area 1 geotechnical logging costs $234,000. Area 2 (north) geotechnical logging costs $222,000.

Hence, summary of costs:-

		$	
Outline drilling Area 2 north	-	750,000	
Infill drilling Area 1	-	2,040,000)	alternative
Infill drilling Area 2 north	-	1,890,000)	
Geotechnical logging Area 1	-	234,000)	alternative
Geotechnical logging Area 2	-	222,000)	

III GEOTECHNICAL ASSESSMENT

1. The GA report is appended complete to this Section. It should be appreciated that this assessment is the result of a very brief preliminary appraisal and it will require revision from time to time as further data become available.

2. Some comments on the GA assessment are given below:-

(i) Both breakdown and dessication hardening of the claystone are superficial phenomena and stability effects depend on depth and rate of penetration and the extent to which affected material remains in position. The implications are probably more serious for the stability of spoil heaps than for working or ultimate slopes.

(ii) It is important that shear testing relates to appropriate groundwater conditions, ie drained, partially drained and undrained, to correspond to different requirements of slope life.

(iii) The strata dips and scale of faulting is far greater than that usually found in active coal mining areas, and it is greater than that found in locations such as the Peace River Dam, where clay mylonites were first noted. It is possible that mylonites/gouge zones could be moderately extensive, covering several benches rather than a single bench. As mentioned in Section II, faults could delimit areas of marked difference in groundwater and drainage conditions, which may have a bearing on stability, but it is unlikely that major slides will occur along the predominantly vertical fault planes.

(iv) The observations on mudslides are accepted and the need to identify geometry and controls, as a major preliminary to deeper geotechnical investigations, is emphasised.

(v) Asymptote B reflects mobilized cohesion which reduces with time, by weathering, creep etc. The relationship between bentonite content and cohesion appears uncertain.

(vi) In general, long term slopes (asymptote A) will be stable at an angle equal to half the residual friction angle of the soil material for non-artesian, but surface-saturated, conditions. This is indeed dependent on drainage conditions.

(vii) As indicated in (iii) above, mylonites may not be of limited extent: clearly their presence must be checked in infill drilling. However, it is doubtful whether the extent can be ascertained, since, although these zones are parallel to bedding, lithological correlation is imprecise.

(viii) GA could have discussed the aspects of "severity of stability problems in relation to depth".

(ix) Sampling and testing of the basal slip surface should be undertaken and the piezometric elevations beneath the superficial deposits should also be examined.

(x) Whilst some stratigraphic correlation could be useful, both to determine the continuity of critical horizons, and as a check to DCA's work, the amount of this checking should be limited.

1. INTRODUCTION

On the instructions of PD-NCB Consultants Ltd., acting on behalf of B.C. Hydro and Power Authority, Golder Associates Ltd. have carried out a review of available information related to the geotechnical aspects of open pit mining of the Hat Creek coal deposits. In addition, a limited amount of field work has been carried out in order to provide a preliminary assessment of the nature and extent of slope stability problems which may be associated with mining. Preliminary conclusions resulting from this study are presented in this report together with proposals for additional investigations which are considered necessary for a more reliable definition of slope stability problems and their solution.

2. BACKGROUND GEOLOGICAL INFORMATION

The number 1 Hat Creek coal deposit was reported by Dr. G.M. Dawson of the Geological Survey of Canada in 1877 and 1894. A limited amount of exploitation of the outcropping coal took place in 1924 and again between 1933 and 1942 but systematic subsurface exploration of the site was not carried out until 1957 when a subsidiary of the B.C. Electric Co. Ltd. carried out some diamond drilling in the area.

The present exploration programme, involving a considerable amount of diamond drilling in both Areas 1 and 2, was commenced in mid-1974 by B.C. Hydro and Power Authority, present holders of coal licences covering most of the Upper Hat Creek valley.

It is not the purpose of this report to review the substantial amount of geological information which is already available and which is currently being accumulated in Hat Creek. This data has been adequately su[illegible] in the following reports:

1. P.T. McCullough, Memorandum on Hat Creek Exploration Program-Progress Report, July 1975. B.C. Hydro and Power Authority.
2. B.C. Research and Dolmage Campbell & Associates Ltd., Preliminary Environmental Impact Study of the Proposed Hat Creek Development. Submitted to B.C. Hydro and Power Authority, August 1975.
3. Dolmage Campbell & Associates Ltd., Progress Report No. 1, No. 2 coal deposit, Hat Creek project. Submitted to B.C. Hydro and Power Authority, September 1975.

A very brief review of the geological setting of the Hat Creek coal deposits is considered appropriate and is as follows:

The valley of Upper Hat Creek is largely underlain by tertiary sediments that form an elongated basin-like structure conforming to the valley walls. The basin lies north-south along the valley of Upper Hat Creek for a length of about fifteen miles with an average width of about three miles. The valley sides at higher elevations are comprised of terraces of younger volcanic rocks that apparently represent remnants of younger volcanic deposits that once covered the valley.

The sedimentary rocks are predominantly very weak poorly cemented silts and clays and semi-consolidated sands. The sediments can be easily broken or cut with a knife and many sections are so soft that they can be remoulded by hand. The underlying coal is contained in two major seams; the upper seam is about 400-600 feet thick and the lower seam is approximately 1600 feet thick.

A prominent feature of the tertiary bedrock is the presence of major, regional nearly vertical block faults. The principal known faults cross the north end of the valley in an east north-east direction and are spaced about one mile apart. The second major fault trend is roughly parallel to the Upper Hat Creek valley. These faults are probably related to the Fraser River Fault system lying approximately four miles to the west. These large block faults have played an important role in localizing the coal.

3. GEOTECHNICAL BACKGROUND

The geotechnical studies which were carried out on the Hat Creek project before August 1975 are summarised in the following report:

> Dolmage Campbell & Associates Ltd., Hat Creek Project, No. 1 Deposit Rock Mechanics Data. Submitted to B.C. Hydro and Power Authority, August 1975.

Most of the data presented has been obtained from tests carried out at the suggestion of Ralph B. Peck whose two letters, dated February 10, 1975 and August 25, 1975, summarise the considerations which have been given to pit slope stability. The conclusions reached by Peck may be summarised as follows:

1. Most of the materials overlying the coal show very severe breakdown during slaking tests and are expected to exhibit the behaviour of heavily overconsolidated clays or clay shales of the unbonded variety.
2. Because of the great height of the anticipated slopes, the cohesive strength of these materials will not play a significant role in slope stability which will be dependent upon the frictional resistance of the material. This frictional resistance will have its lowest value along bedding planes.

3. The peak friction angle for smooth bedding surfaces is estimated at between 20 and 30 degrees.

4. When the bedding planes are oriented in such a manner that failure would occur across rather than along the planes, a roughness component $i = 15^{\circ}$ can be added to the average friction angle of the bedding planes $\emptyset = 25^{\circ}$ to give an effective friction angle $\emptyset + i = 40^{\circ}$ along the failure surface.

5. The friction angles given above are based upon the assumptions that
   a. the water pressures along the sliding surfaces are close to zero and
   b. the materials have not been subjected to previous shear displacements which would have reduced the frictional resistance from peak to residual values.

6. Significant groundwater pressures could result in a reduction of about 50% in stable slope angles.

7. The presence of gouge or mylonites in areas subjected to previous shear displacement could reduce the friction angle parallel to bedding to as low as 3° in local areas. Because the areas subjected to previous movement are not expected to constitute a large percentage of the total volume of material overlying the coal, the presence of gouge or mylonite is expected to influence local bench failure rather than overall slope stability.

8. On the assumption that some effective drainage can be implemented, it is concluded that slopes in which beds dip towards the cut at angles in excess of 20° will be stable at the inclination of the bedding, provided that this does not exceed 40°. Elsewhere, the stable slope angle is estimated at 40°.

4. REVIEW OF PECK'S CONCLUSIONS

Golder Associates Ltd. have reviewed Peck's conclusions in the light of the information which has become available up to the time of writing this report. The following comments are offered on the conclusions summarised on pages 3 and 4.

1. Although Peck notes that severe breakdown of the materials overlying the coal occurs on exposure, he does not discuss this in relationship to the stability of the slopes. This breakdown may prove to be particularly significant in relation to operating bench slopes which are only required to remain stable for limited periods of time.

2. Peck's conclusion that the cohesive strength of the materials should be ignored in slope design applies specifically to ultimate pit slopes which will be of great height. The stability of many slopes of limited height will be critical during development and operation of the pits and the cohesive strength of the materials will be important in determining the safe angle at which these slopes can be mined.

3. Comments by a number of people associated with the early geotechnical studies suggest that all shear tests carried out by Peck were on conglomerates and sandstones underlying the coal. Consequently, the estimated friction angle of between 20 and 30 degrees may not be applicable to the Coldwater claystones and the coals which will be exposed in many of the pit slopes. No shear tests have been carried out on the claystones but Golder Associates Ltd. are giving serious attention to obtaining representative samples of these

materials in order to conduct a limited number of shear tests on both claystones and coals.

4. It is the opinion of Golder Associates Ltd. that the joint roughness component $i = 15^{o}$ suggested by Peck can only be applied to the conglomerates and sandstones underlying the coal and to the small areas of volcanic rock and burnt materials which occur on site. The Coldwater claystones are very weak and their failure will probably be plastic rather than dilatant.

5. The friction angles suggested by Peck are based upon the assumptions that the water pressures in the rock mass are low and that the materials have not been subjected to previous shear displacement. Golder Associates Ltd. consider that there will be a number of areas in the pit in which neither of these conditions will be satisfied and, consequently, the slopes will be significantly less stable than an analysis based upon peak shear strengths under drained conditions would indicate.

6. Examination of the site reveals that there is a large amount of groundwater present and that only a limited movement of this water occurs. This suggests that drainage of the materials in which the slopes are to be cut may be very difficult to achieve.

7. Exploration drilling has revealed that the area is extensively faulted and that there has been some slumping during deposition of the sediments. Both these conditions are associated with large shear movements and with the formation of gouge or mylonites

which will seriously reduce the stability of slopes in which they occur. Golder Associates Ltd. agree that such instability will probably be restricted to local benches since the gouge zones will probably not be of sufficient extent to influence the ultimate slopes. While the faults appear to be continuous over considerable distances, they are generally near vertical and should not play a major role in slope stability.

8. In view of the reservations expressed above, Golder Associates Ltd. are of the opinion that Peck's estimate of a stable slope angle of 40^{o} on three sides of the pit is probably too optimistic. Since Peck's comments do not cover the stability of the glacial drift overlying the coal or the Coldwater claystones which will represent significant proportion of the waste material to be mined, separate detailed discussions on these materials are given below.

5. STABILITY OF SURFACE MATERIALS

The surface materials, which vary in depth from 10 to 300 feet across the site, may be broadly described as glacial drift. The American Geological Institute's Glossary of Geology (1972) defines drift as all rock material (clay, sand, gravel, boulders) transported by a glacier and deposited directly by or from the ice, or by running water emanating from a glacier.

In places, this drift is the result of detritus being removed or "washed out" from the glacier by meltwater streams which has formed stratified layers of sand, gravels and boulders. Because most of the clay sized fraction has been washed away this outwash material has good drainage and forms stable slopes. A good example of such a stable slope occurs on the east side of the road entering the northern end of the valley and is illustrated in Plate 1.

Much of the clay present in the till is derived from volcanic materials which have a high percentage of active clays, principally montmorillonite. With water, these clays become mobile and can produce mud slides.

Examination of the air photographs of the Hat Creek valley and a study of surface features during field work indicates that there are a number of areas in which mud slides have occurred or are presently active. The distribution of these areas is indicated on the map reproduced in Figure 1. Detailed air photo overlays are given in Appendix I of this report. On these overlays, the mudflows have been classified as follows:

1. Active - movement currently taking place.
2. Inactive - no apparent movement at present but movement has occurred and could be reactivated by undercutting of the slide toe or by changes in groundwater conditions.
3. Possible - areas in which movement may have taken place or could take place under adverse conditions.

It must be emphasised that this distribution is based upon a very limited amount of work and must be regarded as preliminary. Since the presence of major mud slides is likely to have a significant influence upon not only the open pit mines but also on the waste dumps and plant site locations, it is considered essential that every effort should be made to define the mud slide distribution more precisely. To this end, further field work is being carried out within the Stage 1 programme and any results which become available will be presented in supplements to this report. A suggested programme for further work is outlined in a later section of this report.

6. STABILITY OF CLAYSTONES AND COALS

A high proportion of the open pit walls will be cut in Coldwater claystone and in coal and it is therefore necessary to consider the stability of these materials.

The claystones contain varying amounts of bentonite and the consequences are as follows:

a) When the bentonite content is low, the claystone hardens on exposure and core which has been stored in a dry core-shed for more than a week. can give a very misleading impression of the in situ material. Point load testing of core which has hardened as a result of dessication indicates an average uniaxial compressive strength of approximately 2000 lb/in^2. It is estimated that this represents an increase of 4 or 5 times on the strength of the in situ material.

b) When the bentonite content is high, the core breaks down extremely rapidly upon exposure. This is illustrated in Plate 2 which shows a core tray in which severe breakdown of sections of the core has occurred. This breakdown will have a significant influence on the stability of benches excavated in claystone with high bentonite content.

Careful examination of fresh core as it is recovered from the borehole suggests that the Coalwater claystone will have the following in situ characteristics.

a) It is easily broken or cut with a knife and many sections can be remoulded by hand.

b) The average uniaxial compressive strength is estimated at approximately 500 lb/in^2.

c) The material is generally isotropic with respect to strength although there are a number of weak joints, particularly parallel to bedding, which will result in local instability.

d) The claystone will have a very low permeability and effective drainage will be very difficult to achieve in this material.

e) When significant percentages of bentonite are present in the claystone, severe breakdown will occur upon exposure.

In view of these observations, the following comments on slope stability are offered.

1. It is considered likely that slope failures in the claystone will occur in a circular failure mode although some local small scale joint controlled slides will also occur.
2. In relationship between the stable slope angle and the height of slopes excavated in this material will be of the form illustrated in Figure 2.
3. The asymptote B, which defines the height to which a vertical face can be cut, will depend upon the bentonite content of the material forming that slope and upon the length of time which the face is exposed. Based purely upon experience, it is suggested that the maximum value of B may be of the order of 50 feet but it is clearly

necessary to establish this value with a greater degree of confidence since it will be important in designing working bench faces.

4. The asymptote A, which defines the stable angle of very high slopes, will depend upon the angle of friction of the material and upon the degree of drainage in the slopes. While it is possible to estimate the value of the angle of friction from shear tests, it will be very difficult to predict the groundwater conditions in this material and hence the definition of a value for A will be subjected to considerable uncertainty. Based upon the very limited amount of information currently available, it is suggested that a range of values of A from a maximum of 25° down to a minimum of approximately 15° may emerge from more detailed studies.

5. It should not be forgotten that there are known to be gouge zones and mylonites present in the rock mass as a result of previous movements. Where these occur, the stability of even the flattest slopes could be jeopardised. In-fill diamond drilling will assist in defining areas in which such materials are likely to occur in significant quantitites but the danger of exposing an unanticipated lens of gouge or mylonite will always exist. Present indications are that these materials are limited in extent and will be more significant in terms of bench stability than in the overall stability of the pit slopes.

While the in-situ coal is fairly strong, it tends to crumble readily on release from confinement and upon exposure. It may have a much higher permeability than the claystone, discussed on the previous pages, and may even be an aquifer within the Hat Creek system. Some stability problems can be anticipated in benches cut in coal but these should not be as severe as those in equivalent benches in claystone.

7. STABILITY OF CONGLOMERATES AND SANDSTONES

The principal materials underlying the coal are conglomerates and sandstones. Faulting has resulted in some of these materials outcropping in local areas and it appears that most of Peck's work was directed towards these materials. In general, these materials are harder and more competent than the claystones, they are more permeable and failures are likely to be controlled by structural features such as bedding planes and joints rather than by the weakness of the materials.

In general terms, Golder Associates Ltd. are in agreement with Peck's conclusions on the stability of these materials.

8. GENERAL CONCLUSIONS ON SLOPE STABILITY

Based upon a review of existing data and a very limited amount of field work, the following general conclusions on slope stability are offered:

1. There does not appear to be any strong preference for mining areas 1 or 2 on the basis of geotechnical considerations. Slope stability problems of equal severity can be anticipated in both areas and the choice should be based upon mining considerations.

2. The overall slope angle of 25° currently being used for pit planning is probably as reasonable an average value as can be arrived at at this time. It is probable that this angle will be too optimistic in many areas but it may be possible to steepen the slopes slightly in other areas.
3. Because of anticipated difficulties in achieving effective drainage, it is suggested that 25° be considered as an upper bound for average slope angles and some preliminary consideration should be given to the consequences of flattening the slopes if required.
4. Since it is probable that mining of the Hat Creek coal deposits cannot be achieved without inducing slides of a variety of sizes, it is considered important that any mine plan should be designed to accommodate, as far as possible, such slides. Hence, working outward from a central stable ramp so that production can be switched from one side of the pit to the other as required, is highly desirable. Consideration should be given to the economic implications of mining areas 1 and 2 in echelon since this would also represent a possible means of achieving regular coal production in the event of slope failure developing in one of the pits.
5. Provision should be made for the detailed study of the stability of access ramps into the pit or pits and for any remedial measures which may have to be used in order to create stable slopes. It will almost certainly be necessary to provide drainage in these areas and it may also be necessary to utilise compaction of existing materials or of replacement materials to achieve the required degree of security.

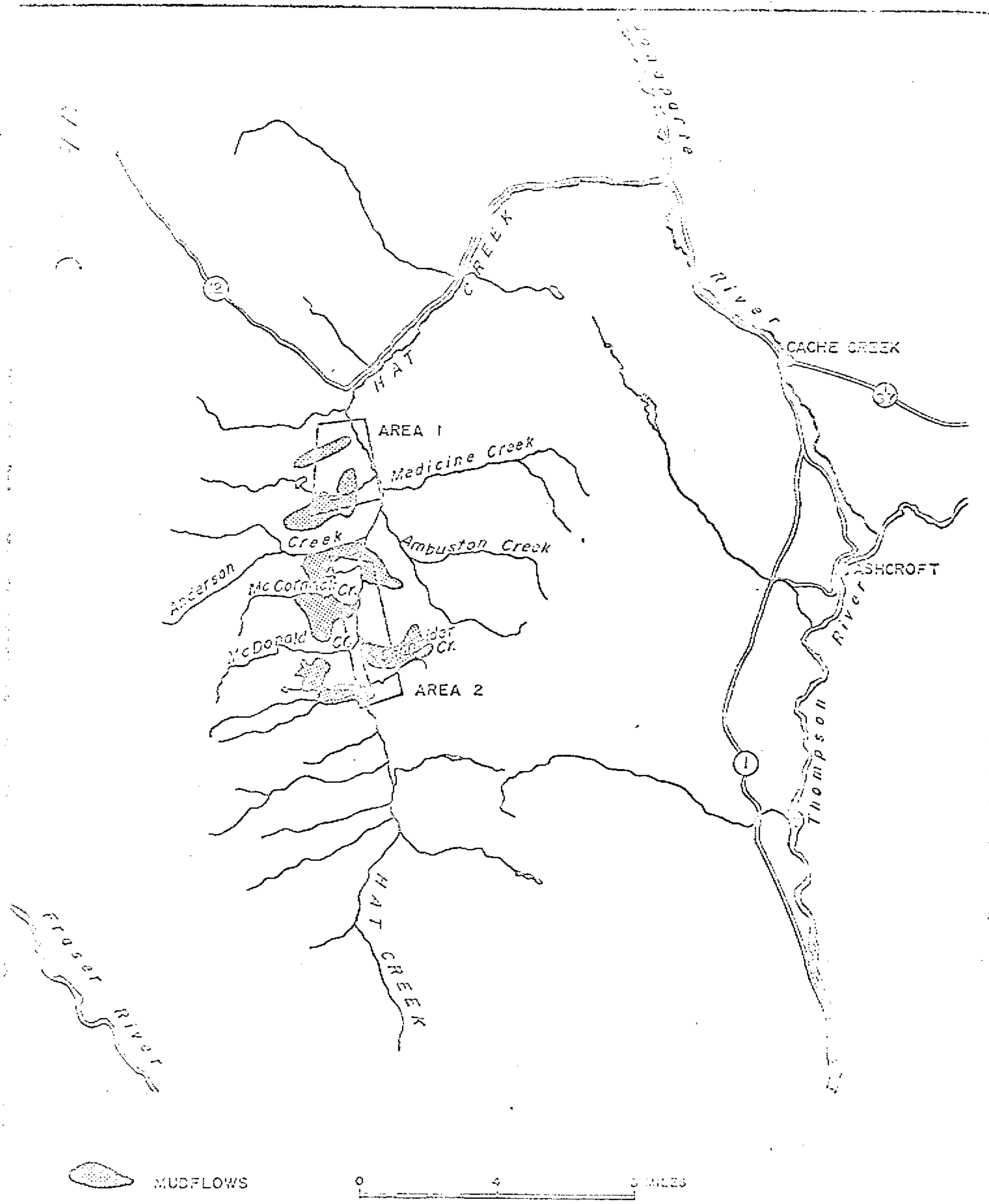
6. The presence of significant areas of potential mudslide material in the Hat Creek valley has serious implications in all aspects of mining. As a first step, it is considered essential to carry out a study, based on air-photo interpretation and field work, to define the extent and nature of these mudslides. Undermining of this mudslide material will probably remobilise the slides and hence consideration must be given to methods for dealing with instability in these materials. In general, these slides will be slow moving but, under certain conditions, can move very rapidly and can be extremely hazardous to men and equipment in the mine.

7. The presence of mudslide material also represents a problem in the disposal of waste and in the location of plant sites. The material itself is extremely poor as a foundation material for almost any structure. Its stability in a waste dump will be severely limited and special provisions such as selective dumping and the use of retaining dykes around dumps may be required.

8. The presence of groundwater has an adverse effect upon stability and its importance has been emphasised throughout this report. A comprehensive programme for the study of groundwater problems in the Hat Creek valley has already been proposed by B.C. Hydro an Power Authority - see Appendix II of this report - and it will be necessary that any geotechnical program should be closely coordinated with this study.

APPROXIMATE LOCATIONS OF POSSIBLE AND PROBABLE MUDFLOWS - HAT CREEK

FIGURE



Golder Associates

[illegible] STABLE [illegible] SLOPE [illegible]
[illegible] FOR [illegible]

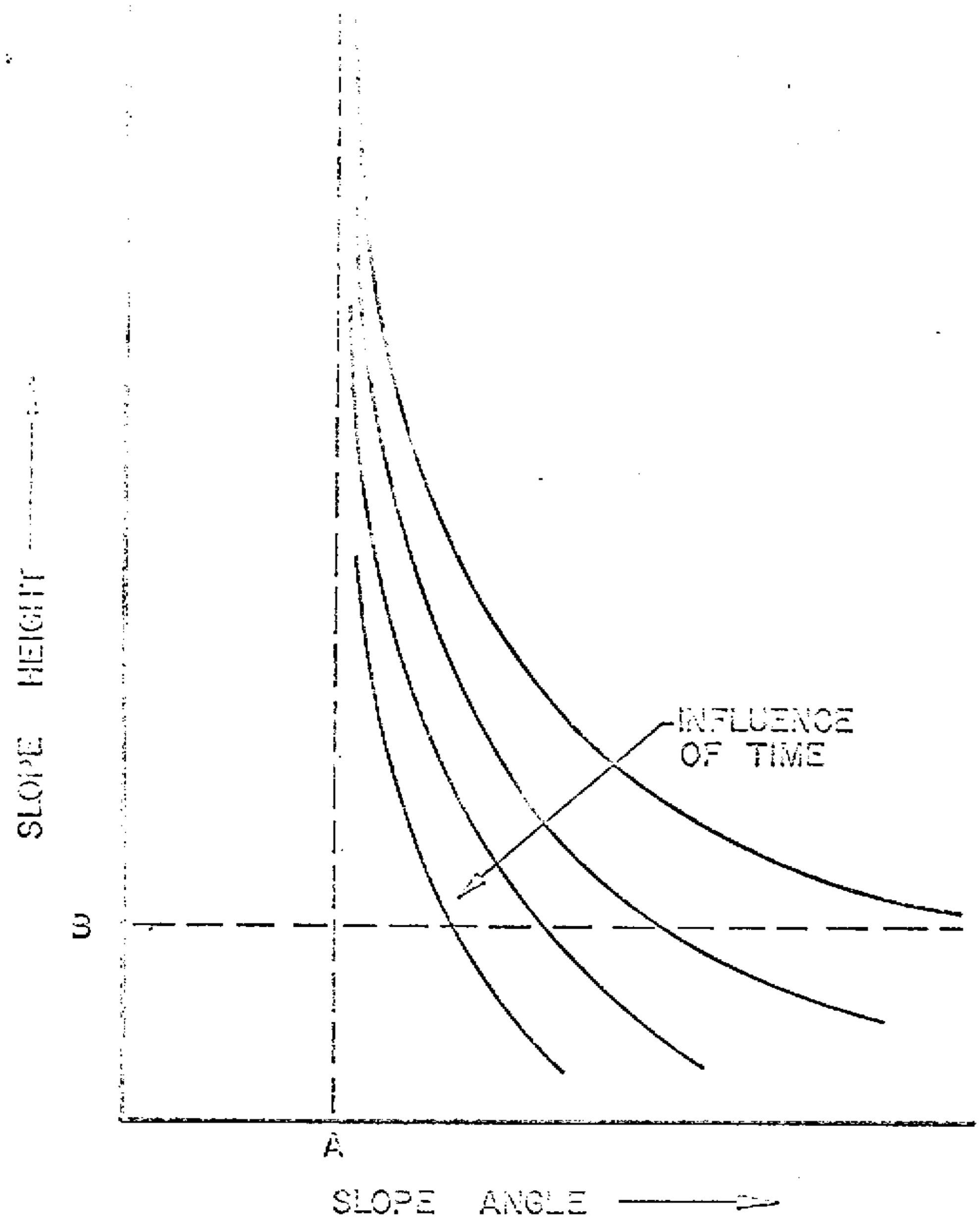


PLATE 1 - STABLE GLACIAL OUTWASH MATERIAL

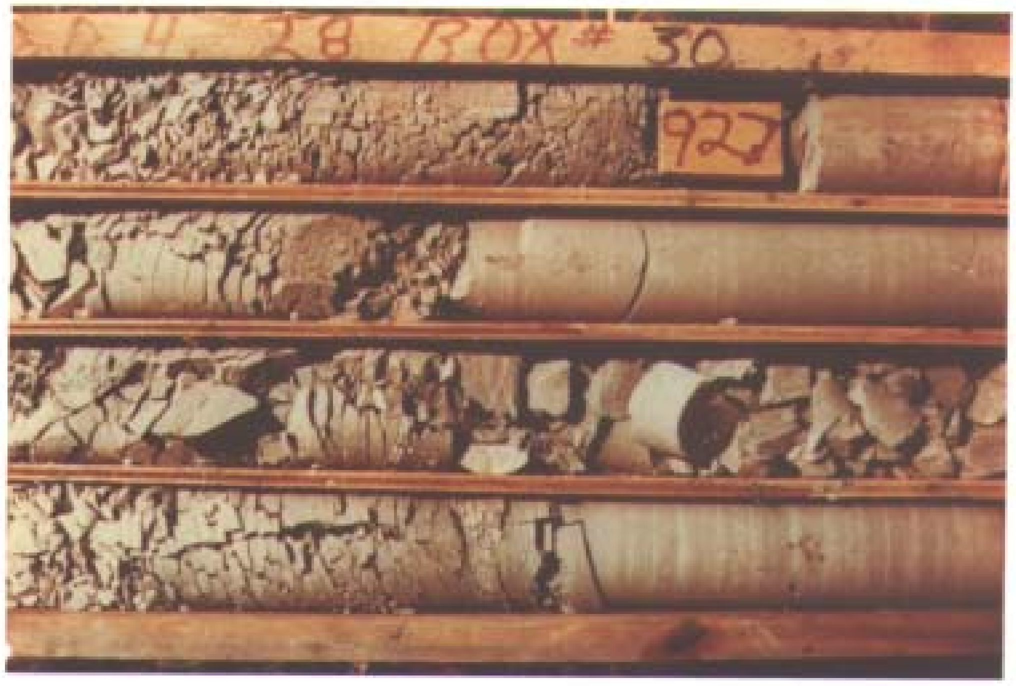


PLATE 2 - EXAMPLE OF CORE BREAKDOWN ON DRYING

APPENDIX I

AIRPHOTO OVERLAYS INDICATING
POSSIBLE MUDSLIDE AREAS IN
THE HAT CREEK VALLEY

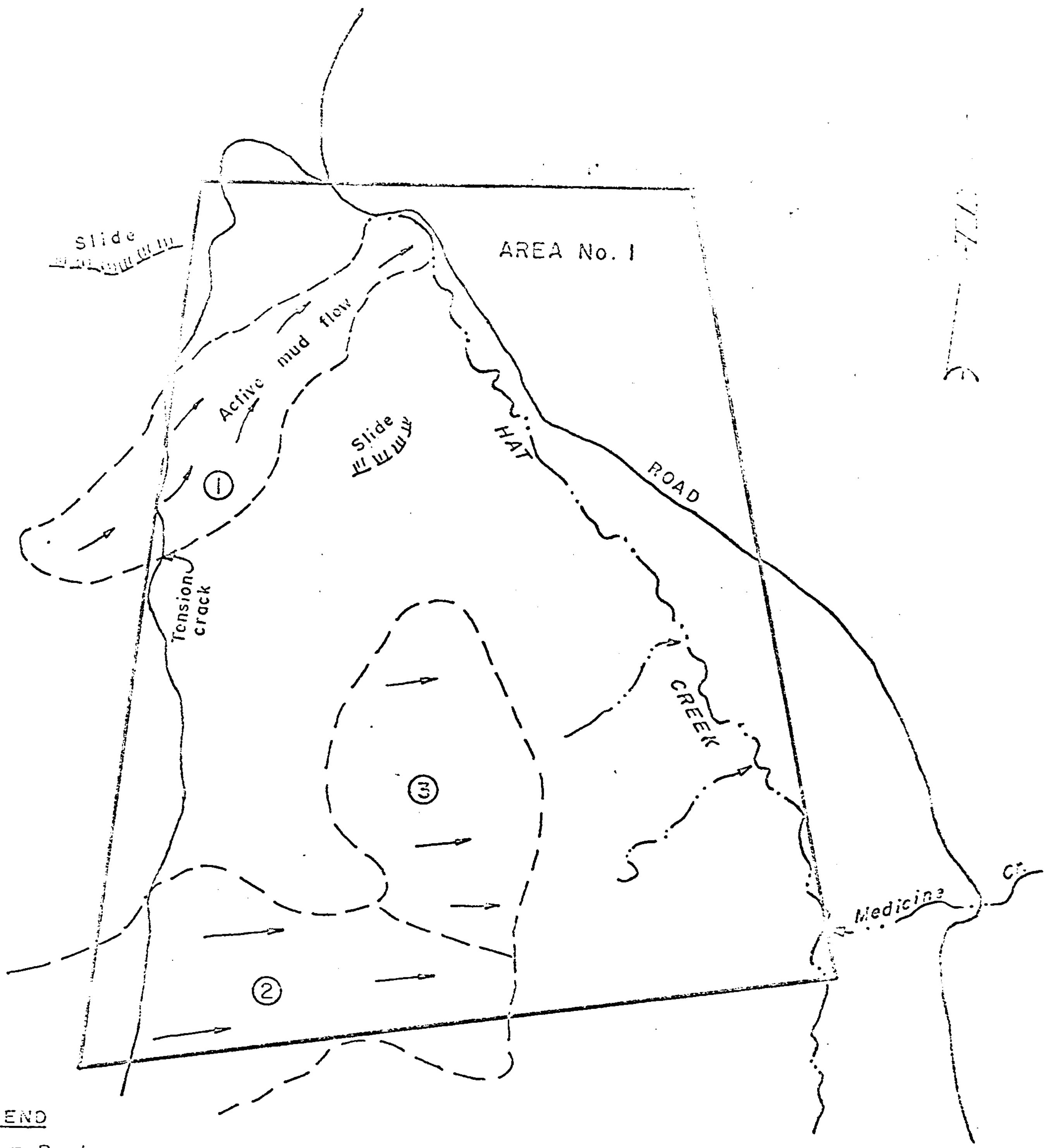
NOTE: Air photos are supplied by:

McELHANNEY SURVEYING & ENGINEERING LTD.,
Photogrammetric Division,
1200 West Pender Street,
Vancouver 1, B.C.

The individual numbers are on the drawings.

AIRPHOTO INTERPRETATION OVERLAY OF
MUDSLIDE REGIONS AT HAT CREEK
AIRPHOTO No. [illegible]

FIGURE [illegible]



LEGEND

Road

Limit of Area 1

Creek

Approximate Mudflow Boundary

Mudflow Designations

1. Active
2. Inactive
3. Possible

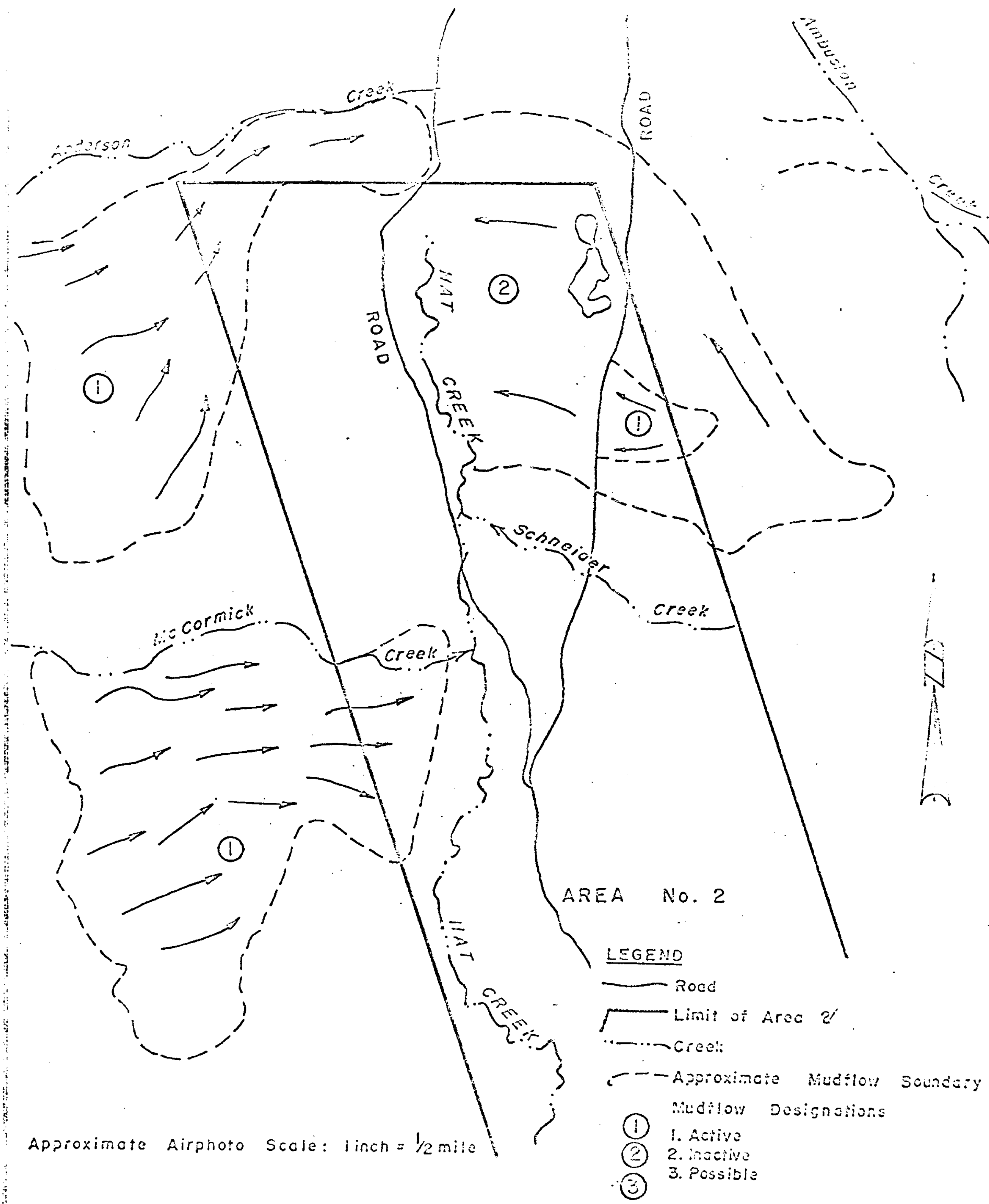
Approximate Airphoto Scale: 1 inch = 1/2 mile

Golder Associates

Drawn
App'd
Date

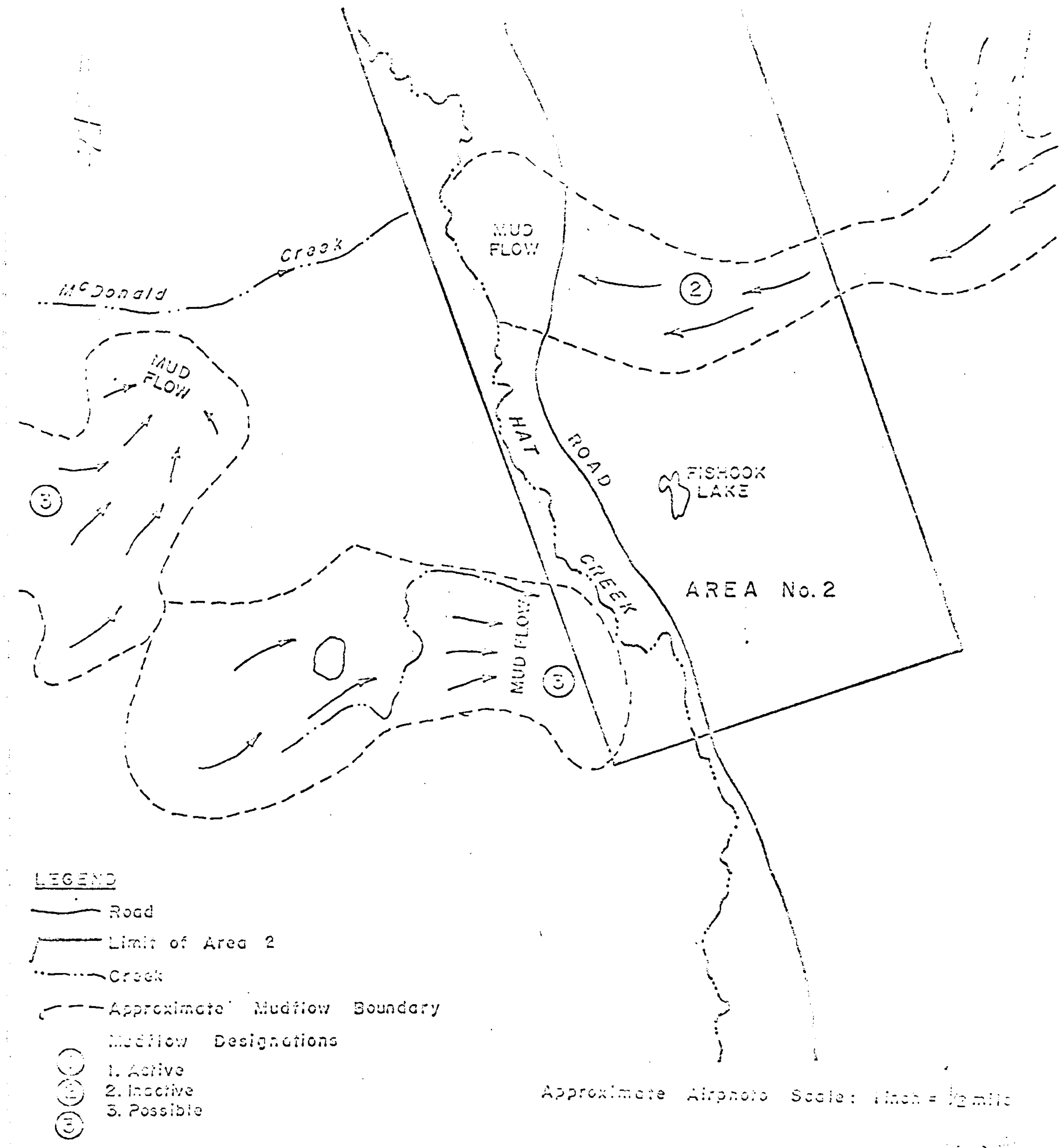
AIRPHOTO INTERPRETATION OVERLAY OF MUDSLIDE REGIONS AT HAT CREEK AIRPHOTO No. MA903-03155-0-3395

FIGURE II-2



Golder Associates

Drawn S.W.
App'd
Date



Golder Associates

9. PROPOSED BUDGET FOR FURTHER WORK

In the original Golder Associates proposal for geotechnical studies, attached to the PD-NCB Consultants Limited proposal dated September 16, 1975, the scope of the work to be carried out in Stage I was outlined. The work carried out in this stage is covered in this report.

It should be noted that the complexity of the superficial materials and the presence of potential mudslides necessitated a slight change of emphasis in the work and some work remains to be done to complete Stage I. This additional work will result in some over-run on the original budget of $15,000 allowed for Stage I.

The proposals which follow are for both Stage II and Stage III which relate to the detailed feasibility assessment of the mining area chosen for the commencement of mining at Hat Creek. These proposals are divided into three sections:

(1) Assessment of stability of the superficial deposits, including the inferred mudslides observed at the site.

(2) Assessment of stability of open pit slopes principally in the Coldwater sediments and coal deposits.

(3) Hydrology studies.

1. Stability of Superficial Slopes

To provide an assessment of the stability of slopes in superficial material, the following questions have to be considered:

(a) What is the lateral extent and thickness of such materials?

(b) What class and kind of material are they composed of?

(c) What are their shear strength characteristics?

(d) What are the piezometric elevations in the deposits?

To provide answers to these questions the following program of work is proposed.

(a) Soil drilling, approximately 15,000 feet, and surface dozer trenching.

(b) Field instrumentation consisting of piezometer and slope indicator installations.

(c) Laboratory classification and shear strength testing.

(d) Stability analyses to define pit slope angles in overburden.

(e) Preparation of a report presenting the Stage II data, our analyses, and an assessment of the results relative to stability of pit slopes in surficial overburden.

The estimated cost for the above work would be $400,000.

2. Stability of Pit Slopes

In order to arrive at specific design conclusions on pit slopes, side slopes for the access ramp and operating benches, it is necessary to log all existing and future core for geotechnical information, to determine structural orientations, carry out detailed air photo interpretation and rock strength tests.

This information will be used to carry out detailed stability analyses on the proposed pit slopes.

The estimated total cost of this work for the complete evaluation of the chosen pit site is $1,000,000.

3. <u>Hydrology Study</u>

At the request of Mr. Conrad Guelke of B.C. Hydro, the estimate for the hydrology study was based upon the terms of reference set out in a B.C. Hydro document dated September 2, 1975. This proposal, which is far more detailed than those given above, is attached.

EH:ctr
V75346

E. Hoek,
Principal, Golder Associates.

TABLE OF CONTENTS

SUMMARY

This proposal covers a three level program for the investigation of the subsurface hydrology of the Hat Creek Coal leases (British Columbia, Canada) held by B. C. Hydro & Power Authority.

The proposal sets out in some detail the recommended investigation approach, and the likely cost and timing associated with each level.

The Level I program is a preliminary evaluation, with little field work. Completion is scheduled for the end of January, 1976, at a cost of $17,000. This program will summarize and coalesce available hydrology information, and will identify more closely future investigatory requirements. A preliminary mining feasibility study will also be presented.

The Level II program is a more complete mine feasibility study, and will provide the hydrologic information necessary for preliminary mine design and for the hydrological environmental impact assessment. The program could be complete by the end of December 1976, at a cost of about $1 million. The program will have a major emphasis on field testing for parameters, and synthesis of the results into the hydrologic simulation models required for mine design, stabilization strategy evaluation, and environmental impact assessment.

The Level III program is a follow-up program required for longer term environmental monitoring, and for detailed mine design. No time or cost estimate for this level can be made at present, although such a program would be considerably smaller than the Level II program.

Golder Brawner & Associates Ltd. are well suited to conduct this program, being a British Columbian firm, with uniquely appropriate experience in large energy mineral hydrology and geotechnical studies.

INTRODUCTION

Three levels of study are considered here:

Level I - Preliminary study

Level II - Mine feasibility study

Level III - Baseline & pit design study

The preliminary study has been included as an option for completion by the end of January, 1976, and will cost $17,000. The mine feasibility study, which will involve considerable field work and will cost $996,000, could be completed by the end of December, 1976. The baseline and pit design study would follow Level II. Each study builds on the information developed during the precedent study.

This short report identifies the aims, methodologies, outputs, schedules, and costs of the first two study levels, and an outline of the third. It is a preliminary assessment, and the costs are developed on the assumption that the studies do not in general use holes drilled as a part of other programs. Obviously some economy over the reported costs will be obtainable if conjunctive use of drill holes and personnel can be achieved.

The overall scope of the study required was made available to us in the form of a B. C. Hydro document entitled "Draft Terms of Reference - Engineering Assignment to Hydroelectric Design Division - Ground Water Survey". A copy of this document is attached as Appendix 1.

The Level I study has already been presented as a separate item in Golder Associates' proposal for a preliminary geotechnical study of Hat Creek coal mining. It is re-presented in this proposal in greater detail for the sake of completeness.

SITE CONDITIONS AND INVESTIGATION PHILOSOPHY

Based on drilling and field experience on the site to date, we expect conditions at Hat Creek to be difficult for hydrologic investigations. Specifically, we expect that:

i. holes planned to depths of greater than 1,000 feet have less than a 50 per cent chance of reaching target depth.

ii. Packer testing will be complicated by the presence of materials which have an affinity for water, particularly montmorillonites.

iii. Piezometer installations may prove difficult to complete, particularly multiple installations in a single hole.

iv. Pumping well installations may prove to be exceedingly difficult, especially at depth.

As a result, we would suggest that both time and cost estimates should be regarded as being very approximate. Further, it will almost certainly be necessary to adjust the testing approach in the light of conditions encountered. This proposal contains our present view of a feasible, effective program, based on our experiences in similar conditions.

LEVEL I - PRELIMINARY HYDROLOGY STUDY (Cost Estimate $17,000)

A. SCOPE

The Level I study will concentrate on developing a model of likely hydrologic conditions and materials parameters based on existing information and readily obtainable data. The study orientation will be towards environmental and mining hydrology considerations. Field testing will have to be minimized in order to fit the program into the nominated timespan of six weeks from authorization to report presentation.

B. PROPOSED PROGRAM

1. Review of Available Information

All relevant information obtained in this and precedent studies will be collected and assessed. Data to be evaluated will include geology, topography, meteorology, lithology, surface hydrology, subsurface hydrology, drilling logs, geophysical information, air photo information, water conductivity, materials, tests, and historical information. A considerable amount of this information has been already assembled by Dolmage, Campbell and B. C. Hydro.

2. Site Visit and Investigation

A field reconnaisance to the site will be required to familiarize project personnel with field conditions and to perform a limited near-surface piezometer installation program. This program would involve the installation of a set of driven piezometers at a maximum of three locations to obtain an indication of near-surface (to 100 foot) piezometric profiles. If any drilling were being conducted, a set of piezometers would be located in these holes during drill string withdrawal.

Surface hydrology will not be thoroughly evaluated as the program will be conducted in midwinter, under snow and freezing conditions. However, the larger stream flow readings may be obtained through the surface ice.

3. Testing and Analysis

Selected core samples of rock and soil materials will be tested for permeability characteristics in the laboratory. Bulk material permeability estimates based on these tests will then be integrated with a simple geological model to allow the first analysis of mine inflow, mine dewatering conditions, and hydrological input into a crude pit slope analysis. While this analysis phase will be little more than conceptual, we feel it is essential for a first assessment of mining feasibility, and for the proper definition of critical areas for future investigation. A crude regional hydrologic impact analysis will also be performed if the data obtained above warrants it.

4. Report

The report will summarize:

- the state of knowledge of the hydrology of the Hat Creek area,
- the results of tests and information developed in the Golder Associates program,
- the preliminary mine feasibility, inflow, stability and dewatering analysis results,
- the preliminary hydrologic impact assessment if this is performed,
- recommendations for future investigation activities to obtain hydrology information necessary for adequate definition

of mining and environmental questions.

C. TIMING

The Level I program will require an elapsed time of six weeks from approval to completion. Providing approval were given by December 15, 1975, the final report could be produced by January 31, 1976. Field work would be performed during the first week in January, 1976 in this case.

LEVEL II - MINE FEASIBILITY HYDROLOGY STUDY (Cost Estimate $96,000)

Before a decision on mine feasibility, mining methods and pit design can be realistically undertaken, an adequate understanding of the site geotechnical conditions is appropriate. A significant portion of the input to this program must be provided by an hydrologic study. Some additional hydrologic work will be required for the detailed mine design and this is discussed as Level II.

A. OBJECTIVE AND SCOPE

The aim of the Level II study is to provide hydrologic input to mine design and cost estimating. The data developed must be capable of defining the surface and subsurface hydrologic conditions with sufficient accuracy that mine feasibility can be evaluated and an appropriate pit slope and pit slope control program can be designed and costed.

B. METHOD OF APPROACH

The method of approach will be to combine field parameter measurement with laboratory and numerical analysis.

A field program will be performed to evaluate hydrologic parameters and present piezometric conditions. A laboratory program will evaluate material properties from core samples.

Finally, the parametric information will be used to synthesize the hydrologic response to mining and to analyse the impact of various slope stabilizing strategies.

C. DETAILS OF FIELD PROGRAM

The detailed field program will have three primary objectives:

1. To evaluate the existing regional piezometric pressure regime.

2. To evaluate field values of permeability for the major materials encountered.

3. To evaluate the time dependent behaviour of the hydrologic system (i.e. evaluate the porosity and specific storage of the various media).

The field program will comprise packer testing, pumping tests, piezometer installations, instrumentation of the trial pit, and a detailed field reconnaisance.

For the first three of these activities a network of test wells will be required. We recommend the drilling of ten holes on or near each of two major sections, 7000'N and 15000'S, and additionally the use of six of the holes proposed in other programs along the valley outline. Hole depth will typically be 1,000 feet on the two sections, with the outer holes penetrating only 500 feet. New drilling will total about 15,000 feet. The arrangement of test holes is shown on Figure 1.

Finally, a network of shallow holes will be drilled near the trial pit, and an appropriate piezometric observation points installed.

In total, twenty-six deep holes at various locations will have hydrologic information available from them, with packer data in ten of the holes, and pump test responses at many of them.

The field testing will comprise the following:

1. Packer Tests

   These tests have the capability to identify the permeability of the media which make up short sections of the hole. They are relatively inexpensive and, when properly performed, give reproducible values of material permeability.

   We propose the exhaustive testing of ten coreholes, with up to a total of 100 tests being performed. The packer testing will provide the major definition of the permeability range of each of the major lithologic units of relevance to the project.

2. Pumping Tests

Although packer tests, when properly performed, offer the most inexpensive method of evaluating complex geohydrologies, some testing must be performed on a more regional basis in order to verify the spot testing, to evaluate the anisotropy of horizontal permeabilities, and to obtain an understanding of the porosity and/or the specific storage of the media encountered. This information is essential for the evaluation of any time dependent hydrological activity, particularly stabilization strategies.

We propose the performance of two such tests on this site - one in each of the proposed excavation locations. Each test will involve the drilling of a number of holes around a central pumping well, and completion of these holes with multi-level piezometers (probably six per hole). The main well will be rotary drilled, of large diameter (probably 12") and will require slotted casing and selective gravel pack screening.

3. Piezometer Installations

The most sensitive indicator of the regional hydrologic conditions is the undisturbed piezometric regime. The dynamic equilibrium condition of the piezometric pressure system indicates the direction of flow of water in the formation, the relative permeability of each bed, and the seasonal variation of those conditions. Simulation of the observed equilibrium piezometric pressure system can provide an excellent check on obtained permeability values, and simulation of the changes observed during a long term pumping test can identify time dependent behaviour parameters and further verify effective permeability values. The installation of observation points along the two sections utilized for packer testing is recommended.

4. Trial Pit

We understand that a trial open pit in coal is to be excavated in order to obtain fuel samples. Details of this pit have not been made available, however, we would propose to observe the hydrological impact of this feature as it develops, and after it has finished. Installation of observation piezometers in shallow holes will form part of our study, and the results of the investigation will be analysed using our finite element method flow analogue capability. We would require involvement in the planning stage of this trial pit if optimal hydrologic information is to be obtained.

5. Detailed Field Reconnaisance

In addition to the above testing, a detailed field reconnaisance will be undertaken to add to information currently available. This will concentrate on locating springs, quicksand areas, seeps, and other evidence relating to subsurface hydrology.

D. DETAILS OF DATA REDUCTION AND LABORATORY PROGRAM

Analysis of information will be concurrent with collection of field information. The analytical activities will have as their aim the assembly of a hydrologic model of the pit areas, and of the Hat Creek Basin.

1. Packer tests - deviation of permeabilities.
2. Pump Tests - analysis and simulation.
3. Piezometric information - refinement of simulator using

observed values.

4. Geology/Geophysical and other information - assembly and input to hydrology model.

5. Core permeability program - testing and estimation of actual bulk permeability values.

E. DETAILS OF ANALYTICAL PROGRAM

The predictive hydrology model will be used to analyse situations of mining hydrology significance. This phase will include analysis of hydrology of various pit slopes under undrained and drained conditions, and evaluation of the effectiveness of various water pressure reduction strategies.

Analyses will be performed with the aid of our in-house 2 and 3 dimensional finite element method hydrology simulation capability, which has been used with considerable effect on several recent hydrology studies of this type.

F. TIMING

The program set out above as the Level II study will require an elapsed time of one year. Of this, six months will be required for field work, and a further non-concurrent four month period will be needed for the completion of the analysis. We anticipate that mobilization will require an elapsed time of between one and two months. Thus, it is reasonable to expect that, if the study were initiated in January, 1976 it could be finished by December, 1976.

It should be noted that a preliminary report could be prepared in May, 1976, which would be useful in making minability decisions. This would include results of testing to that date and a preliminary version of the hydrology model for the basin. This report would probably fulfil some of the early needs with respect to pit design and stability control strategy.

G. OUTPUTS AND PROJECT CONTROL

In a project of this size and timespan a fixed series of check points and coordination meetings are essential. We would propose that such a control system be set up between B. C. Hydro, PD - NCB and ourselves prior to project initiation. We have found in similar projects that monthly meetings and quarterly reports are usually fairly near optimal for good communication.

LEVEL III - ENVIRONMENTAL BASELINE AND PIT DESIGN STUDY

At the conclusion of the Level II study there will remain two outstanding study areas.

1. Environmental Baseline

   The network of piezometric observation points developed in Level II will be adequate for this purpose. Accordingly, all that will be required for this program will be continued reading of the piezometers and development of a body of baseline information. This could readily be performed by personnel supplied by the client, with occasional engineering input from Golder Associates.

2. Pit Design Study

   When specific pit locations are chosen it may be necessary to make a more detailed study of the hydrology of the proposed pit walls.

We regard detailed planning and costing of this phase as inappropriate at this time. However, part of the Level II study will be to provide a program for Level III.

COSTING

Costing for the first two levels of this study has been estimated as follows:

	Manpower	Expenses	Total
Level I	$ 10,550	$ 6,450	$ 17,000
Level II	$368,000	$628,000	$996,000

Details of the way in which these figures were arrived at are included in Appendix II.

PERSONNEL

The key personnel involved in this project will be:

Principal and Project Director	Dr. E. Hoek
Project Engineer and Senior Hydrologist	A. Brown
Geologist	K. Rippere
Computer Analyst	I. Miller
Project Coordinator	B. Brooking

Support personnel will be drawn from the resources of the Golder group as necessary. Cirricula Vitae of the above personnel are attached.

EXPERIENCE

Golder Brawner & Associates has extensive experience of large hydrology projects for the coal and fossil fuel industries:

- Morwell Open Cut, Victoria, Australia. Study involved assessment of dewatering strategies for aquifers beneath a major lignite (brown coal) mine, and slope stabilization in lignite and clayey overburden. Golder Brawner & Associates were also retained for hydrology evaluation of the adjacent proposed Loy Yang Open Pit, which involved a major regional analysis of basin hydrology as well as stabilization studies.

- Black Thunder Project, Wyoming (Eastern Powder River Basin). Evaluation of regional subsurface hydrology for input into stability and environmental assessments for an open pit coal operation.
- Colorado Tract C-b Oil Shale Hydrology. Evaluation of mine inflow, underground mine stability, regional mine impact and environmental baseline for subsurface hydrology.
- Carbon Creek Coal Project, B. C. Evaluation of undisturbed regional subsurface and surface hydrology, and likely impact of strip and underground mining on hydrology.

This specific experience is in addition to a broad association with mining and civil engineering geotechnical projects throughout the world.

We trust that this proposal meets your present needs. Should you have any questions, please do not hesitate to call us.

Yours very truly,

GOLDER BRAWNER & ASSOCIATES LTD.

E Hoek

Per: Dr. E. Hoek, Principal

Adrian Brown

A. Brown, P. Eng.

AB/jh

V 75921

PROPOSED HYDROLOGY PROJECT
BOREHOLE LOCATIONS
LEVEL II PROGRAM
Figure 1
FAULT
INFERRED FAULT - Strong evidence
- Weak evidence
VOLCANIC DEPOSITS OF POSSIBLE RECENT AGE
DRILL HOLE
NO COAL
MAJOR COAL
PROPOSED DRILL HOLE
AREA UNDERLAIN BY COAL
Reference :- From plan supplied by Dolmage Campbell & Associates
Legend
New hole
Presently proposed Stage 3 hole (Dolmage Campbell)
Detail of Well Test Holes
Pumped Well
1000'
100'
Scale in feet
INDIAN RESERVE
No 1 PLANT SITE AREA
No 1 COAL DEPOSIT
No 2 COAL DEPOSIT
See Detail
See Detail
OUTLINE OF COAL LICENSES
LONGITUDINAL SECTION
WEST BOUNDARY FAULT
FAULT
7000 N
7000 N
Golder Associates
Drawn R.D.
App'd
Date Dec '72

APPENDIX I

TERMS OF REFERENCE

(Dated: 2nd September 1975)

DRAFT

TERMS OF REFERENCE

ENGINEERING ASSIGNMENT TO HYDROELECTRIC
DESIGN DIVISION
GROUNDWATER SURVEY

1.0 Provide engineering services to identify the groundwater conditions that could affect the design and development of an openpit coal mine in the Hat Creek Valley. The survey including field investigations, evaluation of laboratory analysis and mathematical modelling is to be completed in conjunction with the Stage 3 detailed drilling program, the mining of a coal sample, and the conceptual design of the mine.

2.0 The assignment is to include but not necessarily be limited to the following considerations:

2.1 An estimate of the magnitude of groundwater flow rates.

2.2 An evaluation of cleft water and seepage pressure conditions.

2.3 An evaluation of the groundwater table elevation and its variations due to climatic changes such as spring thaw and winter conditions.

2.4 An assessment of the groundwater flow response to surface hydrogeological changes and the influence of surficial deposits, vegetative covering and topography.

2.5 An investigation of the spatial and directional variations in permeability of materials in and around the openpit.

2.6 The influence on groundwater conditions of major geological features such as fault zones and impermeable seams.

3.0 Methods to be considered for obtaining and assessing data are to include:

3.1 Field Measurement Surveys

- installation of piezometers in diamond drill holes in and around the designated openpit area for obtaining water pressure data. Both short term and permanent monitoring installations are to be considered with frequency of readings depending on changing hydrogeologic conditions.
- pumping, falling head and/or borehole water packer permeability tests in diamond drill holes.
- field measurements as required to maximize the input of useful data generated prior to, during and after the excavation of a test trench for removal of a coal test burn sample. Measurements such as groundwater flows toward the excavation, required pit dewatering and water level changes in surrounding boreholes are to be included.

3.2 Evaluation of Laboratory Analysis

- assessment of the permeability of materials in and around the openpit by evaluation of laboratory permeability tests on diamond drill core and bulk samples.

3.3 Mathematical Modelling

- mathematical simulation of groundwater flow.
- establishment of a control grid of piezometers to assess the boundary conditions of the mathematical model and check predictions of pressure distribution within and around the openpit.

4.0 The assignment is to be completed in cooperation with the following plans and schedules:

4.1 Stage 3 Drilling - Dolmage Campbell and Associates Ltd. would be managing contractor and geological consultants for approximately 50,000 feet of diamond drilling. The detail drilling of the selected openpit is scheduled to commence in November 1975 after a decision is made whether to develop openpit No. 1 or 2. A maximum of three drilling rigs will be involved in the program including geophysical logging and related studies.

4.2 Mining Conceptual Design - work will commence in September 1975 and will require input from the groundwater survey to determine the influence of groundwater conditions on slope and pit base stability and dewatering requirements.

4.3 Mining Coal Sample - a 50,000 ton coal sample is to be mined between March and May 1976.

4.4 Laboratory Analysis - both Dolmage Campbell and Associates and the System Design Division are arranging for extensive coal and overlying material analysis. Specific testing for permeability, porosity and other useful data for groundwater assessment can be arranged in conjunction with drill core and bulk sample analysis.

5.0 A cost estimate of the assignment is to be submitted by 30 September 1975.

6.0 Prepare and submit a summary report in draft form by 1 May 1976 and in final form by 1 June 1976. Interim reports will be required as necessary to convey data for other studies.

7.0 The survey is to be controlled and coordinated by Mr. C. B. Guelke, Assistant Manager of the Generation Planning Department.

JCE

JCE:gj

2 September 1975

APPENDIX II

DETAILS OF COSTING

APPENDIX II

DETAILS OF COSTING

Details of costing are presented for Level I and Level II of the program, in order to indicate how totals were obtained. It should be noted that actual expenditures may not follow this breakdown, due to changes in approach or timing dictated by field conditions. The information here should therefore be regarded as informatory, rather than contractual.

Actual billing will be on the basis of the schedule of rates of the Association of Engineers of British Columbia, with expenses re-billed to the client at cost plus 5 per cent handling charge, as set out in the attached sheet (Appendix III).

A. LEVEL I

Costs are presented on a weekly basis, and include all anticipated charges for the Level 1 study.

1. Review of Available Information

Senior Hydrologist	- 1 week @ $1,200/wk.	$ 1,200.
Junior Engineer	- 2 weeks @ $ 800/wk.	1,600.
Communications		500.
		$ 3,300.

2. Site Visit

Senior Hydrologist	- 1 week @ $1,200/wk.	$ 1,200.
Junior Engineer	- 1 week @ $ 800/wk.	800.
Equipment (piezometers)		2,000.
Travel		200.
Accommodation		400.
Light Aircraft (Reconnaisance)		500.
		$5,100.

3. Testing and Analysis

Senior Hydrologist	- 1 week @ $1,200/wk.	$ 1,200.
Junior Engineer	- 1 week @ $ 800/wk.	800.
Technician	- 1 week @ $ 600/wk.	600.
Computer Analysis		2,000.
		$4,600.

4. Report

Senior Hydrologist	- 1 week @ $1,200/wk.	$ 1,200.
Junior Engineer	- 1 week @ $ 800/wk.	800.
Draftsperson	- 1 week @ $ 500/wk.	500.
Typist	- 1/2 week @ $400/wk.	200.
Reprographic (20 copies)		500.
		$ 3,200.

5. Management

Principal - 1/4 week @ $1,800/wk.	$ 450.
Communications	350.
	$ 800.
TOTAL (LEVEL 1)	$17,000.

B. LEVEL II

We have provided below a rough estimate of the order of cost of the proposed Level II program. It must be remembered that:

a. The program outlined is likely to be substantially modified in the light of the finings of the Level I study.

b. Field conditions may make sections of the program impossible to perform and may require major modification to the proposed program.

The cost estimate is broken up into the same general sections as the work, in order to show their relative contributions.

1. Field Program - Direct Costs

Drilling:	15,000 ft. @ $30/ft.	$450,000.
Packer Equipment Hire	100 @ $50./test	$ 5,000.
Piezometers:	5 x 26 @ $100 (total)	$ 13,000.
Casing:	2 x 1000 ft. @ $20/ft.	$ 40,000.
Well preparations:	2 x $15,000	$ 30,000.
		$538,000.

2. Field Program - Personnel

Senior Engineer	1 x 25 weeks @ $1,200/wk.	$ 30,000.
Junior Engineer	1 x 50 weeks @ $ 800/wk.	$ 40,000.
	1 x 25 weeks @ $ 800/wk.	$ 20,000.
Technician	1 x 50 weeks @ $ 600/wk.	$ 30,000.
Per Diem, transport, communication etc. ($40/manday allowed)		$ 42,000.
		$162,000.

3. Data Reduction and Laboratory Program

Senior Engineer	1 x 25 weeks @ $1,200/wk.	$ 30,000.
Junior Engineer	1 x 25 weeks @ $ 800/wk.	$ 20,000.
Technician	1 x 25 weeks @ $ 600/wk.	$ 15,000.
Computer Costs (lump sum estimate)		$ 15,000.
		$ 80,000.

4. Analytical Program and Report

Senior Engineer 2 x 25 weeks @ $1,200/wk.	$ 60,000.
Junior Engineer 2 x 25 weeks @ $ 800/wk.	$ 40,000.
Draftsperson 1 x 25 weeks @ $ 500/wk.	$ 20,000.
Typing/Clerical 1 x 10 weeks @ $ 400/wk.	$ 4,000.
Direct Report Cost 300 pp. @ $10/page	$ 3,000.
Computer Costs (lump sum estimate)	$ 20,000.
	$147,000.

5. Management of Project

Principal	5 weeks @ $1,800/wk.	$ 9,000.
Intermediate Engineer	50 weeks @ $1,000/wk.	$50,000.
Communications, Travel, etc.		$10,000.
		$69,000.
Total		$996,000.

The total is comprised of $538,000 in direct drilling and well installation cost, and $458,000 in personnel and associated expense. We have added the management category as an explicit expense, and we regard the proportion of managerical to engineering expense (7 per cent of total, 15 per cent of personnel and associated expense) as about optimal for a project of this size.

APPENDIX A

TERMS OF REFERENCE FOR HAT CREEK MINING STUDY

1. Provide engineering services to determine the feasibility and cost of a mine to produce 12 million short tons per year of coal for the 35 year life of a 2,000 MW thermal generating station. The normal maximum daily output of the mine would be 40,000 short tons.

2. The feasibility study shall include a preliminary assessment of reclamation procedures. The cost of reclamation is considered part of the mining cost.

3. The consultant would assess the field work and exploration to date. This data would be used for the following:-

   (a) Recommendation of further geological and geotechnical work.

   (b) Recommendation of the area(s) and plan for detail pit drilling to establish proven reserves and mineability.

   (c) The preparation of preliminary mine plans and sections.

   (d) Preliminary selection of waste and ash dump areas.

4. A preliminary assessment of the mining costs shall be made where information is adequate on the area(s) selected for detail drilling.

5. The mine shall be equipped as necessary to deliver coal to the thermal plant storage pile. Two alternatives shall be considered:-

   (a) Run of Mine with waste removed by selective mining.

(b) Coal mined and beneficiated.

6. Various open pit mining methods shall be studied and compared on the basis of cost, reliability and flexibility.

7. Coal cost estimates expressed in \$/ton and ¢/ton and ¢/10^6 Btu are to be calculated for the methods studied. Capital cost estimates shall be broken down to clearly itemise the component costs.

8. Cost estimates shall be in 1975 dollars and shall be broken down by years. The interest on capital and interest during construction shall be taken as 10%.

9. A project schedule shall be prepared on the basis of an earliest in-service date of the first generating unit of January 1983 with capacity reaching 2,000 MW by April 1985.

10. The report should contain a study outline and recommendations for further development of the mine including final mine site selection, preliminary design, and final design.

11. Prepare and submit an interim report on Items 3(a) and 3(b) by the end of November 1975. This report shall be based on geological and geotechnical assessments completed by that time.

Prepare and submit reports as follows:-

(a) Openpit No 1 - in draft form by the end of February 1976 and in final form by the end of March 1976.

(b) Openpit No 2 - in draft form by the end of April 1976 and in final form by the end of May 1976.

In addition, brief monthly reports shall be submitted of activities and cost to date.

12. The successful execution of the work would require considerable local knowledge with respect to labour, transportation and legislation. The consultant would be required to carry out the major part of the work in western Canada or establish an affiliation with a suitable local company to ensure adequate site dependent input.

13. The study is to be controlled and co-ordinated by the Assistant General Manager, Engineering Group, of BC Hydro or his appointee.

14. Provide budget capital estimates as requested by BC Hydro during the course of the study.

APPENDIX B

LIST OF DOCUMENTS AND DRAWINGS HELD BY PD-NCB AS AT 19TH NOVEMBER 1975

REPORTS

1. Thermal Coal Resources - Hat Creek. Summary Report No 1 Openpit Deposit, DCA, 1 Jan 1975.

2. No 1 Deposit Rock Mechanics Data, Klohn Leonoff Consultants Ltd., Aug 1975.

3. No 1 Deposit Coal Quality Data, DCA, 27 June 1975.

4. No 1 Deposit Statistical Tables of Proximate Analysis Data, DCA, 15 July 1975.

5. Ultimate and Washability Data, DCA, 15 Sept 1975.

6. Memorandum re Mining Costs, 25^{o} Openpit, from Douglas D. Campbell to Dr. Ellis, 7 April 1975.

7. Hat Creek Coalfield, BC Minister of Mines Report, 1924.

8. Progress Report No 1, No 2 Coal Deposit, DCA, 19 Sept 1975.

9. Engineering Tests on Hat Creek Samples, Klohn Leonoff Consultants Ltd., 12 Dec 1974.

10. Engineering Tests on Hat Creek Samples, Klohn Leonoff Consultants Ltd., 20 Jan 1975.

11. Groundwater and Related Conditions in the Upper Hat Creek Valley, Memorandum from P.J. Street to Dr. D.D. Campbell, 25 Feb 1975.

12. Hat Creek Project Progress Report, System Engineering Division BCH, Aug 1975.

13. Preliminary Environmental Impact Study of the Proposed Hat Creek Development, BC Research and DCA, 1975.

14. Memorandum on Hat Creek Exploration Programme Progress Report, BCH, July 1975.

15. Hat Creek - Coal Fired Thermal Power Plant, Feasibility Report and Cost Estimate (Report No 104), BCH, July 1975.

16. Report on Pollution Control Objectives for the Mining, Mine-Milling and Smelting Industries of BC, Dept. of Lands, Forests and Water Resources, Dec 1973.

17. Hat Creek No 1 Deposit Field Rock Slaking Tests, Winter 1975, origin not stated.

BOREHOLE AND GEOLOGICAL DATA

1. Loring Laboratories Ltd., Analysis Report No 10151, DDH No 75-74, Sample No 74-37 and 74-41, Oct 1975.

2. Provisional Geological Legend, 6 March 1975.

3. Record of Completed Drill Holes 1974, DDH 74-23 to DDH 74-48, and RH 74-1 and RH 74-2. Record of Completed Drill Holes 1975, DDH 75-49 to DDH 75-61, and RH 75-3 and RH 75-4.

4. Drill Record Sheets for DDH 74-25, 74-48, 75-49, 75-63, 75-68, 75-72, 75-75, 75-76, 75-77, 75-79, 75-81, 75-85, 75-88, 75-89, 75-90, 75-91, 75-92, 75-93.

5. Telex from Wrightenga 3 Nov 1975, giving drilling information of DDH 75-93, 75-94, 75-95, 75-96, 75-97, 75-98, 75-99, 75-100, 75-101, 75-102, 75-103.

6. Sepia copies of graphical logs for drill holes DDH 74-01, 74-02, 74-23, 74-24, 74-25, 74-26, 74-27, 74-28, 74-29, 74-30, 74-31, 74-32, 74-33, 74-34, 74-35, 74-36, 74-37A, 74-38, 74-39, 74-40, 74-41, 74-42, 74-43, 74-44, 74-45, 74-46, 74-47, 74-48, 75-03, 75-R4, 75-49, 75-50, 75-51, 75-52, 75-53, 75-54, 75-55, 75-56, 75-57, 75-58, 75-59, 75-60, 75-61, 75-62, 75-63, 75-64, 75-64A, 75-65, 75-66, 75-67, 75-68, 75-69, 75-70, 75-71, 75-73, 75-74, 75-77, 75-78, 75-79A, 75-80, 75-82, 75-83, 75-83A, 75-84, 75-86, 75-87.

7. Prints of the above logs with the exception of DDH 75-68.

8. Roke Engineering Geophysical logs - sepia. DDH RH 74-01, 74-02, 74-25, 74-26, 74-37A, 74-38, 74-39, 74-40, 74-41, 74-42, 74-43, 74-44, 74-45, 74-46, 74-47, 74-48, RH 75-03, RH 75-04, 75-49, 75-50, 75-51, 75-53, 75-56, 75-57, 75-58, 75-59, 75-60, 75-61, 75-62, 75-63, 75-67, 75-68, 75-69, 75-70, 75-71, 75-72, 75-73, 75-74, 75-76, 75-77, 75-78, 75-79, 75-79A, 75-80, 75-81, 75-82, 75-83, 75-84, 75-85, 75-87, 75-88, 75-89, 75-90, 75-91.

9. Paper prints of each of the above.

10. Loring Laboratories Ltd., Analysis Report No 10636, DDH 75-68, 20 Oct 1975.

11. Commercial Testing and Engineering Company, Analysis Report No 64-11201-11206, DDH 75-73, 3 Nov 1975.

12. Loring Laboratories Ltd., Analysis Report No 10635, DDH 75-74, 20 Oct 1975.

13. Loring Laboratories Ltd., Analysis Report No 10634, DDH 75-82, 20 Oct 1975.

14. Loring Laboratories Ltd., Analysis Report No 10078, DDH 75-53-401, undated.

15. Ultimate samples containing internal unanalysed waste sections, DDH 75-57, DCA and Commercial Testing and Engineering Company, Analysis Report No 64-10450, 25 Aug 1975.

16. Ultimate samples containing internal unanalysed waste sections, DDH 75-60, DCA and Commercial Testing and Engineering Company, Analysis Report No 67-10453, 25 Aug 1975.

17. Commercial Testing and Engineering Company, Analysis Report No 67-10456, DDH 75-61, 25 Aug 1975.

18. Loring Laboratories Ltd., Analysis Report No 10087, DDH 75-62, undated.

19. Ultimate samples containing internal unanalysed waste sections, DDH 75-68, DCA and Loring Laboratories Ltd., Analysis Report No 10090, undated.

20. Ultimate samples containing internal unanalysed waste sections, DDH 75-73, DCA and Commercial Testing and Engineering Company, Analysis Report No 64-10732, 1 Oct 1975.

21. Loring Laboratories Ltd., Analysis Report No 10464, DDH 75-74, 1 Oct 1975.

22. Commercial Testing and Engineering Company, Analysis Report No 64-10741, DDH 75-77, 1 Oct 1975.

23. Loring Laboratories Ltd., Analysis Report No 10465, DDH 75-80, 1 Oct 1975.

24. Ultimate samples containing internal unanalysed waste sections, DDH 75-82 DCA and Loring Laboratories Ltd., Analysis Report No 10466, DDH 75-82, 1 Oct 1975.

PLANS AND SECTIONS

1. Transparencies and prints of topographical plans, scale 1 in = 400 ft, sheets 1 to 8.

2. Transparent plan of No 1 and 2 areas showing topography, scale 1 in = 2000 ft.

3. Print of plan showing drilling progress, period Aug 16 - 31, 1975, scale 1 in = 4000 ft.

4. Print of plan showing exploration areas, Stage 2c and 3a Drilling, Aug 1975, scale 1 in = 4000 ft.

5. Print of plan showing No 1 Openpit Deposit Drill Holes, 1 March 1975, scale 1 in = 800 ft.

6. Prints and transparency of plan Hat Creek Exploratory Drilling 1975, July 1975, scale 1 in = 2000 ft.

7. Prints and transparency of No 2 Deposit Drill Exploration, Sept 1975, scale 1 in = 800 ft.

8. Prints and transparency of No 2 Deposit Geophysical Anomalies, Sept 1975, scale 1 in = 800 ft.

9. Prints and transparency of BC Hydro Preliminary Stratum Contour Map, 24 June 1975, scale 1 in = 400 ft.

10. Geological Longitudinal Section Upper Hat Creek Valley, July 1975, scales 1 in = 2000 ft horizontal, 1 in = 500 ft vertical.

11. Sepia prints of cross sections on a scale of 1 in = 400 ft at 7000S, 10000S, 15000S, 16500S, 19000S, 22000S, 25000S, 30000S, 7000N, and proposed drilling sections at 7000S, 10000S, 15000S, 16500S, 19000S, 22000S, 25000S.

12. Paper prints of sections on a scale of 1 in = 400 ft at 7000S, 10000S, 15000S, 16500S, 19000S, 22000S, 25000S, 30000S, 13500N, 10500N, 6500N, 3000N.

HAT CREEK

BUDGETS
(000 C$)

	1975-6		1976-7	1977-8	1978-9	1979-80	1980-1	1981-2	1982-3	1983-4	1984-5
Geological Exploration of Preferred Pit Locations	(200) 200	Selection of Mining Area	(600) 550								
Geotechnical Exploration of Preferred Pit Locations	(25) 25		(25) 25								
Hydrological Study	50		550	400							
Geological Exploration to complete the Selected Mine Design			(1000) 1040	(1000) 1000							
Geotechnical Exploration to complete the Selected Mine Design			(400) 834	(500) 800							
Mining Engineering to complete the Selected Mine Design			(600) 600	(600) 600							
Mining Equipment and Services					(300) 300	(6500) 6500	(8300) 8300	(7800) 7800	(7400) 7400		
Continuing Exploration and Geotechnical					(300) 300	(300) 300	(200) 300	(100) 300	(100) 300		
Creek Diversion						(1000) 1000	(1000) 1000				
Incline Box-cut and Pre-production Overburden Removal (Cash only)						(6300) 6300	(8200) 8200	(7000) 7000	(7000) 7000		

NOTE: 1. 20th October budget shown in brackets for comparison.

2. All figures in 1975 prices.

APPENDIX D

DRILLING DENSITY

Item		Area 1	Area 2 North	Area 2 South
Area,	ft^2	13,900,000	12,400,000	9,600,000
Average depth to coal,	ft	100	150	225
In situ thickness of coal including partings (to depth of 600 ft),	ft	500	450	375
In situ tonnage including partings,	tons	316,000,000	248,000,000	163,000,000
Boreholes to slope toe,	No	26	6	6
Boreholes 1974/1975,	No	9	6	6
Surface drilling density,	ft^2/bh	500,000	2,070,000	1,600,000
Borehole spacing,	ft	700	1,400	1,300
Coal yield (approx),	tons/ft	22,000	90,000	120,000
Proposed addition boreholes in coal (each 600 ft),	No	84	70	-
Proposed drilling density. All drilling (borehole spacing),	ft	350	-	-
Drilling recent (borehole spacing),	ft	400	400	-

Note: Tonnages exclude coal in slopes

APPENDIX "B"

LIST OF DOCUMENTS AND DRAWINGS RECEIVED BY PD-NCB FROM 19TH NOVEMBER, 1975 TO 23RD FEBRUARY, 1976

APPENDIX "B"

LIST OF DOCUMENTS AND DRAWINGS RECEIVED BY PD-NCB FROM 19TH NOVEMBER, 1975 TO 23RD FEBRUARY, 1976

Reports

(i) "Hat Creek Basin Gravity Survey", C.A. Ager and Associates Limited (data).

Geological Data

(ii) Approximate locations and intersections of DDH 111, 112, 112A, 113, 114 and 115.

(iii) E-W and N-S sections of No 1 coal deposit, 1 in to 200 ft.

(iv) Down-hole geophysical logs for drill holes 76-111, 112, 113, 115 to 119 inclusive.

(v) Hat Creek drill progress location plan, complete for Stage 3A.

(vi) Geological logs for drill holes 75-105 to 75-110 (excluding 105A), and 76-111,112, 112A, 113 to 116 inclusive.

(vii) Geological logs (draft) for drill holes 76-117, 118 and 119.

(viii) Field slaking tests on cores from drill holes 75-60, 64A, 80, 84 and 87.

(ix) Record of completed drill holes, Table 1974 and Table 1975.

(x) Geological logs for drill holes 75-79, 92, 96, 97, 101 to 110 inclusive.

(xi) Down-hole geophysical logs for drill holes 75-105, 107 to 110 inclusive.

APPENDIX "C"

UNDERGROUND MINING

APPENDIX "C"

UNDERGROUND MINING

INTRODUCTION

1. It could become desirable to extract the lower portion of the No 1 coal deposit by underground means. Possible reasons for this could include:-

   (i) Economically more favourable than continuing to opencast.

   (ii) Too dangerous to continue deepening the openpit because of increasing slope instability.

RESERVES AND PRODUCTION

2. It is expected that the need to extract the coal by underground means would not occur until after the initial pit to the 2,400-ft elevation has been completed. The quantity of coal remaining below the 2,400-ft level is estimated to be about 560 million tons.

3. The quantity required at the thermal power station for one year at that time would be 12 million tons. The estimated waste content for underground mining, which would necessarily be unselective mining, is 22% and therefore the annual tonnage of in-situ coal to be won would be about 15 million tons.

UNDERGROUND MINING METHODS

4. Mining the Hat Creek deposit by underground methods could only be done after an assessment of the following hazards:-

   (i) Gas emission; ventilation of the areas being mined would vary depending on the mining method. Methane drainage might be necessary and might even be profitable if the methane were supplied to the power station.

   (ii) Spontaneous combustion; liability of the coal to spontaneous combustion would restrict the mining methods that could be used.

   (iii) Strata control in conditions of weak rock, particularly slaking mudstones and groundwater.

5. Various broad methods of mining are examined further in the light of the above and other factors.

Traditional Coal Mining Methods

6. These comprise longwall and room and pillar systems. Either system could be considered if the coal were mined in lifts and the area below filled solid.

7. Gas emission and spontaneous combustion could be controlled without undue difficulty. Lack of homogeneity in the coal and rock mixture of the deposit would cause difficulties. These methods would be very expensive in both capital and operating costs. It is unlikely that adequate production could be maintained.

Other Methods

Open and Shrinkage Stoping

8. Neither of these methods would be practicable due to weak roof and walls.

Cut and Fill

9. This method could be used with expensive support systems and close-up filling of extracted areas. This method would be expensive and could not produce the high annual output needed.

Cascade Mining

10. This method is more suitable for tabular deposits inclined between 25° and 45°.

Hydraulic Mining

11. This method is suitable for moderately inclined seams with strong floor and well-defined roof. The absence of these features, together with the slaking effect of mudstone, render this method unlikely.

Caving Methods

12. In order to produce the high output required, these would seem to be the most practicable and economic underground mining methods. If they were to be used successfully, the following conditions would require to be met:-

(i) the coal must not be liable to spontaneous combustion

(ii) high methane emission must be controlled by drainage methods

(iii) the coal must break-up in a manner suitable for caving after blasting. The clay material mixed with the coal may make caving impracticable.

13. Both block caving and sub-level caving have high initial development costs but high production and productivity are subsequently possible, given suitable mining conditions. Control of the roof and sides would be crucial throughout mining and new methods, involving cable support of large exposed areas or other similar techniques, would need to be investigated. Closer control of blasting, mining and caving should be possible with sub-level caving rather than with block caving, therefore sub-level caving is considered further.

CONCEPTUAL METHOD

14. The area would be worked with four main shafts and four ventilation shafts. Levels would be at 300-ft intervals and coal would be blocked out on levels working downwards. A suitable block size would be 300-ft high by 300-ft long by 150-ft wide (625,000 tons). Each block would be split by sub-levels and coal loaded by front-end loader and shuttle car or load-haul-dump equipment into winzes leading to the level. This would be a typical sub-level caving layout. Detailed planning of the extraction procedure would enable 24 such blocks to be caved per year. To produce 43,000 tons per day would require 10 loading areas, requiring 350 men on a three-shift basis, but if, as expected, difficulty is experienced in supporting the shafts and roadways, considerably more labour would be required.

15. The problems of shaft-sinking alone would be formidable and weak rock material, structural weakness and water may well require freezing methods. Similarly, the problems of support of ore packs, main haulage roads, development roads, drain pockets, etc could well be insurmountable.

APPENDIX "D"

BASIC FINANCIAL DATA

APPENDIX "D"

BASIC FINANCIAL DATA

1. Inflation Rate (expressed as percentage) -

Year	Labour and Materials
1975	base
1976	10
1977	10
1978	10
1979	10
1980	5 thereafter

Basic estimates are in 1975 dollars, without discounting, as stated in the Terms of Reference. The above inflation figures have been used for inflated cash-flow curves and any other general statements or comparisons which have been necessary.

2. Discount Rate -

evaluations carried out at 15%.

3. Power Costs -

an incremental energy cost of 10 mills/kWh has been used.

4. Interest During Construction Calculation -

IDC in year N is half the interest rate x the Nth year capital cost, plus the interest rate x the accumulated expenditures, including previous IDC in the preceding N-1 years. I = 10%.

5. Corporate Overhead Rate -

A corporate overhead of 5% has been added to the uninflated direct costs of the project (see Plate 42).

6. Tax and Debt Equity Ratio -

Assumed that no tax paid by BC Hydro and that financial structure is 100% debt.

7. Sales Taxes -

Federal sales tax of 12% not applied on generator or mining equipment used for power generation. Provincial sales tax omitted.

8. The Provincial Royalty -

Assumed to be 75¢ per long ton (equivalent to 67¢ per short ton).

APPENDIX "E"

LABOUR REQUIREMENTS

APPENDIX "E"

LABOUR REQUIREMENTS

MINE LABOUR FORCE

1. The mining labour requirements and associated wage costs related to the development and operation of the mine, but excluding building and construction work, are shown in Table XVIII. There is gradual build-up in the mine labour force to some 660 employees by the end of stage 6, each major section of the mine operation being as follows:-

Mobile mining equipment	500
Fixed installations	130
Infrastructure	32
Total	662

2. This total requirement increases to 987 during stage 7 and subsequently to 1,148 at the beginning of stage 8, all of the projected increase occurring in the mobile mining equipment section.

3. Although there are upwards of ten labour grades in most current union contracts, in Table XVIII these have been simplified into the three main categories given below, the percentage figures relating to the relative proportions of each at stage 6.

Equipment operators	-	53%
Maintenance personnel	-	37%
General mining labour	-	10%

4. The wage rates are in line with those pertaining to current mining contracts in the general area. These would seem to be slightly lower than those paid in some of the coal mining operations in the Eastern Kootenays so it is possible that some upward revision might be necessary, depending on the particular union contract. In this respect, it must be emphasised that the type of mining envisaged is closer to that practised in metal mining so that local mining traditions could have a greater influence on rates and conditions than coal mining traditions in other areas.

MANAGERIAL, TECHNICAL AND ADMINISTRATIVE STAFF

5. The numbers of managerial, technical and administrative staff relating to each stage of the project are shown in Table XIX.

6. In developing the schedule it has been assumed that the mine will be operated as a captive mine and that BCH will provide the corporate planning, marketing, financial and data processing functions from head office. Again the salary costs would seem to be in line with those currently paid at other mines in the general area. It should not be overlooked, however, that the projected rapid growth in coal mining in Western Canada will cause demand for senior coal mining personnel to exceed supply and that severe upward pressures on coal mining salaries could result.

Explanatory Note

There are a number of minor discrepancies in the following tables which are, however, without significance in the context of the conceptual design and the level of confidence of the estimates.

TABLE I

BASIC PLANNING DATA

Density of in-situ coal	-	1.39 short tons/bank yd^3
Swell	-	25%
Density of in-situ waste in coal	-	1.87 short tons/bank yd^3
Swell	-	50%
Density of ROM coal	-	1.29 short tons/bank yd^3
Density of superficial deposits	-	1.56 short tons/bank yd^3
Swell	-	15%
Density of claystone (assumed wet)	-	1.87 short tons/bank yd^3
Swell	-	40%
Density of conditioned ash	-	1.2 short tons/yd^3 (loose)
Estimated in-situ waste content	-	22%
Estimated waste extraction by selective mining	-	15%
Waste remaining in ROM coal	-	7%
Working days per year	-	350
Hours per shift	-	8
Teams of men	-	4
No of producing shifts per week	-	20
No of maintenance shifts per week	-	1

TABLE II

OPEN PIT NO. 1 VOLUMES, TONNAGES AND RATIOS

A. Referred to In situ Coal

Pit Stage	Pit Floor Elevation ft	Superficials		Overlying Waste		Total Waste		In situ Coal		Cumulative Stripping Ratio byd^3/st	Instantaneous Stripping Ratio byd^3/st
		Stage 10^6 byd^3	Cumulative 10^6 byd^3	Stage 10^6 byd^3	Cumulative 10^6 byd^3	Stage 10^6 byd^3	Cumulative 10^6 byd^3	Stage 10^6 st	Cumulative 10^6 st		
1	2,680	19	19	-	-	19	19	2	2	9.5	7.6
2	2,600	25	44	9	9	34	53	23	25	2.1	2.5
3	2,400	13	57	20	29	33	86	46	71	1.2	1.4
4	2,400	46	103	32	61	78	164	83	154	1.1	1.1
5	2,400	61	164	51	112	112	276	100	254	1.1	1.4
6	2,400	73	237	90	202	163	439	96	350	1.2	2.3
7	2,400	87	324	156	358	243	682	72	422	1.6	6.4
8	2,400	50	374	117	475	167	849	30	452	1.9	7.1
9	1,500	-	-	-	-	779	1,628	460	912	1.8	12.7

B. Referred to ROM Coal

Pit Stage	Pit Floor Elevation ft	Superficials		Overlying and Segregated Waste		Total Waste		ROM Coal		Cumulative Stripping Ratio byd^3/st	Instantaneous Stripping Ratio byd^3/st
		Stage 10^6 byd^3	Cumulative 10^6 byd^3	Stage 10^6 byd^3	Cumulative 10^6 byd^3	Stage 10^6 byd^3	Cumulative 10^6 byd^3	Stage 10^6 st	Cumulative 10^6 st		
1	2,680	19	19	-	-	19	19	2	2	9.5	8.2
2	2,600	25	44	11	11	36	55	20	22	2.5	2.7
3	2,400	13	57	24	35	37	92	39	61	1.5	1.5
4	2,400	46	103	38	73	84	176	71	132	1.3	1.2
5	2,400	61	164	59	132	120	296	85	217	1.4	1.5
6	2,400	73	237	97	229	170	466	82	299	1.6	2.4
7	2,400	87	324	162	391	249	715	61	360	2.0	6.9
8	2,400	50	374	120	511	170	885	25	385	2.3	7.7
9	1,500	-	-	-	-	816	1,701	390	775	2.2	13.7

Note: 1. Specific gravities used:

Waste	1.87 st/byd^3
In situ coal	1.39 st/byd^3
ROM coal	1.29 st/byd^3

2. Stripping ratio defined as waste production (byd^3) : coal production (short tons)
3. Cumulative stripping ratio based on total pit volumes to end of stage
4. Instantaneous stripping ratio based on the volumes mined in the last increment, at the end of the stage
5. Segregated waste assumed to be 15% out of the 22% waste in the in situ coal.

TABLE III

PRODUCTION SCHEDULE - YEARLY AND CUMULATIVE

Item	Unit	Stage 1				Stage 2		Stage 3			Stage 4						Stage 5					
		1979-80	1980-81	1981-82	1982-83	1983-84	1984-85	1985-86	1986-87	1987-88	1988-89	1989-90	1990-91	1991-92	1992-93	1993-94	1994-95	1995-96	1996-97	1997-98	1998-99	1999-2000
Power station requirements (6,000 Btu/lb coal)	10^6 tons	-	-	-	1	4	7	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
In situ coal production	10^6 tons																					
Yearly		0.1	0.2	0.5	0.5	5.1	8.6	16	15	16	15	15	16	15	16	15	15	16	15	16	15	15
Cumulative		0.1	0.3	0.8	1.3	6.4	15	31	46	62	77	92	108	123	139	154	169	185	200	216	231	246
ROM coal production	10^6 tons																					
Yearly		0.1	0.2	0.4	0.4	4.3	7.6	13	13	13	13	13	14	13	13	13	13	13	13	13	13	13
Cumulative		0.1	0.3	0.7	1.1	5.4	13	26	39	53	65	78	92	105	118	131	144	157	170	183	196	209
Segregated waste	10^6 tons																					
Yearly		-	-	0.1	0.1	0.8	1	3	2	3	2	2	2	2	3	2	2	3	2	3	2	2
Cumulative		-	-	-	0.2	1	2	5	7	10	12	14	16	18	21	23	25	28	30	33	35	37
Superficials	10^6 yd^3																					
Yearly		6	6	6	6	6	6	7	8	4	5	8	7	7	10	11	9	8	10	10	10	10
Cumulative		6	12	18	24	30	36	43	51	55	60	68	75	82	92	103	112	120	130	140	150	160
Pit waste	10^6 yd^3																					
Yearly		-	-	-	-	3	3	5	7	7	7	7	7	7	6	6	6	7	7	8	9	9
Cumulative		-	-	-	-	3	6	11	18	25	32	39	46	53	59	65	71	78	85	93	102	111
Total waste (excluding segregated)	10^6 yd^3																					
Yearly		6	6	6	6	9	9	12	15	11	12	15	14	14	16	17	15	15	17	18	19	19
Cumulative		6	12	18	24	33	42	54	69	80	92	107	121	135	151	168	183	198	215	233	252	271
Yearly stripping ratio (in situ coal basis)		60	30	12	12	1.8	1	0.8	1	0.7	0.8	1	0.9	0.9	1	1.1	1	0.9	1.1	1.1	1.3	1.3
Instantaneous stripping ratio (in situ coal basis)		-	-	-	7.6	6.5	5	2.4	2	1.7	1.3	1.2	1.1	1.1	1.1	1.1	1.1	1.2	1.2	1.3	1.3	1.4
Pit waste plus segregated waste	10^6 tons																					
Yearly		-	-	0.1	0.1	6.4	7	12.4	15	16	15	15	15	15	13	13	13	16	15	18	19	19
Cumulative		-	-	0.1	0.2	6.6	13.6	26	41	57	72	87	102	117	130	143	156	172	187	205	224	243
Yearly stripping ratio (ROM coal basis)		60	30	15	15	2.1	1.2	1.1	1.2	1	1	1.2	1.1	1.1	1.4	1.4	1.2	1.3	1.4	1.5	1.5	1.5
Instantaneous stripping ratio (ROM coal basis)		-	-	-	9.5	7.7	5.7	3	2.3	2.1	1.5	1.4	1.3	1.3	1.3	1.3	1.3	1.5	1.5	1.6	1.5	1.5

/continued

TABLE III

(continued)

Item	Unit	Stage 6						Stage 7					Stage 8		Stage 9						
		2000-01	2001-02	2002-03	2003-04	2004-05	2005-06	2006-07	2007-08	2008-09	2009-10	2010-11	2011-12	2012-13	2013-14	2014-15	2015-16	2016-17	2017-18	2018-19	2019-20
Power station requirements (6,000 Btu/lb coal)	10^6 tons	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	8	5-1=4*
In situ coal production	10^6 tons																				
Yearly		16	15	16	15	15	16	15	16	15	16	15	15	16	15	15	16	15	16	11	5
Cumulative		262	277	293	308	323	339	354	370	385	401	416	431	447	462	477	493	508	524	535	540
ROM coal production	10^6 tons																				
Yearly		14	13	13	13	13	13	13	13	13	14	13	13	13	13	13	13	13	14	9	4
Cumulative		223	236	249	262	275	288	301	314	327	341	354	367	380	393	406	419	432	446	455	459
Segregated waste	10^6 tons																				
Yearly		2	2	3	2	2	3	2	3	2	2	2	2	3	2	2	3	2	2	2	1
Cumulative		39	41	44	46	48	51	53	56	58	60	62	64	67	69	71	74	76	78	80	81
Superficials	10^6 yd^3																				
Yearly		10	13	11	12	12	12	12	19	18	18	19	23	27							
Cumulative		170	183	194	206	218	230	242	261	279	297	316	339	366							
Pit waste	10^6 yd^3																				
Yearly		11	11	15	15	15	15	22	32	33	33	39	42	60							
Cumulative		122	133	148	163	178	193	215	247	280	313	352	394	454							
Total waste (excluding segregated)	10^6 yd^3																				
Yearly		21	24	26	27	27	27	34	51	51	53	56	65	87							
Cumulative		292	316	342	369	396	423	457	508	559	612	668	733	820							
Yearly stripping ratio (in situ coal basis)		1.3	1.6	1.6	1.8	1.8	1.7	2.3	3.2	3.7	3.8	4.5	4.9	5.1							
Instantaneous stripping ratio (in situ coal basis)		1.4	1.5	1.6	1.8	1.9	2.0	2.3	3.7	4.4	5.3	6.1	6.7	7.0							
Pit waste plus segregated waste	10^6 tons																				
Yearly		23	23	30	30	30	31	43	63	64	64	75	81	115							
Cumulative		266	289	319	349	379	410	453	516	580	644	719	800	915							
Yearly stripping ratio (ROM coal basis)		1.6	1.9	2.1	2.2	2.2	2.2	2.7	4.1	4	3.9	4.4	5.2	6.8							
Instantaneous stripping ratio (ROM coal basis)		1.6	1.8	2.0	2.1	2.1	2.5	2.7	4.6	5.1	6.1	7	7.5	7.7							

* 1 million tons stock consumed

Notes:

1. No 1 generator starts July 1983, closes March 2018
2. No 2 generator starts July 1984, closes July 2019
3. No 3 generator starts April 1985, closes March 2020
4. Reserves of 2,400 ft pit depleted during year 2013-14

TABLE IV

MOBILE MINING EQUIPMENT REQUIREMENTS

Item	Stage 1				Stage 2		Stage 3			Stage 4						Stage 5						Stage 6						Stage 7					Stage 8	
	1979-80	1980-81	1981-82	1982-83	1983-84	1984-85	1985-86	1986-87	1987-88	1988-89	1989-90	1990-91	1991-92	1992-93	1993-94	1994-95	1995-96	1996-97	1997-98	1998-99	1999-00	2000-01	2001-02	2002-03	2003-04	2004-05	2005-06	2006-07	2007-08	2008-09	2009-10	2010-11	2011-12	2012-13
<u>Coal</u>																																		
Drills and compressors	1	1	1	1	2	3	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
Shovels	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3
Trucks	1	1	1	1	2	4	6	6	6	6	6	7	7	7	7	8	8	8	8	8	8	8	8	8	8	8	8	9	9	10	10	10	10	10
Bulldozers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2
Wheeldozers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water tankers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Diesel tankers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance vehicles	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Graders	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Pick-up trucks - 1 ton	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	3
Sump pumps	2	2	2	2	2	2	4	4	4	4	4	4	4	4	4	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8	8
Explosives trucks	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2
<u>Segregated Waste</u>																																		
Shovels		Use Coal	Shovel		1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Trucks	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2
<u>Pit Waste</u>																																		
Drills and compressors	-	-	-	-	1	1	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	4	4	5	5	5	5	7	10	11	11	13	14	20
Shovels	-	-	-	-	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	4	4	4	4	5	7	7	7	8	9	13
Trucks	-	-	-	-	2	2	4	6	6	6	6	6	6	6	6	6	7	7	8	9	9	11	11	15	15	15	15	23	34	35	35	41	48	68
Bulldozers	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	3	4	4	4	5	5	6
Wheeldozers	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water tankers	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	3	4	4	4	4	5	5
Diesel tankers	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	3	4	4	4	7	7	7
Maintenance vehicles	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	3	4	6	6	8	8	10
Graders	-	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	4	4	5	5	6	6	6
Pick-up trucks - 1 ton	-	-	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	4	4	5	5	6	6	9
Explosives trucks	-	-	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	3	3
<u>Superficials</u>																																		
Scrapers	6	6	6	6	6	6	6	7	4	5	7	7	7	9	10	9	8	10	10	10	10	10	12	12	12	12	12	14	20	20	20	21	27	31
Pushers/bulldozers	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	4	4	4	4	4	4	5	5	5	5	6	6
Water tankers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2
Diesel tankers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	3	3
Maintenance vehicles	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	4	4
Graders	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	4	4
Compactors	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	3	3

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TABLE V

SCHEDULE OF MOBILE MINING EQUIPMENT - INITIAL AND REPLACEMENT COSTS

($$10^3$)

Item	Stage 1		Stage 2		Stage 3		Stage 4		Stage 5		Stage 6		Stage 7		Stage 8		Total Number of Machines	Total Cost
	1977-1983		1983-1985		1985-1988		1988-1994		1994-2000		2000-2006		2006-2011		2011-2013			
	No	Cost	No	Cost	No	Cost	No	Cost	No	Cost	No	Cost	No	Cost	No	Cost		
<u>Coal</u>																		
Drills and compressors	2	200	3	300	8	800	15	1,500	15	1,500	15	1,500	12	1,200	5	500	75	7,500
Shovels	1	1,390	-	-	1	1,390	1	1,390	2	2,780	1	1,390	2	2,780	-	-	8	11,120
Trucks	2	1,040	4	2,080	10	5,200	20	10,400	24	12,480	24	12,480	26	13,520	11	5,720	121	62,920
Bulldozers	2	460	1	230	2	460	3	690	3	690	3	690	4	920	2	460	20	4,600
Wheeldozers	2	320	1	160	2	320	3	480	3	480	3	480	2	320	1	160	17	2,720
Water tankers	1	210	1	210	1	210	1	210	2	420	1	210	1	210	1	210	9	1,890
Diesel tankers	1	30	1	30	1	30	1	30	2	60	1	30	1	30	1	30	9	270
Maintenance vehicles	1	30	1	30	1	30	1	30	2	60	1	30	1	30	1	30	9	270
Graders	2	340	1	170	2	340	3	510	3	510	3	510	2	340	1	170	17	2,890
Pick-up trucks - 1 ton	2	20	2	20	2	20	2	20	4	40	1	10	3	30	1	10	17	170
Explosives trucks	1	40	1	40	1	40	1	40	1	40	1	40	3	120	1	40	10	400
Sump pumps	2	60	-	-	2	60	2	60	8	240	6	180	2	60	-	-	22	660
Sub-total		4,140		3,270		8,900		15,360		19,300		17,550		19,560		7,330		95,410
<u>Segregated Waste</u>																		
Shovels	-	-	1	1,390	-	-	1	1,390	-	-	1	1,390	-	-	-	-	3	4,170
Trucks	-	-	1	520	2	1,040	3	1,560	3	1,560	4	2,080	4	2,080	2	1,040	19	9,880
Sub-total		-		1,910		1,040		2,950		1,560		3,470		2,080		1,040		14,050
<u>Pit Waste</u>																		
Drills and compressors	-	-	1	100	4	400	6	600	9	900	16	1,600	34	3,400	22	2,200	92	9,200
Shovels	-	-	1	1,390	1	1,390	1	1,390	1	1,390	3	4,170	6	8,340	6	8,340	19	26,410
Trucks	-	-	2	1,040	10	5,200	18	9,360	23	11,960	38	19,760	91	47,320	67	34,840	249	129,480
Bulldozers	-	-	1	230	2	460	3	690	3	690	5	1,150	10	2,300	6	1,380	30	6,900
Wheeldozers	-	-	1	160	2	320	3	480	3	480	3	480	2	320	1	160	15	2,400
Water tankers	-	-	1	210	1	210	1	210	2	420	2	420	4	840	2	420	13	2,730
Diesel tankers	-	-	1	30	1	30	1	30	2	60	3	90	8	240	3	90	19	570
Maintenance vehicles	-	-	1	30	1	30	1	30	2	60	1	30	4	120	8	240	18	540
Graders	-	-	2	340	4	680	6	1,020	6	1,020	9	1,530	13	2,210	6	1,020	46	7,820
Pick-up trucks - 1 ton	-	-	2	20	2	20	2	20	2	20	4	40	7	70	7	70	26	260
Explosives trucks	-	-	1	40	1	40	1	40	2	80	2	80	3	120	2	80	12	480
Sub-total		-		3,590		8,780		13,870		17,080		29,350		65,280		48,840		186,790
<u>Superficials</u>																		
Scrapers	12	4,440	6	2,220	10	3,700	24	8,880	30	11,100	36	13,320	48	17,760	31	11,470	197	72,890
Pushers/bulldozers	6	1,380	3	690	6	1,380	9	2,070	9	2,070	12	2,760	10	2,300	6	1,380	61	14,030
Water tankers	1	210	1	210	1	210	1	210	2	420	1	210	3	630	1	210	11	2,310
Diesel tankers	1	30	1	30	1	30	1	30	2	60	1	30	3	90	2	60	12	360
Maintenance vehicles	1	30	1	30	1	30	1	30	3	90	1	30	2	60	3	90	13	390
Graders	2	340	1	170	2	340	3	510	5	850	6	1,020	7	1,190	4	680	30	5,100
Compactors	2	320	1	160	2	320	3	480	3	480	3	480	5	800	3	480	22	3,520
Sub-total		6,750		3,510		6,010		12,210		15,070		17,850		22,830		14,370		98,600
Total		10,890		12,280		24,730		44,390		53,010		68,220		109,750		71,580		394,850
10% Initial spares		646		726		662		166		264		828		2,875		2,658		
Depreciation/stage, $		9,840		9,470		20,400		44,690		52,750		68,940		100,950		60,280		367,320
Average Depreciation, $/short ton		-		0.79		0.52		0.57		0.68		0.87		1.53		2.32		0.97
Average Depreciation, ¢/10^6 Btu		-		7		2		5		6		8		14		21		9

TABLE VI

SCHEDULE OF TYPICAL EQUIPMENT

Category	Type	Manufacturer	Model	Capacity
Shovel	Electric	Bucyrus Eyrie	195	15 yd^3
Drills	Compressed Air	Gardner Denver	3100A	4-in holes
Compressors	Diesel	Gardner Denver	SP600	600 ft^3/min
Off-highway trucks	Diesel	Wabco	150 C	117 tons coal
Bulldozers	Diesel	Caterpillar	D9H	-
Wheeldozers	Diesel	Caterpillar	824	-
Graders	Diesel	Caterpillar	16G	-
Scrapers	Diesel	Caterpillar	666	41 bank yd^3
Compactors	Diesel	Caterpillar	825	-
Water tanker	Diesel	Caterpillar	631	10,000 US gals

Note: The manufacturer's name and the model numbers have been given to enable production details and costs to be specified concisely. They are not intended to indicate any preference.

TABLE VII

MOBILE MINING EQUIPMENT - COST ALLOCATION

Activity	% Total Cost	Machine Type	% Total Cost
Superficials	25	Shovels	11
Pit waste	47	Trucks	51
Coal extraction	24	Scrapers	19
Segregated waste	4	Drills and compressors	4
		Bulldozers	6
		Graders	4
		Others	5
	100		100

TABLE VIII

ELECTRIC POWER - CONSUMPTION AND COSTS

Item	Ultimate Pit Installed kW	1979-80			1980-81			1981-82			1982-83			1983-84			1984-85			1985-86			1986-87		
		MWh	MVA Max Demand	Cost 10^3 \$	MWh	MVA Max Demand	Cost 10^3 \$	MWh	MVA Max Demand	Cost 10^3 \$	MWh	MVA Max Demand	Cost 10^3 \$	MWh	MVA Max Demand	Cost 10^3 \$	MWh	MVA Max Demand	Cost 10^3 \$	MWh	MVA Max Demand	Cost 10^3 \$	MWH	MVA Max Demand	Cost 10^3 \$
<u>Mobile Mine Equipment</u>																									
Shovels	11,300	3,500	0.78	35.0	3,500	0.78	35.0	3,500	0.78	35.0	3,500	0.78	35.00	3,500	0.78	35.00	3,500	0.78	35.00	7,000	1.56	70.00	7,000	1.56	70.00
Pumps	1,792	2,240	0.25	22.4	2,240	0.25	22.4	2,240	0.25	22.4	2,240	0.25	22.40	2,240	0.25	22.40	2,240	0.25	22.40	4,480	0.50	44.80	4,480	0.50	44.80
Pit and segregated waste:																									
Shovels	700	-	-	-	-	-	-	-	-	-	-	-	-	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	10,500	2.00	105.00
Sub-totals	13,792	5,740	1.03	57.4	5,740	1.03	57.4	5,740	1.03	57.4	5,740	1.03	57.40	12,740	2.59	127.40	12,740	2.59	127.40	18,480	3.62	184.80	21,980	4.06	219.80
<u>Fixed Installations</u>																									
In pit conveyors	4,830	-	-	-	-	-	-	-	-	-	-	-	-	2,180	2.15	21.80	4,025	3.90	40.25	19,500	5.38	195.00	19,500	5.38	195.00
Crusher feeder stations	1,347	-	-	-	-	-	-	-	-	-	-	-	-	5,450	0.56	54.50	10,900	1.12	109.00	16,550	1.68	165.50	16,550	1.68	165.50
Stockpile conveyors, crushers, etc.	4,379	-	-	-	-	-	-	4,550	1.26	45.5	4,550	1.26	45.50	11,790	3.26	117.90	11,790	3.26	117.90	17,700	5.00	177.00	17,700	5.00	177.00
Boom stacker	696	-	-	-	-	-	-	935	0.26	9.35	935	0.26	9.35	1,870	0.52	18.70	1,870	0.52	18.70	2,800	0.72	28.00	2,800	0.72	28.00
Bucket wheel reclaimer	1,100	-	-	-	-	-	-	-	-	-	-	-	-	2,220	0.61	22.20	2,220	0.61	22.20	4,440	1.22	44.40	4,440	1.22	44.40
Waste and ash disposal:																									
Waste to dump conveyors	3,409	-	-	-	-	-	-	-	-	-	5,700	1.62	57.00	5,700	1.62	57.00	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50
Ash conveyor	90	-	-	-	-	-	-	-	-	-	-	-	-	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62
Boom stackers	1,400	-	-	-	-	-	-	-	-	-	2,820	0.78	28.20	2,820	0.78	28.20	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40
Miscellaneous:																									
Lighting	100	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02
Communications	5	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20
Sub-totals	17,356	422	0.11	4.22	422	0.11	4.22	5,907	1.63	59.07	14,427	4.03	144.27	32,814	10.01	328.14	50,979	15.30	509.79	81,164	19.89	811.64	81,164	19.89	811.64
<u>Infrastructure</u>																									
Hat Creek diversion pumps	1,335	4,940	1.48	49.40	4,940.	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40
Buildings and roads:																									
Administration	150																								
Stores and workshops	475	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20
Changing house	50																								
Services, etc:																									
Water/sewage	450	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40
Sub-totals	2,460	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00
Totals	33,608	15,662	3.87	156.62	15,662	3.87	156.62	21,147	5.39	211.47	29,667	7.79	296.67	55,054	15.33	550.54	73,219	20.62	732.19	109,144	26.24	1,091.44	112,644	26.68	1,126.44
System maximum demand (Diversity 0.75)			2.90			2.90			4.05			5.85			11.50			15.50			19.70			20.00	
Energy consumption, kWh/short ton		156.00			78.50			52.70			74.50			12.80			9.60			8.30			8.70		
Energy cost, \$/short ton		1.56			0.78			0.52			0.74			0.13			0.09			0.08			0.09		
Energy cost, ¢/10^6 Btu		14			7			5			7			1			1			1			1		

/continued

TABLE VIII
(continued)

Item	1987-88			1988-89			1989-90			1990-91			1991-92			1992-93			1993-94			1994-95		
	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $
<u>Mobile Mine Equipment</u>																								
Shovels	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00
Pumps	4,480	0.50	44.80	4,480	0.50	44.80	4,480	0.50	44.80	4,480	0.50	44.80	4,480	0.50	44.80	4,480	0.50	44.80	4,480	0.50	44.80	8,960	1.00	89.60
Pit and segregated waste:																								
Shovels	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00
Sub-totals	21,980	4.06	219.80	21,980	4.06	219.80	21,980	4.06	219.80	21,980	4.06	219.80	21.980	4.06	219.80	21,980	4.06	219.80	21,980	4.06	219.80	26,460	4.56	264.60
<u>Fixed Installations</u>																								
In pit conveyors	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00
Crusher feeder stations	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50
Stockpile conveyors, crushers, etc.	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00
Boom stacker	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00
Bucket wheel reclaimer	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40
Waste and ash disposal:																								
Waste to dump conveyors	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50
Ash conveyor	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62
Boom stackers	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40
Miscellaneous:																								
Lighting	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02
Communications	20	0.01	0.02	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20
Sub-totals	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64
<u>Infrastructure</u>																								
Hat Creek diversion pumps	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40
Buildings and roads:																								
Administration Stores and workshops Changing house	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20
Services, etc:																								
Water/sewage	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40
Sub-totals	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00
Totals	112,644	26.68	1,126.44	112,644	26.68	1,126.44	112,644	26.68	1,126.44	112,644	26.68	1,126.44	112,644	26.68	1,126.44	112,644	26.68	1,126.44	112,644	26.68	1,126.44	117,124	27.18	1,171.24
System maximum demand (diversity 0.75)		20.00			20.00			20.00			20.00			20.00			20.00			20.00			20.40	
Energy consumption, kWh/short ton	8.70			8.70			8.70			8			8.70			8.70			8.70			9		
Energy cost, $/short ton	0.09			0.09			0.09			0.08			0.09			0.09			0.09			0.09		
Energy cost, ¢/10^6 Btu	1			1			1			1			1			1			1			1		

/continued

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TABLE VIII

(continued)

Item	1995-96			1996-97			1997-98			1998-99			1999-2000			2000-01			2001-02			2002-03		
	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $
Mobile Mine Equipment																								
Shovels	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00
Pumps	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60
Pit and segregated waste·																								
Shovels	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00	14,000	2.65	140.00	14,000	2.65	140.00	17,500	3.05	175.00
Sub-totals	26,460	4.56	264.60	26,460	4.56	264.60	26,460	4.56	264.60	26,460	4.56	264.60	26,460	4.56	264.60	29,960	5.21	299.60	29,960	5.21	299.60	33,460	5.61	334.60
Fixed Installations																								
In pit conveyors	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00
Crusher feeder stations	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50
Stockpile conveyors, crushers, etc.	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00
Boom stacker	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00
Bucket wheel reclaimer	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40
Waste and ash disposal:																								
Waste to dump conveyors	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50
Ash conveyor	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62
Boom stackers	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40
Miscellaneous:																								
Lighting	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02
Communications	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20
Sub-totals	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64
Infrastructure																								
Hat Creek diversion pumps	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40
Buildings and roads:																								
Administration Stores and workshops Changing house	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20
Services, etc:																								
Water/sewage	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40
Sub-totals	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00
Totals	117,124	27.18	1,171.24	117,124	27.18	1,171.24	117,124	27.18	1,171.24	117,124	27.18	1,171.24	117,124	27.18	1,171.24	120,624	27.83	1,206.24	120,624	27.83	1,206.24	124,124	28.23	1,241.24
System maximum demand (diversity 0.75)		20.40			20.40			20.40			20.40			20.40			20.80			20.80			21.20	
Energy consumption, kWh/short ton	9			9			9			9			9			8.50			9.20			9.50		
Energy cost, $/short ton	0.09			0.09			0.09			0.09			0.09			0.09			0.09			0.10		
Energy cost, ¢/10^6 Btu	1			1			1			1			1			1			1			1		

/continued

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TABLE VIII

(continued)

Item	2003-04 MWh	2003-04 MVA Max Demand	2003-04 Cost 10^3 \$	2004-05 MWh	2004-05 MVA Max Demand	2004-05 Cost 10^3 \$	2005-06 MWh	2005-06 MVA Max Demand	2005-06 Cost 10^3 \$	2006-07 MWh	2006-07 MVA Max Demand	2006-07 Cost 10^3 \$
Mobile Mine Equipment												
Shovels	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00	7,000	1.56	70.00
Pumps	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60
Pit and segregated waste:												
Shovels	17,500	3.05	175.00	17,500	3.05	175.00	17,500	3.05	175.00	21,000	3.46	210.00
Sub-totals	33,460	5.61	334.60	33,460	5.61	334.60	33,460	5.61	334.60	36,960	6.02	369.60
Fixed Installations												
In pit conveyors	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00
Crusher feeder stations	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50
Stockpile conveyors, crushers, etc.	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00
Boom stacker	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00
Bucket wheel reclaimer	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40
Waste and ash disposal:												
Waste to dump conveyors	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50
Ash conveyors	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62
Boom stackers	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40
Miscellaneous:												
Lighting	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02
Communications	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20
Sub-totals	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64
Infrastructure												
Hat Creek diversion pumps	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40
Buildings and roads:												
Administration, Stores and workshops, Changing house	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20
Services, etc:												
Water/sewage	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40
Sub-totals	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00
Totals	124,124	28.23	1,241.24	124,124	28.23	1,241.24	124,124	28.23	1,241.24	127,624	28.64	1,276.24
System maximum demand (diversity 0.75)		21.20			21.20			21.20			21.50	
Energy consumption, kWh/short ton	9.50			9.50			9.50			9.80		
Energy cost, \$/short ton	0.10			0.10			0.10			0.10		
Energy cost, ¢/10^6 Btu	1			1			1			1		

Item	2007-08 MWh	2007-08 MVA Max Demand	2007-08 Cost 10^3 \$	2008-09 MWh	2008-09 MVA Max Demand	2008-09 Cost 10^3 \$	2009-10 MWh	2009-10 MVA Max Demand	2009-10 Cost 10^3 \$	2010-11 MWh	2010-11 MVA Max Demand	2010-11 Cost 10^3 \$
Mobile Mine Equipment												
Shovels	7,000	1.56	70.00	10,500	2.00	105.00	10,500	2.00	105.00	10,500	2.00	105.00
Pumps	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60	8,960	1.00	89.60
Pit and segregated waste:												
Shovels	28,000	4.25	280.00	28,000	4.25	280.00	28,000	4.25	280.00	31,500	4.60	315.00
Sub-totals	43,960	6.81	439.60	47,460	7.25	474.60	47,460	7.25	474.60	50,960	7.60	509.60
Fixed Installations												
In pit conveyors	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00	19,500	5.38	195.00
Crusher feeder stations	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50	16,550	1.68	165.50
Stockpile conveyors, crushers, etc.	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00	17,700	5.00	177.00
Boom stacker	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00	2,800	0.72	28.00
Bucket wheel reclaimer	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40	4,440	1.22	44.40
Waste and ash disposal:												
Waste to dump conveyors	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50	13,750	3.82	137.50
Ash conveyors	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62	362	0.40	3.62
Boom stackers	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40	5,640	1.56	56.40
Miscellaneous:												
Lighting	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02	402	0.10	4.02
Communications	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20	20	0.01	0.20
Sub-totals	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64	81,164	19.89	811.64
Infrastructure												
Hat Creek diversion pumps	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40	4,940	1.48	49.40
Buildings and roads:												
Administration, Stores and workshops, Changing house	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20	2,720	0.75	27.20
Services, etc:												
Water/sewage	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40	1,840	0.50	18.40
Sub-totals	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00	9,500	2.73	95.00
Totals	134,624	29.43	1,346.24	138,124	29.87	1,381.24	138,124	29.87	1,381.24	141,624	30.22	1,416.24
System maximum demand (diversity 0.75)		22.05			22.40			22.40			22.70	
Energy consumption, kWh/short ton	10.30			10.60			9.90			10.90		
Energy cost, \$/short ton	0.10			0.11			0.10			0.11		
Energy cost, ¢/10^6 Btu	1			1			1			1		

/continued

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TABLE VIII

(continued)

Item	2011-12			2012-13		
	MWh	MVA Max Demand	Cost 10^3 $	MWh	MVA Max Demand	Cost 10^3 $
Mobile Mine Equipment						
Shovels	10,500	2.00	105.00	10,500	2.00	105.00
Pumps	8,960	1.00	89.60	8,960	1.00	89.60
Pit and segregated waste:						
Shovels	35,000	5.00	350.00	49,000	6.55	490.00
Sub-totals	54,460	8.00	544.60	68,460	9.55	684.60
Fixed Installations						
In pit conveyors	19,500	5.38	195.00	19,500	5.38	195.00
Crusher feeder stations	16,550	1.68	165.50	16,550	1.68	165.50
Stockpile conveyors, crushers, etc.	17,700	5.00	177.00	17,700	5.00	177.00
Boom stacker	2,800	0.72	28.00	2,800	0.72	28.00
Bucket wheel reclaimer	4,440	1.22	44.40	4,440	1.22	44.40
Waste and ash disposal:						
Waste to dump conveyors	13,750	3.82	137.50	13,750	3.82	137.50
Ash conveyors	362	0.40	3.62	362	0.40	3.62
Boom stackers	5,640	1.56	56.40	5,640	1.56	56.40
Miscellaneous:						
Lighting	402	0.10	4.02	402	0.10	4.02
Communications	20	0.01	0.20	20	0.01	0.20
Sub-totals	81,164	19.89	811.64	81,164	19.89	811.64
Infrastructure						
Hat Creek diversion pumps	4,940	1.48	49.40	4,940	1.48	49.40
Buildings and roads:						
Administration Stores and workshops Changing house	2,720	0.75	27.20	2,720	0.75	27.20
Services, etc:						
Water/sewage	1,840	0.50	18.40	1,840	0.50	18.40
Sub-totals	9,500	2.73	95.00	9,500	2.73	95.00
Totals	145,124	30.62	1,451.24	159,124	32.17	1,591.24
System maximum demand (diversity 0.75)		23.00			24.10	
Energy consumption, kWh/short ton	11.20			12.20		
Energy cost, $/short ton	0.11			0.12		
Energy cost, ¢/10^6 Btu	1			1		

TABLE IX

SUMMARY OF ELECTRICAL ENERGY COSTS
(10^3 \$)

Item	Stage 1					Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-20	2000-06	2006-11	2011-13
Mobile Mine Equipment	57.40	57.40	57.40	57.40	229.60	127.40	127.40	254.80	624.40	1,318.80	1,587.60	1,937.60	2,268.00	1.229.20
Fixed Installations	4.22	4.22	59.07	144.27	211.78	328.14	509.79	837.93	2,434.92	4,869.84	4,869.84	4,869.84	4,058.20	1.623.28
Infrastructure	95.00	95.00	95.00	95.00	380.00	95.00	95.00	190.00	285.00	570.00	570.00	570.00	475.00	190.00
Totals	156.62	156.62	211.47	296.67	821.38	550.54	732.19	1,282.73	3,344.32	6,758.64	7,027.44	7,377.44	6.801.20	3,042.48
Average Cost, \$/st rom	-	-	-	-	-	0.13	0.09	0.11	0.09	0.09	0.09	0.09	0.10	0.12
Average Cost, ¢/10^6 Btu	-	-	-	-	-	1	1	1	1	1	1	1	1	1

TABLE X

SCHEDULE OF EQUIPMENT - FIXED INSTALLATIONS AND CAPITAL COST

(10^3 \$)

Item	Ultimate Pit Quantity	Unit Cost including Erection	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	1983-84	1984-85
Coal Handling										
Primary crusher feeder station	3	1,400	-	-	-	140	420	1,400	1,260	980
Main incline conveyors	3	3,270	-	-	-	654	1,308	4,905	654	2,289
Interchange station	1	2,000	-	-	-	132	264	1,002	132	470
Conveyor to crusher station	2	873	-	-	-	87	175	698	175	611
Stockpile cross conveyor	3	426	42	84	300	42	84	342	84	300
Stocking conveyor	3	1,200	120	240	840	120	240	960	240	840
Boom stacker	3	2,250	225	450	1,575	225	450	1,800	450	1.575
Bucket wheel reclaimer	2	3,000	-	-	-	300	600	2,400	600	2,100
Reclaim conveyor	2	1,255	-	-	-	125	250	1,005	250	880
Conveyor stockpile to thermal plant silos	2	873	-	-	-	87	174	699	174	612
Ground hopper and conveyor	2	200	-	-	-	20	40	160	40	140
Secondary crushers	3	1,200	120	240	840	120	240	960	240	840
Dozer and compactor	1+1	300	30	60	210	-	-	-	-	-
Sub-total	-	-	537	1,074	3,765	2,052	4,245	16,331	4,299	11,637
Waste and Ash Disposal										
Interchange conveyor	3	218	21	42	155	21	42	176	42	155
Distribution conveyor (a)	1	1,200	120	240	840	-	-	-	-	-
Distribution conveyor (b)	1	1,200	-	-	-	120	240	840	-	-
Distribution conveyor (c)	1	328	-	-	-	32	64	232	-	-
Boom spreader/tripper	2	3,100	310	620	2,170	310	620	2,170	-	-
Dump conveyor	2	2,020	202	404	1,414	202	404	1,414	-	-
Ash conveyor and bunker	1	110	-	-	-	11	22	77	-	-
Dozer	1	140	14	28	98	-	-	-	-	-
Sub-total	-	-	667	1,334	4,677	696	1,392	4,909	42	155
Miscellaneous										
Waste bench flexible cables	Per mile	130	-	26	234	-	26	234	-	-
Transformers and switchgear			14.5	130.5	-	-	20	180	34.5	310.5
Overhead line distribution	Per mile	120	126	294	-	-	-	-	-	-
Fixed high tension cables	Per yard	0.04	7	63	-	-	8	72	16	144
Lighting and ancillaries	-	-	125	-	125	-	-	125	-	125
Communications	-	-	-	-	70	-	-	70	-	70
Cut off drains	-	-	-	-	50	-	-	-	-	-
Sub-total	-	-	272.5	513.5	479	-	54	681	50.5	649.5
Total	-	-	1,476.5	2.921.5	8,921	2,748	5,691	21,921	4.391.5	12,441.5
Initial spares (10%)	-	-	147.65	292.15	892.1	274.8	569.1	2,192.1	439.15	1,244.15
Total including spares	-	-	1,624	3,214	9,813	3,023	6,260	24,113	4,831	13,686
Depreciation	-	-	162	483	1,464	1,766	2,492	4,903	5,386	6.755
Depreciation, \$/short ton	-	-	-	-	-	-	-	-	1.25	0.89
Depreciation, ¢/10^6 Btu	-	-	-	-	-	-	-	-	10	8

Notes:

1. Sequence repeated every ten years.
2. Ten-year cost apportioned over four years.

TABLE XI

MINE WASTE AND POWER STATION ASH DISPOSAL

Stage	Superficials				Overlying and Segregated Waste				ROM Coal Production		Dry Ash Production		Conditioned Ash				Total Waste	
	10^6 bank yd^3		10^6 loose yd^3		10^6 bank yd^3		10^6 loose yd^3		10^6 short tons		10^6 short tons		10^6 short tons		10^6 loose yd^3		10^6 loose yd^3	
	stage	cum.	stage	cum.	stage	cum.	stage	cum.	stage	cum.	stage	cum.	stage	cum.	stage	cum.	stage	cum.
1	19	19	22	22	-	-	-	-	2	2	-	-	-	-	-	-	22	22
2	25	44	29	51	11	11	15	15	20	22	6	6	7	7	6	6	50	72
3	13	57	15	66	24	35	34	49	39	61	13	19	15	22	13	19	62	134
4	46	103	53	119	38	73	53	102	71	132	22	41	26	48	22	41	128	262
5	61	164	70	189	59	132	83	185	85	217	27	68	32	80	27	68	180	442
6	73	237	84	273	97	229	135	320	82	299	27	95	32	112	27	95	246	688
7	87	324	100	373	162	391	227	547	61	360	19	114	22	134	18	113	345	1,033
8	50	374	58	431	120	511	168	715	25	385	8	122	10	144	8	121	234	1,267
9	345	719	397	828	471	982	659	1,374	390	775	126	248	149	293	124	245	1,180	2,447

Notes:

1. Bank volumes from Table II
2. Swell - superficials 15%, overlying and segregated waste 40%
3. Dry ash 32% of coal by weight
4. Ash conditioned to 15% moisture (maximum water consumption approximately 1 ft^3/sec)
5. Loose density of conditioned ash 1.2 short tons/yd^3
6. Proportions of waste to superficials assumed for Stage 9

TABLE XII

DUMPING SPACE AVAILABLE AT 100-FT INTERVALS
(10^6 yd^3)

Dump Elevation	Dump No 1		Dump No 2		Dump No 3		Dump No 4		Total	
	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.
ft										
< 2,800	17	17	-	-	-	-	-	-	17	17
2,800-2,900	7	24	17	17	-	-	-	-	24	41
2,900-3,000	22	46	47	64	-	-	-	-	69	110
3,000-3,100	14	60	72	136	-	-	-	-	86	196
3,100-3,200	45	105	91	227	-	-	-	-	136	332
3,200-3,300	15	120	114	341	-	-	8	8	137	469
3,300-3,400	29	149	135	476	-	-	29	37	193	662
3,400-3,500	28	177	151	627	-	-	52	89	231	893
3,500-3,600	30	207	165	792	11	11	100	189	306	1,199
3,600-3,700	37	244	176	968	21	32	143	332	377	1,576
3,700-3,800	43	287	187	1,155	27	59	197	529	454	2,030
3,800-3,900	49	336	200	1,355	33	92	244	773	526	2,556
3,900-4,000	57	393	216	1,571	36	128	307	1,080	616	3,172

TABLE XIII

HAT CREEK DIVERSION COSTS

Description	Unit	Quantity	Rate	Amount
			$	$
Earth Dam (140,000 yd^3)				
Stripping dam area	yd^3	8,000	1.50	12,000
Main body dam fill	yd^3	106,000	2.00	212,000
Filter material	yd^3	22,000	4.00	88,000
Impervious zone	yd^3	14,000	3.00	42,000
Engineering and contingencies				86,000
Sub-total				440,000
Pumping				
Pumps and motors, 9,000 gal/min, VTP, 600 hp, 1,180 rev/min, to include starters and pump columns	each	5	50,000	250,000
Pumphouse	ft^2	400	40.00	16,000
Butterfly valves, 18 in centre-line AA-M, includes gear operator	each	5	1,800	9,000
Check valves, 18 in Atco 618	each	5	3,000	15,000
Pressure relief valves	each	2	4,000	8,000
Pump control valves, 10 in Clayton 2061 PA	each	5	2,900	14,500
Miscellaneous fittings	lump sum	-	Allow	6,000
Electrical switchgear	lump sum	-	Allow	5,000
Powerline to pumphouse transmission, circuit breaker, etc	mile	1.3	69,000	90,000
Pump tower	lump sum	-	Allow	20,000
Miscellaneous	lump sum			135,000
Engineering and contingencies				142,000
Sub-total				710,000

/continued

TABLE XIII

(continued)

Description	Unit	Quantity	Rate	Amount
			$	$
Water Pipelines to Drainage Channel				
2 x 2,100 ft of 30 in dia. steel pipe allowing 1.5 for water hammer. Wall thickness 0.250 in	lb	337,000	0.65	219,000
Miscellaneous and additional pipe				135,000
Engineering and contingencies				89,000
Sub-total				443,000
Drainage Channel				
To by-pass main pit	yd^3	15,000	1.50	22,500
Drainage channel lining	yd^3	2,000	5.00	10,000
Rock protection rip-rap at drainage channel outlet	yd^3	150	20.00	3,000
Concrete lining at channel inlet	yd^3	20	50.00	1,000
Miscellaneous				10,000
Engineering and contingencies				11,500
Sub-total				58,000
Emergency Overflow Weir				
Concrete weir and abutment walls	yd^3	100	150.00	15,000
Excavate open channel	yd^3	2,000	2.00	4,000
Channel lining	yd^3	400	5.00	2,000

/continued

TABLE XIII

(continued)

Description	Unit	Quantity	Rate	Amount
			$	$
Rock rip-rap and channel cobbling	yd^3	50	20.00	1,000
Install 72 in dia. CPM culvert	lb	30,000	0 60	18,000
Miscellaneous				36,000
Engineering and contingencies				21,000
Sub-total				97,000
<u>Diversion of Small Stream Below Dam</u>			Allow	2,000
Total Cost for Hat Creek Diversion				1,750,000

TABLE XIV

HAT CREEK SURFACE DRAINAGE COSTS

Description	Unit	Quantity	Rate	Amount
			$	$
Excavate ditches	Lin ft	30,000	1.00	30,000
Minor drainage, works, allow	lump sum	-	-	5,000
Clear bushes, trees, grading, etc	lump sum	-	-	5,000
Engineering and contingencies				10,000
Total for Surface Drainage				50,000

TABLE XV

HAT CREEK ROAD RE-LOCATION COSTS

Description	Unit	Quantity	Rate	Amount
			$	$
Stripping of turf and topsoil and dumping	yd^3	18,000	1.25	22,500
Cut to fill	yd^3	63,000	1.30	81,900
Additional cut for rock encountered	yd^3	7,000	3.50	24,500
Cut of ditches	yd^3	500	2.20	1,100
Base course layer at, say, 12-in thickness	yd^3	18,000	4.50	81,000
Culverting layer, 18-in diameter, Armco, 12 gauge	Lin ft	750	12.00	9,000
Culvert over Hat Creek	lump sum	-	Allow	3,000
Engineering and contingencies				57,000
Total for Hat Creek Road Re-Location				280,000

TABLE XVI

SURFACE MINE BUILDINGS AND HOUSING COSTS

Description	Amount $
Surface Mine Buildings	
Administration office	286,000
Change house	236,000
Maintenance bays, workshops and warehouse with equipment and tools	3,805,000
Core shed	4,000
Powder magazines	20,000
Total Surface Mine Buildings	4,351,000
Single Work Force and Senior Staff Camp	
Eleven 40-men bunkhouses	770,000
Eleven recreational units	172,000
Kitchen-diner	56,000
First aid	5,000
Company store	20,000
Sewage treatment for both camps (half of full price)	25,000
All services (electrical, sewer, gas, water)	51,000
Power generator for both camps (half of full price)	75,000
Single work force camp	1,174,000
Senior staff camp	427,000
Total Single Work Force and Senior Staff Camp	1,601,000
Permanent Housing	
Manager's house	75,000
Senior staff houses	1,496,000
Basic houses	5,859,000
Town houses	1,404,000
Apartments	1,360,000
Total Permanent Housing	10,194,000

TABLE XVII

COST SCHEDULE OF EQUIPMENT - INFRASTRUCTURE

(10^3 \$)

Item	Unit Cost	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
		1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-00	2000-06	2006-11	2011-13
<u>Hat Creek Diversion</u>																	
Earth dam	440	440	-	-	-	-	-	440	-	-	-	-	-	-	-	-	-
Pumps	710	710	-	-	-	-	-	710	-	-	-	-	-	-	-	-	-
Pipelines	443	443	-	-	-	-	-	443	-	-	-	-	-	-	-	-	-
Drainage channel	58	58	-	-	-	-	-	58	-	-	-	-	-	-	-	-	-
Overflow weir	97	97	-	-	-	-	-	97	-	-	-	-	-	-	-	-	-
Diversion of stream	2	2	-	-	-	-	-	2	-	-	-	-	-	-	-	-	-
Sub-total		1,750	-	-	-	-	-	1,750	-	-	-	-	-	-	-	-	-
<u>Hat Creek Road Relocation</u>	280	280	-	-	-	-	-	280	-	-	-	-	-	-	-	-	-
<u>Buildings and Roads</u>																	
Administration block	286	-	286	-	-	-	-	286	-	-	-	-	-	-	-	-	-
Change house	236	-	236	-	-	-	-	236	-	-	-	-	-	-	-	-	-
Shops and warehouse	3,805	-	805	3,000	-	-	-	3,805	-	-	-	-	-	-	-	-	-
Core sheds	4	4	4	4	4	4	4	24	4	4	8	20	16	24	24	8	-
Magazines	20	-	-	20	-	-	-	20	30	-	30	-	-	-	-	-	-
Roads	860	230	240	230	30	30	30	790	70	-	70	-	-	-	-	-	-
Sub-total		234	1,571	3,254	34	34	34	5,161	104	4	108	20	16	24	24	8	-
<u>Services</u>																	
Power and water supply	615	200	315	100	-	-	-	615	-	-	-	-	-	-	-	-	-
Buses	15	30	-	30	60	30	-	150	60	30	90	90	270	225	405	130	-
Sewage disposal	50	25	25	-	-	-	-	50	-	-	-	-	-	-	-	285	-
Pick-up trucks	6	30	-	30	-	30	60	150	30	36	66	108	306	276	240	204	-
Graders	168	168	-	168	168	168	-	672	168	168	336	-	336	168	336	168	-
Sub-total		453	340	328	228	228	60	1,637	258	234	492	198	912	669	981	787	-
Total		2,717	1,911	3,582	262	262	94	8,828	362	238	600	218	928	693	1,005	795	-
Allowance for initial spares		35	55	70	70	70	70	370	-	-	-	-	-	-	-	-	-
Total		2,752	1,966	3,652	332	332	164	9,198	362	238	600	218	928	693	1,005	795	-
Allowance for depreciation		293	293	293	293	310	310	1,792	310	310	620	930	1,860	1,860	1,860	1,550	620
Average cost, \$/short ton		-	-	-	-	-	-	-	0.07	0.04	0.05	0.02	0.02	0.02	0.02	0.02	0.02
Average cost, ¢/Btu		-	-	-	-	-	-	-	negligible								
<u>Employee Housing</u>																	
Trailer camps	1,600	-	-	-	-	-	-	400	200	200	400	800	-	-	-	-	-
Permanent	10,200	-	-	-	-	-	-	-	-	-	-	2,000	4,100	4,100	-	5,100	5,100
<u>Land Development Cost</u>		-	-	-	-	-	-	-	-	-	-	-	-	2,000	-	-	2,000

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TABLE XVIII

LABOUR SCHEDULE AND PAYROLL COSTS
(10^3 $)

Category	Hourly Rate	Rate with Fringe Benefit	Annual Rate	Stage 1						Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
				1978-79*	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13
	$	$	$															
Mobile Mining Equipment																		
Equipment operators	6.90	8.30	17,300	398	398	398	398	398	1,990	1,298	1,367	2,665	9,890	20,890	22,420	26,480	62,780	19,690
Maintenance personnel	7.20	8.70	18,100	344	344	344	344	344	1,720	1,086	1,140	2,226	8,320	17,490	18,840	22,190	30,370	16,450
Labourers	6.45	7.80	16,200	49	49	49	49	49	245	130	130	260	936	1,991	2,136	2,548	3,432	1,840
Overtime	-	-	-	79	79	79	79	79	395	126	132	258	953	2,019	2,170	2,562	3,506	1,900
Sub-total	-	-	-	870	870	870	870	870	4,350	2,640	2,769	5,409	20,099	42,390	45,566	53,780	100,088	39,880
Fixed Installations																		
Equipment operators	6.90	8.30	17,300	104	104	104	104	104	520	225	225	450	4,152	8,304	8,304	8,304	6,920	2,768
Maintenance personnel	7.20	8.70	18,100	36	36	36	36	36	180	72	72	144	1,737	1,374	1,374	1,374	2,895	1,158
Labourers	6.45	7.80	16,200	65	65	65	65	65	325	65	65	130	486	972	972	1,622	1,460	584
Overtime	-	-	-	21	21	21	21	21	105	18	18	36	318	636	636	671	565	226
Sub-total	-	-	-	226	226	226	226	226	1,130	380	380	760	6,693	11,286	11,286	11,971	11,840	4,736
Infrastructure																		
Equipment operators	6.90	8.30	17,300	69	69	69	69	69	345	69	69	138	417	834	834	834	695	278
Maintenance personnel	7.20	8.70	18,100	36	36	36	36	36	180	36	36	72	216	432	432	432	360	144
Labourers	6.45	7.80	16,200	162	162	162	162	162	810	162	162	324	972	1,944	1,944	1,944	1,620	648
Overtime	-	-	-	13	13	13	13	13	65	13	13	26	81	162	162	162	135	54
Sub-total	-	-	-	280	280	280	280	280	1,400	280	280	560	1,686	3,372	3,372	3,372	2,810	1,124
Total cost/year or stage	-	-	-	1,376	1,376	1,376	1,376	1,376	6,880	3,300	3,429	6,729	28,478	57,048	60,224	69,123	114,738	45,740
Production 10^6 tons	-	-	-	-	-	-	-	-	-	4.3	7.6	12	39	79	78	79	66	26
Average cost, $/short ton	-	-	-	-	-	-	-	-	-	0.77	0.45	0.56	0.73	0.72	0.77	0.87	1.74	1.75
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	-	-		7	4	5	7	7	7	8	16	16
Average annual employees				73	73	73	73	73	73	179	187	184	516	537	562	646	959	1,243

* including part 1977-78

TABLE XIX

MANAGERIAL, TECHNICAL AND ADMINISTRATIVE STAFF - NUMBERS AND SALARY COSTS

Category	Basic Annual Salary	Annual Payroll Cost including Fringe Benefits	1978-79		1979-80		1980-81		1981-82		1982-83		1983-84*		1984-85		1986-2000**		2001-2012	
			Number	Annual Cost	Number	Annual Cost	Number	Annual Cost	Number	Annual Cost	Number	Annual Cost	Number	Annual Cost	Number	Annual Cost	Number	Annual Cost	Number	Annual Cost
	$	$		$		$		$		$		$		$		$		$		$
Mine Manager	29,520	36,900	1	36,900	1	36,900	1	36,900	1	36,900	1	36,900	1	36,900	1	36,900	1	36,900	1	36,900
Secretary	10,640	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300
Mining Engineer	24,000	30,000	1	30,000	1	30,000	1	30,000	2	60,000	2	60,000	2	60,000	2	60,000	2	60,000	2	60,000
Shop Superintendent	24,000	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000
Electrical Foreman	21,600	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000
Mechanical Foreman	21,600	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000
Pit Foreman	24,000	30,000	2	60,000	2	60,000	2	60,000	3	90,000	3	90,000	3	90,000	4	120,000	6	180,000	8	240,000
Stockpile Foreman	24,000	30,000	-	-	-	-	1	30,000	1	30,000	2	60,000	3	90,000	4	120,000	4	120,000	5	150,000
Dumps Foreman	24,000	30,000	-	-	-	-	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000	1	30,000
Conveyor Foreman	24,000	30,000	-	-	-	-	1	30,000	1	30,000	2	60,000	3	90,000	4	120,000	4	120,000	4	120,000
Shift Bosses	20,400	25,500	2	51,000	2	51,000	2	51,000	2	51,000	2	51,000	6	153,000	8	204,000	10	255,000	20	510,000
Geologist	21,600	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000	1	27,000
Geological Assistant	10,640	13,300	1	13,300	1	13,300	1	13,300	1	13,300	2	26,600	2	26,600	2	26,600	2	26,600	2	26,600
Engineering Geologist	17,600	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000
Geotechnical Assistant	10,640	13,300	1	13,300	1	13,300	1	13,300	1	13,300	2	26,600	2	26,600	2	26,600	2	26,600	2	26,600
Warehouseman	10,640	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300
Storekeeper	8,240	10,300	2	20,600	2	20,600	2	20,600	2	20,600	3	30,900	3	30,900	4	41,200	4	41,200	4	41,200
Draftsman	14,400	18,000	1	18,000	1	18,000	2	36,000	2	36,000	2	36,000	2	36,000	2	36,000	2	36,000	2	36,000
Clerk	12,000	15,000	1	15,000	1	15,000	2	30,000	2	30,000	3	45,000	3	45,000	4	60,000	5	75,000	6	90,000
Surveyor	15,600	19,500	1	19,500	1	19,500	1	19,500	2	39,000	2	39,000	2	39,000	4	78,000	5	97,500	6	117,000
Rodman	12,000	15,000	1	15,000	1	15,000	1	15,000	2	30,000	2	30,000	2	30,000	4	60,000	5	75,000	6	90,000
Office Manager	20,000	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000
Purchasing Agent	20,000	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000
Accountant	15,360	19,200	1	19,200	1	19,200	1	19,200	1	19,200	1	19,200	1	19,200	2	38,400	3	57,600	4	76,800
Gateman and Dryman	8,240	10,300	2	20,600	2	20,600	3	30,900	3	30,900	3	30,900	4	41,200	4	41,200	4	41,200	4	41,200
Stenographer/Telephone Operator	8,240	10,300	1	10,300	1	10,300	2	20,600	3	30,900	3	30,900	4	41,200	4	41,200	4	41,200	4	41,200
Industrial Relations	20,000	25,000	-	-	-	-	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000	1	25,000
Sampler	10,640	13,300	-	-	-	-	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300	1	13,300
Assistant Sampler	8,240	10,300	-	-	-	-	1	10,300	1	10,300	2	20,600	2	20,600	2	20,600	2	20,600	2	20,600
Environmental Engineer	17,600	22,000	-	-	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000	1	22,000
Environmental Assistant	10,640	13,300	-	-	1	13,300	1	13,300	1	13,300	2	26,600	2	26,600	2	26,600	2	26,600	2	26,600
Safety Engineer	15,360	19,200	-	-	-	-	1	19,200	1	19,200	1	19,200	1	19,200	1	19,200	1	19,200	1	19,200
Total			27	552,300	29	587,600	40	799,000	45	903,800	53	1,039,300	61	1,221,900	73	1,476,400	81	1,656,100	98	2,069,800
Average cost, \$/short ton				-		-		-		-		-		0.28		0.19		0.13		0.16
Average cost, ¢/10^6 Btu				-		-		-		-		-		3		2		1		1

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* start up

** full production

Table XX

MATERIAL AND FUEL COST SUMMARY (1975 PRICES)
(excluding electric power)
(10^3 \$)

Item	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-00	2000-06	2006-11	2011-13
Mobile mining equipment	-	-	1,851	1,869	1,919	1,919	7,558	3,452	4,013	7,465	19,481	43,585	50,143	65,021	93,702	55,471
Explosives (by contract)	-	-	30	30	30	30	120	496	491	987	6,258	12,732	13,392	16,536	21,495	12,100
Exploratory drilling (by contract)	450	450	450	450	450	450	2,700	450	450	900	1,350	2,700	2,700	2,700	2,250	900
Sub-total	450	450	2,331	2,349	2,399	2,399	10,378	4,398	4,954	9,352	27,089	59,017	66,235	84,257	117,447	68,471
Fixed installations	150	150	301	304	317	317	1,539	660	777	1,437	7,577	16,090	17,751	23,387	32,949	19,340
Infrastructure	147	147	147	147	147	147	882	147	263	410	789	1,578	2,378	2,068	1,935	774
Engineering and administration	-	20	20	20	20	20	100	33	34	67	120	253	270	317	413	214
Total cost/year or stage	747	767	2,799	2,820	2,883	2,883	12,819	5,238	6,028	11,266	35,575	76,938	86,634	110,029	152,744	88,799
Production, 10^6 tons										12	39	79	78	79	66	26
Average cost, \$/short ton										0.94	0.91	0.97	1.11	1.39	2.31	3.41
Average cost, ¢/10^6 Btu										9	8	9	10	12	21	31

TABLE XXI

DIRECT OPERATING COST SUMMARY
(10^3 \$)

Item	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-00	2000-06	2006-11	2011-13
Mobile Mining Equipment																
Labour	-	870	870	870	870	870	4,350	2,640	2,769	5,409	20,019	42,390	45,566	53,780	73,608	39,880
Materials, fuel and miscellaneous	450	450	2,331	2,349	2,399	2,399	10,378	4,398	4,954	9,352	27,089	59,017	66,235	84,257	117,447	68,471
Electric power	-	-	57	58	58	58	231	127	128	255	625	1,320	1,590	1,940	2,270	1,230
Sub-total (cumulative)	450 -	1,320 -	3,258 -	3,277 -	3,327 -	3,327 -	14,959 -	7,165 -	7,851 -	15,016 29,975	47,733 77,708	102,727 180,435	113,391 293,826	139,977 433,803	193,325 627,128	109,581 736,709
Fixed Installations																
Labour	-	226	226	226	226	226	1,130	380	380	760	6,693	13,386	13,386	14,208	11,840	4,736
Materials and fuel	150	150	301	304	317	317	1,539	660	777	1,437	7,577	16,090	17,751	23,387	32,949	19,340
Electric power	-	-	4	4	59	144	211	328	510	838	2,436	4,872	4,872	4,872	4,060	1,624
Sub-total (cumulative)	150 -	376 -	531 -	534 -	602 -	687 -	2,880 -	1,368 -	1,667 -	3,035 5,915	16,706 22,621	34,348 56,969	36,009 92,978	42,467 135,445	48,849 184,294	25,700 209,994
Infrastructure																
Labour	-	280	280	280	280	280	1,400	280	280	560	1,686	3,372	3,372	3,372	2,810	1,124
Materials and fuel	147	147	147	147	147	147	882	147	263	410	789	1,578	2,378	2,068	1,935	774
Electric power	-	-	95	95	95	95	380	95	95	190	285	570	570	570	475	190
Sub-total (cumulative)	147 -	427 -	522 -	522 -	522 -	522 -	2,662 -	522 -	638 -	1,160 3,822	2,760 6,582	5,520 12,102	6,320 18,422	6,010 24,432	5,220 29,652	2,088 31,740
Engineering and Administration																
Salaries	-	552	588	799	904	1,039	3,882	1,221	1,476	2,697	4,968	9,936	9,936	12,006	10,350	4,140
Materials	-	20	20	20	20	20	100	33	34	67	120	253	270	317	413	214
Electric power - included in infrastructure	-	-	-	-	-	-	-									
Sub-total (cumulative)	- -	572 -	608 -	819 -	924 -	1,059 -	3,982 -	1,254 -	1,510 -	2,764 6,746	5,088 11,834	10,189 22,023	10,206 32,229	12,323 44,552	10,763 55,315	4,354 59,669
Consultants' fees (cumulative)	- -	500 -	500 -	500 -	500 -	500 -	2,500 -	250 -	250 -	500 3,000	750 3,750	1,500 5,250	1,500 6,750	1,500 8,250	1,250 9,500	500 10,000
Total cost/year or stage (cumulative)	747 -	3,195 -	5,419 -	5,652 -	5,875 -	6,095 -	26,983 -	10,559 -	11,916 -	22,475 49,458	73,037 122,495	154,284 276,779	167,426 444,205	202,277 646,482	259,407 905,889	142,223 1,048,112
Production, 10^6 tons								4.3	7.6	12	39	79	78	79	66	26
Average cost, \$/short ton								2.46	1.57	1.87	1.87	1.95	2.15	2.56	3.93	5.47
Average cost, ¢/10^6 Btu								22	14	17	17	18	20	23	36	49
Total volume of material removed, 10^6 byd^3	-	-	6	6	6	6	24	13	15	28	72	164	169	219	301	174
Average cost, \$/byd^3								0.81	0.80	0.80	1.02	0.94	0.99	0.92	0.86	0.82

TABLE XXII

DEPRECIATION SUMMARY
(10^3 \$)

Item	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13
Mobile equipment	-	-	2,460	2,460	2,460	2,460	9,840	4,540	4,930	9,470	20,400	44,690	52,750	68,940	100,950	60,280
Fixed installations	162	483	1,464	1,766	2,492	4,903	11,270	5,386	6,755	12,141	20,265	40,530	40,530	40,530	33,775	13,510
Infrastructure	293	293	293	293	310	310	1,792	310	310	620	930	1,860	1,860	1,860	1,550	620
Other capitalised costs	50	50	100	100	400	400	1,100	600	800	1,400	3,000	6,000	6,000	6,000	5,000	2,000
Total cost/year or stage	505	826	4,317	4,619	5,662	8,073	24,002	10,836	12,795	23,631	44,595	93,080	101,140	117,330	141,275	76,410
Average cost, \$/short ton	-	-	-	-	-	-	-	2.52	1.68	1.97	1.14	1.18	1.30	1.49	2.14	2.94
Average cost, 10^6 Btu	-	-	-	-	-	-	-	23	15	18	10	11	12	14	19	27

TABLE XXIII

CAPITAL INVESTMENT, INTEREST DURING CONSTRUCTION, INTEREST AND INSURANCE (1975 PRICES)

(10^3 \$)

Item	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13
Mobile mining equipment	-	-	2,884	2,884	2,884	2,884	11,536	6,503	6,503	13,006	25,392	44,556	53,274	69,048	112,625	74,238
Fixed installations	1,624	3,214	9,813	3,023	6,260	24,113	48,047	4,831	13,686	18,517	-	49,923	16,641	66,564	-	-
Infrastructure	2,752	1,966	3,652	332	332	164	9,198	362	238	600	218	928	693	1,005	795	-
Sub-total	4,376	5,180	16,349	6,239	9,476	27,161	68,781	11,696	20,427	32,123	25,610	95,407	70,608	136,617	113,420	74,238
Direct operating costs to start-up	747	3,195	5,419	5,652	5,875	6,095	26,983	-	-	-	-	-	-	-	-	-
Insurance costs to start-up	88	104	327	125	188	543	1,375	-	-	-	-	-	-	-	-	-
Working capital	729	729	729	729	729	729	4,374	729	729	1,458	-	-	-	-	-	-
Total capital costs	5,940	9,208	22,824	12,745	16,268	34,528	101,513	12,425	21,156	33,581	25,610	95,407	70,608	136,617	113,420	74,238
Corporate overhead	846	846	846	846	846	846	5,076	-	-	-	-	-	-	-	-	-
Total capital costs including corporate overhead	6,786	10,054	23,670	13,591	17,114	35,374	106,589	12,425	21,156	33,581	25,610	95,407	70,608	136,617	113,420	74,238
Cumulative capital costs including corporate overhead	6,786	16,840	40,510	54,101	71,215	106,589	106,589	119,014	140,170	140,170	165,780	261,187	331,795	468,412	581,832	656,070
Interest on cumulative capital costs up to beginning of year (10%)	-	713	1,840	4,509	6,387	8,823	22,272	-	-	-	-	-	-	-	-	-
Interest on capital cost during year (5%)	339	503	1,184	680	856	1,769	5,331	-	-	-	-	-	-	-	-	-
Total interest during construction	339	1,216	3,024	5,189	7,243	10,592	27,603	-	-	-	-	-	-	-	-	-
Total investment	7,125	11,270	26,694	18,780	24,357	45,966	134,192	12,425	21,156	33,581	25,610	95,407	70,608	136,617	113,420	74,238
Cumulative investment	7,125	18,395	45,089	63,869	88,226	134,192	134,192	146,617	167,773	167,773	193,383	288,790	359,398	496,015	609,435	683,673
Depreciation	505	826	4,317	4,619	5,662	8,073	24,002	10,836	12,795	23,631	44,595	93,080	101,140	117,330	141,275	76,410
Cumulative depreciation	505	1,331	5,648	10,267	15,929	24,002	24,002	34,838	47,633	47,633	92,228	185,308	286,448	403,778	545,053	621,463
Outstanding investment at year or stage end	6,620	17,064	39,441	53,602	72,297	110,190	110,190	111,779	120,140	120,140	101,155	103,482	72,950	92,237	64,382	62,210
Average outstanding investment	3,310	11,842	28,253	46,522	62,950	91,244	244,121	110,985	115,960	226,945	332,910	541,850	574,360	502,360	408,850	130,080
Interest on outstanding average investment (10%)	331	1,184	2,825	4,652	6,295	9,124	24,411	11,099	11,596	22,695	33,291	54,185	57,436	50,236	40,885	13,008
Insurance (2%)	66	237	565	930	1,259	1,825	4,882	2,220	2,319	4,539	6,659	10,838	11,487	10,047	8,177	2,602

TABLE XXIV

ROM COAL PRODUCTION COST
(10^3 \$)

Item	Stage 1	Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1977-83	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13
Coal production, 10^6 ROM tons	-	4.3	7.6	12	39	79	78	79	66	26
Direct operating cost	-	10,559	11,916	22,475	73,037	154,284	167,426	202,277	259,400	142,223
Depreciation	-	10,836	12,795	23,631	44,595	93,080	101,140	117,330	141,275	76,410
Interest on average investment	-	11,099	11,596	22,695	33,291	54,185	57,436	50,236	40,885	13,008
Insurance	-	2,220	2,319	4,539	6,659	10,838	11,487	10,047	8,177	2,602
Royalty (75¢/long ton, 67¢/short ton)	-	2,880	5,090	7,970	26,130	52,930	52,260	52,930	44,220	17,420
Total cost/year or stage	-	37,594	43,716	81,310	183,712	365,317	389,749	432,820	493,957	251,663
Average cost, \$/short ton	-	8.74	5.75	6.78	4.71	4.62	5.00	5.48	7.48	9.68
Average cost, ¢/10^6 Btu	-	80	52	62	43	42	45	50	68	88

TABLE XXV

CASH FLOW (EXPENSES) AND
UNIFORM SELLING PRICE (1975 PRICES)
(10^3 $)

Item	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8	Present Value
	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13	
Capital investment	7,125	11,270	26,694	18,780	24,357	45,966	134,192	12,425	21,156	33,581	25,610	95,216	70,608	136,617	113,420	74,238	-
Direct operating costs	-	-	-	-	-	-	-	10,559	11,916	22,475	73,037	154,284	167,426	202,277	259,407	142,223	-
Insurance	-	-	-	-	-	-	-	2,220	2,319	4,539	6,659	10,838	11,487	10,047	8,177	2,602	-
Royalty (75¢/long ton, 67¢/short ton)	-	-	-	-	-	-	-	2,880	5,090	7,970	26,130	52,930	52,260	52,930	44,220	17,420	-
Cash flow expenses	7,125	11,270	26,694	18,780	24,357	45,966	134,192	28,084	40,481	68,565	131,436	313,268	301,781	401,871	425,224	236,483	-
Discounted cash flow at 15%	6,196	8,521	17,551	10,737	12,104	19,872	74,986	16,542	13,233	23,775	32,675	40,962	18,019	10,110	4,897	1,653	207,077
Discounted cash flow at 10%	6,477	9,314	20,056	12,827	15,126	25,925	89,725	14,407	18,905	33,312	50,797	77,772	43,943	32,426	20,503	8,013	356,491
Coal production, 10^6 short tons	-	-	0.1	0.2	0.4	0.4	1.1	4.3	7.6	12	39	79	78	79	66	26	-
Discounted coal production at 15%, 10^6 short tons	-	-	0.07	0.12	0.2	0.17	0.56	1.62	2.48	4.1	9.7	10.72	4.56	2.01	0.77	0.18	32.6
Discounted coal production at 10%, 10^6 short tons	-	-	0.08	0.14	0.25	0.23	0.7	2.21	3.55	5.76	15.08	20.11	11.23	6.43	3.16	0.88	63.35

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Uniform Selling Price

	$/short ton	¢/10^6 Btu
15% discount rate	6.35	58
10% discount rate	5.63	51

TABLE XXVI

COAL PRODUCTION COST (INFLATED)
(10^3 \$)

Item	Stage 1	Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8
	1977-83	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13
Coal production, 10^6 ROM tons	-	4.3	7.6	12	39	79	78	79	66	26
Inflation factor (1975-76 = 1)	-	1.78	1.87	1.83	2.06	2.58	3.45	4.68	6.10	7.17
Direct operating costs	-	10,559	11,916	22,475	73,037	154,284	167,426	202,277	259,407	142,223
Depreciation	-	10,836	12,795	23,631	44,595	93,080	101,140	117,330	141,275	76,410
Interest on average investment	-	11,099	11,596	22,695	33,291	54,185	57,436	50,236	40,885	13,008
Insurance	-	2,220	2,319	4,539	6,659	10,838	11,487	10,047	8,177	2,602
Sub-total - uninflated	-	34,714	38,626	73,340	157,582	312,387	337,489	379,890	449,744	234,243
inflated	-	61,791	72,231	134,022	324,652	805,435	1,163,705	1,778,339	2,744,345	1,679,404
Royalty - uninflated (75¢/long ton, 67¢/short ton)	-	2,880	5,090	7,970	26,130	52,930	52,260	52,930	44,220	17,420
Total cost/year or stage	-	64,671	77,321	141,992	350,782	858,365	1,215,965	1,831,269	2,788,565	1,696,824
Average cost, \$/short ton	-	15.04	10.17	11.83	8.99	10.87	15.59	23.18	42.25	65.26
Average cost, ¢/10^6 Btu	-	137	92	107	82	99	142	211	384	593

TABLE XXVII

CASH FLOW (EXPENSES) AND
UNIFORM SELLING PRICE (INFLATED)
(10^3 \$)

Item	Stage 1							Stage 2			Stage 3	Stage 4	Stage 5	Stage 6	Stage 7	Stage 8	Present Value
	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	Total	1983-84	1984-85	Total	1985-88	1988-94	1994-2000	2000-06	2006-11	2011-13	
Capital investment	7,125	11,270	26,694	18.780	24,357	45,966	134,192	12,425	21,156	33,581	25,610	95,216	70,608	136,617	113,420	74,238	-
Inflation factor	1.21	1.33	1.40	1.54	1.61	1.70	1.56	1.78	1.87	1.83	2.06	2.61	3.43	4.63	6.10	7.16	-
Direct operating costs	Capitalised							10,559	11,916	22,475	73,037	154,284	167,426	202,277	259,407	142,223	-
Insurance	-	-	-	-	-	-	-	2,220	2,319	4,539	6,659	10,838	11,487	10,047	8,177	2,602	-
Sub-total - uninflated	7,125	11,270	26,694	18,780	24,357	45,966	134,192	25,204	35,391	60,595	105,306	260,529	249,521	348,941	381,004	219,063	
- inflated	8,621	14,989	38,973	28,921	39,215	77,983	208,702	44,863	66,181	111,044	216,996	680,992	854,784	1,616,473	2,323,708	1,569,557	-
Royalty - inflated (75¢/long ton, 67¢/short ton)	-	-	-	-	-	-	-	2,880	5,090	7,970	26,130	52,930	52,260	52,930	44,220	17,420	-
Cash flow expenses - inflated	8,621	14,989	38,973	28,921	39,215	77,983	208,702	47,743	71,271	119,014	243,126	733,922	907,044	1,669,403	2,367.928	1,586,977	-
Discounted cash flow at 15%	7,497	11,332	25,625	16,534	19,494	33,582	114,064	17,947	23,298	41,245	60,197	93,716	53,243	41,255	26,226	10,706	440.652
Coal production, 10^6 short tons	-	-	0.1	0.2	0.4	0.4	1.1	4.3	7.6	12	39	79	78	79	66	26	-
Discounted coal production at 15%, 10^6 short tons	-	-	0.07	0.12	0.2	0.17	0.56	1.62	2.48	4.1	9.7	10.72	4.56	2.01	0.77	0.18	32.6

Uniform Selling Price

\$/short ton	¢ 10^6 Btu
13.52	123

TABLE XXVIII

BUDGETS

(10^3 \$)

Item	1975-76	1976-77	1977-78	1978-79	1979-80	1980-81	1981-82	1982-83	1983-84	1984-85
		Selection of mining area								
Geological exploration of preferred pit locations	(200) 200	(600) 550								
Geotechnical exploration of preferred pit locations	(25) 25	(25) 25								
Hydrological study	50	550	400							
Geological exploration to complete the selected mine design		(1,000) 1,040	(1,000) 1,000							
Geotechnical exploration to complete the selected mine design		(400) 834	(500) 800							
Mining engineering to complete the selected mine design		(600) 600	(600) 600							
Mining equipment and services (including fixed installations)			2,600	(300) 5,200	(6,500) 6,000	(8,300) 6,000	(7,800) 6,200	(7,400) 7,000		
Coal and ash handling					5,000	6,000	11,000	12,000		
Continuing exploration and geotechnical				(300) 300	(300) 300	(200) 300	(100) 300	(100) 300		
Creek diversion			1,800		(1,000)	(1,000)				
Incline box-cut and preproduction overburden removal (cash only)			1,600	4,000	(6,300) 6,500	(8,200) 6,500	(7,000) 6,800	(7,000) 7,400		

Notes:

1. 20th October budget shown in brackets for comparison
2. All figures in 1975 prices