PD·NCB Consultants Limited

In association with

Wright Engineers Ltd & Golder Associates

Report No.9

Revised Report on Hat Creek Openpit No.1

(Volume I)

to

British Columbia Hydro and Power Authority

London

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(contained in Volume II)

CHAPTER I

INTRODUCTION

1. Appendix "A" lists Reports No 1 to 10 in this series prepared by PD-NCB Consultants in association with Wright Engineers and Golder Associates. This report is an up-date of Report No 2 incorporating the new data available and bringing the economics up to date.

TERMS OF REFERENCE

2. The precise terms of reference confirmed in BC Hydro and Power Authority's (BCH) letter of 25th May, 1976, are as follows:-

"I UP-DATING OF INITIAL MINING REPORTS ON OPENPITS NO 1 AND 2

The implications of the further data that become available up to the end of October will be assessed and an amendment report will be prepared indicating the technical and economic changes that have become evident since the initial mining reports.

This report would be prepared toward the end of 1976 but would be preceded by notification of the major changes, if any, to the client prior to 8th October Board Meeting of the client.

. . . . . .

The overall up-dating and summary report (I above) will be presented to the client by the end of January 1977."

OPENPIT NO 2

3. Whilst reference is made to Openpits No 1 and 2, following a review of Report No 3 which was submitted to BCH in June 1976, a decision was taken by BCH that Openpit No 1 was the more favourable and therefore the geological in-fill drilling programme and the geotechnical field work in the 1976 field season were carried out in that area. Hence no new data were available specifically relating to Openpit No 2. As a consequence of this, Openpit No 2 is not dealt with in this report except for comparative purposes (see Chapters IX and X).

DATA AVAILABLE

4. The terms of reference specified that data received up to October, 1976, would be incorporated, but in the event very little specific data were available at that time although, of course, a large amount of general information had been collected from the field work. Most of the laboratory work, for instance, was still in progress. Whilst the situation was a little better at the beginning of January, 1977, a lot of material was still awaited and much of that received was in draft form. However, it was decided to proceed with drafting this report to avoid further delay. The data received from 11th June, 1976, to 5th January, 1977, are listed in Appendix "B". The reasons for the delay can be stated as:-

(i) greater complexity,

(ii) greater volume of data,

(iii) processing and finalising.

As a result of these delays the report could not be presented by the end of January, 1977.

MAJOR CHANGES

5. The major implications of the field work which were notified to BCH prior to 8th October, 1976, can be summarised as follows:-

(i) overall pit slope of 16^0 confirmed,

(ii) more coal discovered,

(iii) coal deposit can be divided into four zones,

(iv) clay materials weaker than expected,

(v) presence of bentonitic beds within some of the coal,

(vi) quality problem,

etc.

None of these materially affect the concepts adopted in Report No 2 and, in fact, confirm them. Modifications and adjustments have, of course, been made in accordance with these findings.

PRIOR REPORTS

6. This report includes the findings of all the prior reports listed in Appendix "A", Reports No 4 and 6 - Geotechnical, Report No 5 - Computerisation and Report No 8 - Reclamation. Report No 7 deals with the combination of the outputs of the two conceptual openpits described in Reports No 2 and 3 for a 5,000-MW power project. This report deals only with Openpit No 1 for a 2,000-MW (nominal) project. If Report No 3 (Openpit No 2) were to be up-dated in a similar way then Report No 7 could also be up-dated.

7. The geotechnical studies are fundamental to the design of the openpit because of the over-riding necessity to design a stable pit (or at least one where slope failure is limited to an acceptable amount).

8. The computerisation study is also fundamental as it provides the link between all aspects of the mining project and a means of handling and processing the ever-increasing mass of numerical data. The computer system will have the following features:-

(i) drill-hole data base,

(ii) inventory of all minerals in the deposit,

(iii) quality prediction and control,

(iv) production scheduling,

(v) optimisation of pit design.

9. It can, of course, be extended to include such features as:-

(i) cost analysis,

(ii) maintenance scheduling,

(iii) blending, stockpiling and coal preparation,

etc.

It would also interface with a similar system for the power plant.

10. The reclamation study deals with the openpit itself, the waste dumps, water resources, control of pollutants, etc, both during operation and after operations cease, the object being to create as little disturbance as possible and to leave the site in an environmentally acceptable condition. The costs specific to these activities are included in the economic calculations (Chapter IX).

BASIC DATA

11. Table I lists the basic data used in this report. All these items are, of course, subject to verification in the light of better data.

ACKNOWLEDGEMENTS

12. The continued interest and encouragement of BCH at all levels is acknowledged with thanks, as is also the co-operation of consultants working on other aspects of the power project, particularly Dolmage Campbell, Integ/Ebasco, Monenco and Acres.

CHAPTER II

GEOLOGICAL AND GEOTECHNICAL ASSESSMENT

GENERAL

1. This chapter is concerned with identifying those areas where the current knowledge of the deposit appreciably differs from that given in Report No 2, and where items of importance have been confirmed by the findings of the 1976 prospecting and in-fill drilling programme. The relationship between these findings and the geotechnical constraints as determined by the Golder Associates Limited (GA) study is also discussed.

2. The geological setting of the deposit has been given in Chapter III of Report No 6 (GA) in this series and in a paper by Campbell, Jory and Saunders ("Hat Creek Coal Deposits", D.D. Campbell, L.T. Jory, and C.R. Saunders, Coal Division CIM District 6 Meeting 1976). These reports are based on drilling over several field seasons (1957/59 and 1974/76) and the general characteristics of the deposit are much more apparent than at the date of Report No 1. The geological basis of the up-date report is a series of preliminary draft plans and cross sections prepared by Dolmage Campbell and Associates Limited (DCA) in October, 1976. Whilst it is recognised that alternative structural interpretations are possible and that processing of drilling data is still incomplete, it is considered that a re-drafting of sections and plans as undertaken for Report No 2 is inappropriate.

CHARACTERISTICS OF NO 1 DEPOSIT

3. The No 1 Deposit at the northern end of the Hat Creek valley has many of the characteristics of intermontane coal and lignite deposits elsewhere[(1)], ie:-

(i) A coal deposit with substantial variations in seam thickness, being thickest towards the apparent centre of the basin and somewhat thinner at the margins.

(ii) Limited lateral continuity of marker horizons with marked facies changes within intercalated beds and from coal to non-carbonaceous material.

(iii) Variations in displacement along some faults suggesting contemporaneous development of faulting at the time of basin development and coal formation.

STRUCTURE OF NO 1 DEPOSIT

4. The structure is basically that outlined in Report No 2 (Openpit No 1), with a synclinal area to the west of Hat Creek and an anticlinal ridge along or just east of the creek. Drilling in 1976 has, however, shown that a second synclinal trough lies further to the north-east, bounded on the east by a possible N-S fault.

[(1)] Westfield Fife, Scotland. E.H. Francis, "The Economic Geology of the Fife Coalfields, Area II". Memoirs of the Geological Survey, Scotland, 1961.
Elbistan, Turkey. O. Gold, and G. Luttig, "Result of Turkey's Research on Brown Coal" Braunkohle, August 1972, pp 253 to 268.

Fault positions are similar to those shown in Report No 2. Faults have been given numbers rather than names and positions are more tightly pinned. The designations are as follows:-

Former Name	Present Number	Notes
Fault S	1	May be a "roll" or sedimentary break rather than a continuous fault.
Dry Lake Fault	Not present?	Outcrop and burn zone area.
Fault H	5 and 6	Possible fault complex.
Mag Fault	7	SW boundary of NE syncline.
Trig Fault	4	
Finney Fault		1976 drilling not adequate to confirm presence.
-	8	Inferred boundary fault at eastern edge of NE syncline.

5. The structure therefore comprises two synclines, plunging to the south and separated by an anticlinal area possibly with horst-like faulting on the flanks. The area is apparently terminated by sub-surficial outcrops to the west and north and by faults to the south and east. These structural elements are apparent from the structure contours based on coal zone boundaries and from the N-S and E-W cross sections. Owing to problems of correlation, interpretation along the anticlinal ridge incorporating faults 5 and 6 is tentative. It is probable that most of the faulting within the deposit is normal faulting: reverse faulting with repetition of beds is unlikely within the main coal area. The positions of some of the boundary faults are still uncertain, especially to the south where few boreholes have been drilled and where the position of marker horizons in the coal sequence is uncertain due to facies change and the increasing thickness of overburden. Further drilling should, in part, be directed to the delineation of these faults where they lie within or near to the confines of the proposed 600-ft pit (ie, floor at 2,400-ft elevation): to establish the presence of smaller faults with displacement of tens rather than hundreds of feet as well as complex, larger faults, a more refined lithological correlation system would be required with borehole spacings of 200 ft to 500 ft.

6. The inclination of the strata appears to be much as outlined previously. The plunge of the main western syncline ranges from 10^0 to 35^0 whilst that of the smaller north-eastern syncline is probably less than 10^0. The dip of the strata on the flanks of the main syncline ranges between 15^0 and more than 60^0, especially on the eastern side against the anticlinal ridge. The north-eastern syncline is probably more gently inclined with dips of 25^0 or less. Local variations are most probable as changes in dip along vertical drill core are apparent and cannot always be explained by local folding; slumping and/or small-scale faulting with strata disturbance is present (and was noted in underground workings[(2)]). It would be usual in this deposit to anticipate a general decrease in dip in higher beds on the basin flanks. There may be some evidence of this in the western flank of the main syncline.

[(2)] BC Department of Mines Report, 1924, pp 305 to 315.

7. The inferred positions of faults, the inclination of the several coal zones and the displacement along faults (or the direction of movement) are shown on Plates 1 to 15R. Plate 16 is a pictorial representation of the coal deposits within the 600-ft pit.

IMPLICATIONS OF THE STRUCTURE

8. The structure has the following implications so far as mining is concerned:-

(i) The proposed access ramp coincides with the alignment of the anticlinal area as defined by faults 5, 6 and 7. It may be assumed that the fault zone continues NNW outside the coal area towards the shallower part of the ramp, possibly giving rise to the conditions encountered in some of the geotechnical boreholes in this area.

(ii) Coal is inclined with the slope of the proposed pit on the west side of the mine. The dip is generally between 20^0 and 40^0, although in upper zones it may be 10^0 to 15^0. It is probable, therefore, that the western slope may approximate to a footwall situation and its stability must eventually be analysed accordingly.

(iii) To the south and east present indications are that strata will dip along or into rather than out of the pit slopes. Owing to local faulting and folding, there will be some exceptions to this but, in general, these slopes are more analogous to the highwall or hanging wall situation.

(iv) The northern and north-eastern slopes of the pit will have sections with strata inclined towards the excavation and along the faces.

(v) As a very approximate guide with the proposed pit layout:-

- 45% of the coal dips in the same direction as the working pit slopes

- 45% of the coal dips along the pit slopes, ie the strike of the beds is normal to the pit slope

- 10% of the coal dips in the opposite direction to the pit slopes.

(vi) Faults can be expected to intersect pit slopes at different angles; however, with the exception of faults 1 and 4, faults should not be sub-parallel with the faces.

(vii) Very little is known regarding the geological structure outside the coal area. Within a fault block it is reasonable to assume a rough continuity of dip beneath or above the coal zones. Beyond boundary faults such as 4 and 8 no such projection is reasonable. Similarly, other faults outside the coal area are difficult to detect from present drilling or geophysics.

GEOLOGICAL MATERIALS

9. A summary description of the coal, interburden, burn zone, overburden, underlying strata and surficial materials is given below.

Coal

10. The coal has been sub-divided into four zones since the preparation of Report No 2. The basis of this sub-division has been the characteristic traces of the natural gamma and bulk density logs within the coal deposit. The coal, with ash contents of 10% to 40%, is interbedded with varying thicknesses of rock and carbonaceous material. Zones have been based on the uniformity or variation in ash content: in very general terms the zones are:-

A (top)	Widest scatter of ash contents, thick beds of good quality coal.
B	Good quality coal, more partings than Zone D.
C	Poor quality coal, improving to base.
D	Good quality uniform coal with few partings over 1 ft in thickness.

11. The proving of the NE syncline containing about 30 million tons of Zone D coal has improved the overall coal quality and balances the loss of coal resulting from better knowledge of the interbeds. More precise details of the overall quality of each zone are given in Chapter III. Zones B, C and D may be 200 ft to 300 ft thick (possibly less on the margins of the deposit) and Zone A is up to 600 ft thick. Further sub-division of these zones may be possible on the basis of more refined geophysical analysis or lithological correlations. DCA have attempted to extend the zoning of the coal from the centre of the main western syncline to the margins of the deposit. Whilst it is inferred in the sections and structure plans that the zones are lithostratigraphic units, it is recognised that the coal deposition may have been diachronous and that there may not be exact correspondence with some of the zone boundaries and possible lithological marker bands. Lithological variations (both thickness and quality) are marked in the deposit, especially above Zone D, and increases in ash content and oncoming detrital materials are noted on the western and southern parts of the area and are apparent on the cross sections. The precise nature of these variations is uncertain; they may represent true facies variation with interfingering of materials, or gradual compositional variations or local unconformities may be present.

Interburden

12. The interbedded materials comprise mainly claystones and siltstones, some of which contain swelling clays and some iron carbonates. During 1976 an investigation was made of the mineralogy of the coal sequence based on parts of four boreholes. The findings discussed in detail in GA Report No 6 are:-

(i) Interbedded material is absent or thin in the sampled section of Zones B and D (Zone B is, however, much intercalated with detrital sediments to the west and south of the deposit). Zones A and C have thicker and more frequent interbeds. Table II shows this quite clearly.

(ii) Kaolinite (non-swelling) is the main clay mineral in the lowest three Zones - B, C and D.

(iii) Montmorillonite (and feldspar) is more common in Zone A indicating a volcanic source for some of the upper interbeds.

(iv) Interbeds are much sheared and are a likely locus of eventual movements.

Burn Zone

13. Burn zone is defined as the area of outcrop or near outcrop coal and contiguous strata which has been or may be burnt, or which is still burning. The end product is a slag-like clinker suitable for road construction. A burn zone appears to be present over much of the sub-surficial outcrop of Zone D. Its full extent is not known, particularly to the south, but in places it may have an outcrop width of more than 1,000 ft.

Overburden

14. The overlying strata consist mainly of siltstones and clayey siltstones which are weak and frequently bentonitic or carbonaceous. Thin bands of volcanic tuff are present and much of this sequence above Zone A is sheared or brecciated: coarser detrital sediments appear on the east of the deposit.

Underlying Strata

15. The underlying strata beneath Zone D are poorly cemented siltstones, sandstones and conglomerates (called "mixed detrital rocks" by DCA). Some bentonite is present as a cement and bands are calcitic; towards the north the strata beneath Zone D appear to be less coarse and more carbonaceous. It cannot be assumed that the sandstones and conglomerates are necessarily stronger than the siltstones or clayey siltstones, owing to bentonitic fines in the matrix. Both the sandstones and siltstones consist of rock fragments and minerals derived from volcanic sources. Breccias with a more granitic aspect have been recorded on the western limits of the area and strong andesite of the Kamloops Group has been detected on the east, but neither rock type should be an important component of the excavation; further drilling would be required to delimit both areas.

Surficial Materials

16. The surficial materials have been well described in GA Report No 6 and are summarised in their Table I of Section 3. The west side of the valley, the most unstable with existing mudslides and bentonite boils, comprises glacial till, slide debris and bentonite with breccias, possibly of colluvial origin, and glacial lake deposits. The eastern side of the valley mainly comprises glacio-fluvial materials with some lake deposits. Hence the west of the valley is most variable but incorporating much weak, cohesive soil while the east is predominantly granular. Recent drilling indicates an overall slope at the base of the surficials towards the north-east with the possibility of a buried channel lying east of the present course of Hat Creek. GA Fig 8, Report No 6, shows the location of the surficial material referred to above.

MINING IMPLICATIONS OF THE GEOLOGICAL MATERIALS

17. The implications of the geological materials in regard to mining are summarised below.

Mixing of Zones in Mining

18. As a result of faulting and the presence of two synclines and an anticline there appears to be a reasonable balance within the stages of the proposed development of the proportions of the various zones. After the initial coal production in shallow Zone D coal there is sufficient mixing of the zones to prevent wide variations in quality (see Chapter III). This confirms the preliminary findings.

Coal Handling

19. It is concluded that the handling of coal with bentonitic partings will not present severe problems due to inherent stickiness. Bentonitic partings are probably limited to Zone A and of sufficient thickness to permit selective mining. The segregation of interbedded waste will be largely of beds which dip along or out of operating benches.

Handling and Trafficability Problems of Bentonitic Overburden and Underlying Strata

20. Some handling and trafficability problems are possible on the overburden and underlying strata where it is bentonitic and especially when softened or remoulded as in some of the surficial deposits on the western side of the area. This item needs further investigation.

Strength of Materials

21. GA Report No 6 gives details of the material strengths (Section 5) and the strongest materials likely to be encountered are the burn zones, calcareous strata underlying Zone D and boulders of stronger, nearby in-situ material. Limestones and in-situ andesite are not likely to constitute significant excavation areas. Some concretionary ironstone nodules in the coal have been noted. The coal itself is the next strongest (1,250 lb/in^2 uniaxial compressive strength) and in view of the wide spacing of joints and the range of strength within all the sediments, both coal and interburden, light blasting appears to be necessary. On the basis of experience with materials of comparable compressive and tensile strength, bucket-wheel excavators would be able to operate in unblasted overburden and underlying strata. The diggability of the coals by bucket-wheel excavator would, however, require further investigation.

Material Density

22. The likely bulking of the mined materials in transport and in dumps has been assessed (see Section 6, GA Report No 6) and requires further investigation on the field scale. For the purposes of this report, the same bulk densities have been used as in Report No 2 (see Table I), these agreeing with findings during the recent drilling. A small adjustment from 1.29 to 1.27 short tons/bank yd^3 has been made in the case of rom coal to allow for greater separation of higher density parting material.

FURTHER GEOLOGICAL INVESTIGATIONS

23. Further geological investigations are recommended to undertake in-fill drilling within the confines of the pit as proposed herein. The full pit area should be covered by drilling at 500-ft centres, especially to the south of the area. Closer spacing at or near the centre of 500 grid squares should then be undertaken in the following settings:-

(i) proximity of faults or changes of strata dip,

(ii) areas of facies change,

(iii) burn zones.

24. Drilling should lie almost entirely within the pit confines except to investigate geotechnical or major structural problems. Core recovery and extrusion should be of a higher quality than heretofore and any contract for a new drilling programme should be drawn up with the approval of the client's geotechnical and mining advisers as well as their geological consultants.

GEOTECHNICAL CONSIDERATIONS

General Findings

25. The geotechnical aspects of recent investigations have been covered in GA Report No 6; this section highlights the principal findings. To summarise, the excavated slope angle originally suggested by PD-NCB has been confirmed but the disposal of waste materials requires the use of retaining embankments on a larger scale than envisaged in Report No 2.

26. The strength of most overburden, interburden and underlying strata is substantially weaker than was indicated in earlier reports: it is apparent that even a few hours drying appreciably increases the strength of the clay matrix sediments above in-situ values. Moreover, the weathering characteristics of even the conglomerates are poor and rapid breakdown in spoil dumps is expected. There is a wide range of strengths within the constituent rock types as shown in Fig 8, GA Report No 6, but the average uniaxial compressive strength is less than 500 lb/in^2 in the overburden and less than 1,000 lb/in^2 in the coarser underlying strata. The shear strengths of constituent materials, mostly tested triaxially, are also lower than anticipated. The coal is the strongest major constituent with an average uniaxial strength of 1,250 lb/in^2: it is also a major aquifer.

27. Groundwater levels appear to follow topography (and base of surficial contours) rather than specific structural features; there is some indication that Hat Creek is a groundwater discharge area. The claystones and related detrital rocks are distinctly less permeable ($k = 10^{-8}$ cm/sec) than the coal($k = 10^{-4}$ cm/sec). It seems likely that on the scale of several benches distinct aquifers and aquitards will be present as coal seams and clayey partings.

Slopes

28. The overall slope angle of 16^0 for the excavated pit slopes is confirmed: GA Report No 6 considers that, given the low triaxial strengths, circular failure modes are most likely. This slope angle, which clearly varies with slope height and material strength, is based on proposed pit depths and lower-bound peak strengths. Residual strengths were not considered appropriate. In most situations some prior drainage will be necessary to maintain this angle. Where shearing is present major dewatering or slope reduction will be required.

29. The extent of potential structurally controlled slope failures is not clear. Simple analyses of possible west face slopes with strata inclined towards the excavation with dips 5^0 to 10^0 greater than the pit slope show potential failure situations involving four or five benches with a curvilinear failure mode (see Plate 17). A closer examination of various actual slope and structure configurations is necessary at the design stage, together with an appraisal of the effect of likely groundwater conditions. Bench stability will probably be more structurally controlled: since bedding is the most continuous structural feature and is mainly inclined at angles which are greater than the pit slope but flatter than the bench slopes, its orientation will probably be important. Benches may be expected to collapse most frequently on the west and northern sections of the excavation where pit slope and dip often coincide. Pre-drainage may assist bench and pit slope stability in these areas.

Location of Access Ramp

30. The preferred location for the access ramp on geotechnical grounds is some 15^0 further east than originally suggested, along an infilled sub-surficial channel. This will increase the excavation compared with that in Report No 2, but since a greater proportion of the slope would lie in gravel rather than sheared clayey siltstone, the risk of failure is reduced. Where the ramp lies at a depth less than two-thirds of the thickness of gravel and no excess piezometric head is present, side slopes of 33^0 would seem possible. Where less favourable conditions are present, such as clayey siltstone or high piezometric head, the ramp side slopes should be reduced in the lower sections to about 22^0 and permanent dewatering will be required, with advance dewatering, say 2,500 ft from the line of the ramp.

Surface Instability

31. Surface instability is limited to the western side of the Hat Creek valley in the area at Openpit No 1. Several slide areas have been noted, the most important being:-

(i) Between Aleece and Finney Lakes. There has been much displacement in the past but there is little sign of movement at present.

(ii) The mudslide running north-east from Aleece Lake to Hat Creek; this is active and is part of a larger, older and, apparently, inactive slide. The active mudslide has a volume of 17 million yd^3 and will require dewatering before mining and removal at the onset of excavations.

Waste Dumps

32. The waste material is substantially weaker than was previously thought likely. Moreover, given even a low seismic risk, the foundation of the principal waste disposal location in Houth Meadows is such that an embankment will be required to retain all waste (rather than a series of bench embankments). The overall result is that a lower proportion of the waste can go into Houth Meadows and that disposal north-east of the access ramp is impracticable in the absence of valley sides to assist in the retention of waste. Medicine Creek to the south-east of the pit is recommended as an area able to accommodate the balance of the waste.

Waste During Construction

33. It is considered that only selected sand, gravel and till will be suitable for embankment construction and that the remaining waste will be very weak. The embankment construction materials are located mainly on the east side of Hat Creek valley: some of the weakest materials from the west of the valley will require dumping early in the development of the waste dumps. The retaining embankments would have an outside slope of 22^0 and would be constructed in stages at or very near to the locations, where acceptable foundations have been proved (Houth Meadows) or inferred (Medicine Creek). The suggested realignment of the access ramp away from the Houth Meadow embankment will further improve stability. Behind the embankments dumped waste should have slopes not exceeding 6^0 (the gradient of the active mudslide), and when the slope height exceeds 250 ft this should be reduced to less than 3^0. Construction of the waste dumps should be on the upstream principle: downstream dumping from the heads of the valley is unacceptable owing to the risk of flowslides in saturated clayey siltsones. (It is also not in accordance with the Coal Mines Regulations Act, see Report No 8.)

Recommendations for Further Work

34. Recommendations for further geotechnical work are given in the GA Report No 6 and cover aspects of groundwater, slope stability, waste dump embankments, materials testing and excavation for exploration. It is emphasised that further pumping tests are required, especially in the coal, since identical storage coefficients for coal and siltstone are unlikely and present assumptions regarding dewatering could be in error. In view of possible structural control to bench slopes and some pit slopes, future testing should examine far more closely directional and residual strengths in the coal interbeds and subjacent strata. Similarly, use must be made of trial excavations, shafts, etc, to obtain data on bulking and handling characteristics as well as behaviour in excavated facies or waste dumps. One of the principal concerns, however, is that geological prospecting should be linked with geotechnical objectives so that adequate logs and samples can be obtained from acceptable core recovery.

CHAPTER III

COAL QUALITY

GENERAL

1. The accurate determination of coal quality, particularly in terms of heat and ash content, is important because it determines the amount of coal which has to be mined and the amount of ash which has to be handled by the ash disposal system. Other properties of the coal are also important to the boiler designer, eg grindability, propensity to fouling and combustion characteristics. As all these characteristics become more unfavourable so the boiler design becomes more expensive and operation more difficult. Report No 2 dealt only briefly with the subject of coal quality, but considerably more information has become available from the 1976 in-fill drilling programme. (Much of this has not yet been processed and assessed.) Much of the activity in the next phase of work will be directed to a better understanding of the quality of the coal and the whole question of whether or not to beneficiate.

2. In the openpit type of mine under consideration, all the coal (and waste) within the outline of the pit will have to be mined and material which is clearly "coal" will go to the power station whilst material which is clearly "waste" will be dumped. However, in this particular mine there is a whole range of intermediate material which is termed "low-grade coal" or "carbonaceous waste". This material may be low-grade due to the following reasons:-

(i) inherent ash content,

(ii) mixing of beds due to geological processes, eg rubble zones,

(iii) failure to separate "coal" and "waste" in mining because of:-

  (a) inability to recognise the materials,

  (b) inability to separate the materials due to unsuitable machines or lack of operational discipline,

  (c) excessive cost of separation.

3. Assessing the ability to recognise "low-grade" coal visually and by bench sampling, a decision has to be taken whether to label a particular block as "coal" (ie power station fuel), "waste" or possibly "low-grade coal" (ie coal which may become a fuel in the future because of changing energy economics or technology). Therefore, definitions of these materials are required and the simplest criterion to use is ash content. There would then be a "cut-off" ash content for coal and another for low-grade coal, if this material were to be stockpiled.

DETERMINATION OF COAL CUT-OFF GRADE (IE BTU OR ASH CONTENT)

4. It is customary in the case of base metals where grade is variable to attempt to classify the ore reserves in terms of grade, and a curve can be drawn showing the quantities of ore for each grade interval and the cumulative quantities for each cut-off grade. (In many complex deposits this may be a difficult exercise.) For higher cut-off grades the average grade of ore mined is higher but the loss of marginal ore represents a loss of resource and the overall ore and metal recovery decreases.

5. Figure 5.3.1 of the Integ/Ebasco draft report of December, 1976, gives a cumulative histogram of the predicted rom coal quality. They then use this to predict the effect of different degrees of washing and/or blending.

6. Using information from this figure, it is possible to consider the implications of rejecting high-ash coal and the effect different cut-off points would have on the properties and the tonnages of the "saleable" coal. Basically, increasing the level of the cut-off point (in terms of CV) will increase the average CV of the saleable coal but it will also increase the Btus lost in the rejects. This loss of Btus means that more in-situ coal will be required to produce the same saleable output. This applies whether the useful output is measured in Btus or in tons.

7. Plate 18(a) shows the cumulative percentage of the histogram plotted and superimposed by a smooth curve. According to Integ/Ebasco, all the coal has a CV of <8,000 Btu/lb. However, no minimum CV is given explicitly. They state that all material having more than 75% ash on a dry basis has been excluded. There have been a number of Btu against ash regression analyses produced for Hat Creek coal. Of these, one presented by DCA in their report of September, 1976, and based on all data from drill holes 135 and 136, has been used. This gives a CV corresponding to a dry ash content of 75% as about 1,500 Btu/lb (at 20% moisture) and this has been adopted as the zero.

8. From Plate 18(a) it is possible to reconstruct a histogram showing the percentages falling within different calorific value intervals. This is shown, based on 500-Btu/lb intervals, on Plate 18(b).

9. Using Plate 18(a), it is possible to determine directly the percentage of coal that must be rejected for a given cut-off calorific value. For instance, a cut-off of 3,600 Btu/lb results in an average value of 5,950 Btu/lb (Integ/Ebasco figures). Other factors which are important are the effect on the average CV of the "saleable coal" and the percentage heat loss in the rejected coal. These are shown plotted against cut-off CV on Plates 19(a) and 19(b). A scale representing the cut-off ash content is also shown. These are based on the regression analysis quoted earlier.

10. Clearly, these curves will be refined as soon as more data can be included in the data base and processed. The percentages will then be replaced by actual tonnages of reserves. Because of zoning in the coal, which implies abrupt changes in coal properties and also the effect of the mining sequence by stages, it will most likely be necessary to produce such curves for each stage and they may differ considerably. The effect of this could be that the ash cut-off might be different in each stage.

Marginal Coal

11. A further important consideration is that marginal coal (ie near the cut-off grade) is mined at marginal cost. In other words, the particular block of coal has to be mined and dumped or mined and burnt. The marginal cost is therefore the difference between the two, which may well be substantially lower than the average cost (corresponding to average grade). Clearly, however, too much marginal coal would reduce the average grade to unacceptable levels.

Significance of Ash Cut-Off

12. It can, therefore, be concluded that the determination of the ash cut-off is of great significance to the mine in the following respects:-

(i) quantity of coal to be produced,

(ii) average ash content,

(iii) range of ash content,

(iv) loss of resource,

(v) cost of coal and of heat energy.

(All coal preparation processes involving beneficiation (up-grading) result in the same kind of problem as described above, primarily loss of resource in the rejects.)

13. These factors cannot, of course, be considered in isolation from the power plant and eventually the objective should be to produce the lowest cost electrical energy concomitant with the optimum use of the resource. The latter criterion can be changed during the course of mining, particularly in respect of final pit limits but, unfortunately, once the mine has closed it would be very difficult and expensive to re-open it to recover resources which have been abandoned.

QUALITY DETERMINATION

14. Attempts have been made to determine the quality of the coal by zones and by stages of mine operation. In each case the results are interim because all the data now becoming available were not used in their computation. However, they are given below as they are indicative of the situation. In due course the computer system will give far more accurate results - by zone, by stage and by any other production period as required.

Coal Zones

15. Table III gives the results of a statistical investigation of the sample data from drill holes 135 and 136 from the Interim Report by Dr. A.J. Sinclair dated 20th September, 1976. The averages adjusted for 20% nominal moisture content are also given. Since these results only apply to two drill holes they are clearly not statistically representative of the deposit as a whole even though the drill holes penetrated the full sequence of beds. Also, they are not representative of the deposit as it is likely to be mined (ie to the 2,400-ft elevation). However, they can be accepted as indicative of the coal properties of the four zones which might be expected. Clearly, the number of samples is not large enough to reduce the standard deviation to acceptable levels.

16. Dr. A.J. Sinclair also reported (Interim Report "An Evaluation of Pre-1976 Proximate Analyses, No 1 Deposit, Hat Creek", dated 18th August, 1976) the coal zone properties. Some coal samples which could not be assigned to any zone and the effect of interburden dilution are excluded. Some of the results (dry basis) and the results adjusted to 20% nominal moisture are given in Table IV. This table also shows the values used in Report No 2 for the evaluation of the rom coal for comparison. The correspondence of the overall heating value of 5,594 Btu/lb and the assumed value used hitherto, 5,500 Btu/lb, is good, which is encouraging despite the rough methods of calculation used in both estimations. The value of Dr. Sinclair's figures is limited to some extent because of the sampling procedures and the fact that they refer to the whole deposit not the mineable part and also they do not include the more recent data from the 1976 in-fill drilling programme.

17. The zone thicknesses vary considerably, of course, and are not yet clearly defined, but one set of determined values is as follows:-

Zone	Average Thickness ft
A	620
B	262
C	316
D	260
	1,458

Stages of Production

18. It is clearly important to predict the way in which the coal quality will vary in accordance with the sequence of mining, particularly as in this mine the sequence cannot be changed very much to obtain a better blend because of the geotechnical constraints. Therefore, an attempt has been made to calculate the average quality of the coal contained within each stage of mining as defined in Report No 2. (The stages have been adjusted in this report but the same general conclusions apply.) This was done by drawing up a computer program to abstract the drill hole intercepts in each stage from the drill hole data base in the BCH computer. Unfortunately, only 30 drill holes were available at that time. Table V is a summary of the computer print-outs obtained (7th October, 1976). The 20% nominal moisture values have been used and again the overall heating value (5,680 Btu/lb) is in close agreement with Report No 2.

19. Table XII shows the approximate proportion of in-situ coal mined from each zone during each stage of operations, and Plate 20 plots the results in terms of the resulting quality. The quality is high initially (Zone D) and then remains reasonably steady as the lower-grade zones are mined. Again, this plate is only intended to be indicative since insufficient data have been included.

Conclusions

20. The conclusions which can be drawn from these interim calculations can be summarised as follows:-

(i) Zone C is the lowest in quality followed by A, B and D in ascending order (see also Chapter II).

(ii) The stage qualities, which usually contain a mix of zones, are reasonably consistent except the final stage (Stage 8 - Report No 2) which is lower in quality.

(iii) Zone B and Stage 8 have the highest sulphur contents but again the mixing of zones in the stages marginally reduces the variation.

(iv) The short-comings of these estimates illustrate the urgent necessity for developing computer programs to process all the data now available (and new data as they come in).

(v) Selective mining must be examined in more detail and the proposed sampling procedures applied rigorously.

COAL QUALITY FOR POWER STATION

21. Integ/Ebasco, in their interim report (December 1976), have based their "conceptual design other than for the boiler" on the following coal quality (ie rom):-

5,950 Btu/lb

28% ash

20% moisture

(Presumably the boiler design will be based on the extreme range of the quality characteristics.) This is based on an ash cut-off of slightly over 44% (3,600 Btu/lb) and a 600-ft pit. It is assumed that low-grade coal and, of course, waste can be successfully separated in the mine. They also assume that the heating value may range from 3,600 Btu/lb to 8,000 Btu/lb based on 20% moisture, but that blending and preparation can reduce this range. According to Plate 18(a) this would result in the rejection of 10% of the coal with a heating value between 1,500 Btu/lb and 8,000 Btu/lb, but according to Plate 19(b) the heating value of the rejects would represent only about 5% of the total.

22. The 20% moisture comment referred to above is only a nominal figure. Comprehensive tests are required to determine the moisture content of the coal as mined but preliminary results from recent drilling suggest that it will be nearer 25% than 20%. (See coal moisture contents in Table A8-1, GA Report No 6). There would be some drainage and drying in the stockpiles but any wet preparation process used could increase this figure.

23. For the sake of some consistency, the Integ/Ebasco value of 5,950 Btu/lb for rom coal, given in paragraph 21, has been adopted as the basis of this report pending the results of a thorough investigation into selective mining and an evaluation of the loss of resource which might be entailed. Report No 2 used a value of 5,500 Btu/lb for rom coal and was therefore more pessimistic. Table VIII gives the resultant coal production on both bases.

SAMPLING

Borehole Sampling

24. The accurate prediction of the quality of the coal as it comes out of the mine is clearly most important. During the exploratory drilling phase samples were obtained from borehole cores and this, of course, continued during the in-fill drilling phase. These samples have enabled many of the important properties of the coal to be determined but the small size of the samples limits the testing to laboratory scale and for some tests larger samples are required. Also, the sampling procedures have been adjusted to improve the accuracy and flexibility of assessment, particularly to enable selective mining options to be assessed. Table VI summarises the changes in the sampling procedures. It can be seen that these have been refined as knowledge of the deposit has increased and the problems of quality determination become clearer. The mass of data obtained during 1976 is in process of being computerised and the completion of the current work on computer systems should enable a whole range of coal quality reports to be produced.

Bucket-Auger Sampling

25. Pilot-scale washability tests were carried out by Birtley Engineering at their Calgary laboratory in 1976 on 3-ft diameter bucket-auger samples but these samples suffered from a number of limitations, ie:-

(i) The maximum depth reached was 95 ft and hence only coal within that depth limit was sampled. The holes were located between 79,000 N and 80,500 N and 18,000 E and 21,000 E, and the coal sub-outcrops in these areas have subsequently been identified as belonging to Zones A, B and C. Zone D does not appear to be represented.

(ii) The action of the auger churned up the coal and created excessive mixing and fines; also the top size was limited to about 4 in.

Whilst the results of the tests were of interest, they cannot therefore be regarded as representative of the coal which will be produced as mining progresses.

Sample Mine

26. It has been appreciated that, in order to obtain more representative bulk samples, a sample mine would be required, the objectives of this mine being as follows:-

(i) to obtain bulk samples of coal, disturbed as little as possible,

(ii) to obtain bulk samples as representative as possible of all the main coal types - specifically Zones A, B, C and D,

(iii) to determine the extent of weathering of the sub-outcrop and to ensure that the samples are not weathered,

(iv) to observe the behaviour of the exposed coal and interburden surfaces particularly as regards:-

- slaking
- oxidation and heating
- change in strength
- gas emission,

(v) to examine the detailed exposures of the interbeds,

(vi) to assess the strength of the rocks as regards excavation, blasting, slope stability, etc,

(vii) to examine fracture patterns, cleat, faulting, etc,

(viii) to observe water flows,

(ix) to obtain access to freshly-exposed material for the whole range of geotechnical test work.

27. The tests which it is hoped to carry out on the coal samples include:-

(i) size grading, screening, crushing,

(ii) beneficiation,

(iii) burn tests - pf and fluidised bed,

(iv) handling, stockpiling, etc (particularly degradation and spontaneous combustion),

(v) gasification, etc, as required.

28. Two types of sample mine are under discussion, ie:-

(i) surface trench,

(ii) underground.

Surface Trench

29. The surface trench is superficially attractive as the skill required to excavate it is such that any competent civil engineering contractor could do the work. However, it would have to be long and deep to reach the sub-outcrop of the four zones and there would be formidable water and slope stability problems. The cost of such a trench would be in excess of $1 million and the disturbed area would be about 17 acres for the trench itself plus dump space. However a trench would allow the effect of short-term weathering on a slope stability to be observed and the liability of the coal to spontaneous combustion on exposure to be checked. It would also permit bulk handling problems to be examined. The main objection is that only the sub-outcrop coal could be sampled and the depth of weathering is unknown.

Underground Mine

30. This mine would comprise vertical shafts or inclines or both, connected by cross-cuts, and it would give a good cross-section of each zone of the deposit and also check on the extent of weathering. The cost would again be in excess of $1 million but the disturbed area would be minimal. Good interburden and geotechnical data would be obtained. The main disadvantage is that greater technical skill would be required, although suitably qualified contractors are available.

Bench Sampling

31. During mining operations, bench sampling will provide detailed information of the materials within each block and this more accurate information will be substituted for that interpolated from surface boreholes. Design and long-range planning, however, has to rely on the borehole data.

SELECTIVE MINING

32. All mining is selective to some extent, if only in regard to the distinction between economic mineral and waste, the objectives being as follows:-

(i) to reduce the loss of economic mineral,

(ii) to reduce dilution by waste material,

(iii) to improve the grade of economic mineral as mined,

(iv) to reduce the variation in grade, particularly short-term,

(v) to reduce the amount of economic mineral to be handled by removal of the extraneous waste material as early as possible,

(vi) to reduce the amount of impurities which adversely affect later processes and/or reduce the value of the final product,

(vii) to reduce the amount of impurities which have an adverse effect on the environment, eg sulphur in coal.

33. These objectives are achieved by:-

(i) adequate knowledge of the properties of the economic mineral and the waste,

(ii) adequate knowledge of the spatial distribution of the economic mineral and the waste,

(iii) proper selection of mining methods and equipment,

(iv) discipline and quality control during mining operations,

(v) adequate knowledge of the requirements of the downstream processes (or markets).

All these factors cost money and hence a compromise has to be reached between what is desirable and the cost of achieving it. In the case of Hat Creek, the compromise involves a reconciliation of the demands of the power plant designers and operators and the problems which the satisfaction of these demands presents to the mine designers and operators.

34. The spatial distribution of coal and waste in Deposit No 1 at Hat Creek is now fairly well known (see Chapter II) in general terms and four stratigraphic zones have been recognised in the coal. Interburden beds (or partings) occur within the mass of the coal, although continuity is difficult to establish. Also, there are some mixed zones where disturbance has been severe. The beds dip at all angles up to about 60^0 from the horizontal. The interburden material consists largely of claystones, some of which are highly bentonitic, the latter being exceptionally weak mechanically and highly plastic.

35. Because of the shape of the deposit and the openpit method of working which has been selected, all the material within the pit boundary at any time must be mined out, ie it cannot be left in situ. Furthermore, the objective is to extract the majority of the coal above the 2,400-ft elevation in the first instance and hence there can be no question of leaving either coal or waste behind except when determining the position of the final cut at the end of mining operations. Therefore, any selective mining which involves "high-grading" at any stage means that lower grade material will have to be mined at some other stage. Furthermore, from slope stability considerations, the shape of the pit must be kept regular and not be distorted unduly.

36. Since mining will be taking place simultaneously at a number of points around the pit, it will be possible, within limits, to "select" blocks which, when blended, will reduce the variance in the coal quality.

37. Low-grade material, eg rubble zones, can also be readily identified and sent out to the low-grade coal dump.

38. The main problem of selective mining, therefore, is that of dealing with the interburden material. Thick beds of any orientation present no particular problems as the production machines would load them in the normal way and the waste material would then be directed to the waste disposal system. Thin beds,

however, are a different matter, particularly those with awkward orientations. Different equipment has to be used and costs inevitably increase. The methods and equipment required and costs are discussed in Chapter V. Coal quality considerations are dealt with below.

Design Problem

39. The design problem is to determine the following:-

(i) the amount of waste which can feasibly be removed,

(ii) the cost of removing it,

(iii) the corresponding benefit in terms of improved coal quality and reduced tonnage.

In fact, a number of possibilities should be considered and optimised in terms of both mining and the overall power project. In order to handle the amount of data involved in this, a computerisation system is being developed. This will be able to estimate the number and thickness of interburden beds in each mining block and to calculate the effect on coal quality and quantity of selectively removing beds of decreasing thickness. It is possible to remove virtually all the interburden material by hand-labour, eg using shovels, scapers, brushes, etc, but this would obviously not be economic in this project. It has been tentatively concluded that beds down to 3 ft thick can be separated by the main production equipment; beds from 3 ft to 1 ft thick by special smaller equipment; beds less than 1 ft thick would not normally be separated unless they consisted of particularly deleterious material, eg sticky bentonitic clay, high sulphur material, etc.

40. In order to make an interim estimate of the quantities involved, an interburden bed count has been carried out using data from the 1976 drilling programme (see Table II and Chapter II). From the interbed count the proportions of various ranges of interbed (partings) within the whole deposit have been estimated and these proportions used in estimating equipment and costs.

STOCKPILING, BLENDING AND COAL PREPARATION

Stockpiling

41. The size of stockpile tentatively suggested for this project is 1 million tons which would give just over one month's supply at the full output of the power plant. This is considered to be about the minimum necessary, particularly as Openpit No 1 and the power station are captive to each other, ie there is no other source of supply for the power plant and there is no other market for the coal. (This situation could, of course, change if additional mines and/or additional consuming plants were located in the area and provided with high-capacity transport links.)

42. Such a quantity of coal can clearly only be stored in ground stockpiles and it has now been decided to locate these stockpiles at the mine mouth, transporting the coal to the power plant by duplicate conveyors.

Blending

43. These large stockpiles would, in any event, mix the coal to some extent but in this project, since the coal quality is likely to vary considerably despite the best attempts at selective mining in the pit, provision has been made for a blending stockpile. In basic terms, the variance of the input can thereby be reduced to any particular variance of the output (ie boiler feed) by increasing the number of increments or passes of the stacker. The fundamental relationship is as follows:-

$$V_2 = \frac{V_1}{n}$$

where V_1 is the variance of the input

V_2 is the variance of the output

n is the number of increments

(The complete theory, which is based on probability statistics, is more accurate, but whether its use is necessary depends on how critical are the requirements.)

44. The increments can take the form of horizontal layers, windrows or chevrons depending on the stacking system used and the number of increments depends on the requirements and practical considerations. The commonly-used reclaimers are bucket-wheels and bridge-type or barrels with or without rakes. Long ridge piles are preferred to reduce end effects and various degrees of sophistication are possible depending on the importance of quality variance. This again is a matter for some compromise between the mine, the stock-piling/reclaiming system and the boiler requirements. Further comment on blending stockpiles is given in Chapter VI.

Coal Beneficiation

45. Blending does not, of course, change the average quality, but only the variance. If the average quality is too low then some form of beneficiation is required, ie:-

(i) differential crushing and screening,

(ii) dry cleaning,

(iii) wet cleaning.

46. Differential crushing utilises the difference in crushing strengths of coal and waste and it is possible that claystone rejection could be achieved by either the Bradford breaker or the Siebra crusher. Differential screening plays a part in both these crushers and it could also be utilised if waste material concentrates in either the oversize or the undersize.

47. Apart from the obvious problems of loss of Btus and waste disposal, both dry and wet cleaning plants would be of very large size if the total output were treated. It seems, therefore, that the output would have to be split, eg by screening, and only part of it sent to the cleaning plant, the cleaned product being mixed back into the main stream using instantaneous ash monitors or bulk density meters to control the final quality. The feasibility and economics of such schemes would have to be evaluated after the appropriate tests had been carried out. On balance, it is considered that cleaning is to be avoided if an acceptable boiler design can be produced to burn the untreated, but blended, coal.

Other Stockpiles

48. Apart from the main operational stockpile, there will inevitably be a stock of coal which will be accumulated during mine development, presently estimated at 1.1 million tons. This coal will probably not be sized and will not be blended. It could be fed gradually into the blending system when it starts operating or kept as a strategic stockpile, in which case it would have to be safeguarded against spontaneous combustion by grading, rolling and possibly sealing.

49. Additional stockpiles could be established as reserves against other eventualities, eg:-

(i) low sulphur - to be used if the sulphur content were too great at any period or if meteorological conditions were adverse,

(ii) low ash - to be used if the ash content were too great at any period.

50. A separate low-grade coal stockpile or dump is envisaged, this coal being that which is unacceptable at the present time even as a blend component, but which might become usable in the future and should, therefore, not be lost by admixture with other waste materials.

COAL SIZE DISTRIBUTION AND ITS IMPORTANCE

51. The maximum size of coal produced at the face, and indeed the pattern of size distribution, is controlled by the degree and method of blasting. The two main parameters are the burden and spacing of the blast holes and the quantities of explosives used. Both of these will have to be determined by experience, but for estimating purposes a combination of a 10-ft hole spacing and a powder factor of 0.3 lb/ton of coal has been assumed (see Report No 2, page 21). This should result in a product with no single piece exceeding 4 ft in any dimension.

52. The ultimate object is to reduce the coal to pulverised fuel for combustion in the power station boilers. Some of this size reduction will be by intentional crushing or pulverising but some will occur as a by-product of the various transfer, transport and storage stages through which the coal passes. The more significant of these are as follows:-

(i) initial blasting at face,

(ii) loading into shovel and subsequent discharge to off-highway truck,

(iii) tipping from truck on to grizzly,

(iv) oversize lumps broken to pass through grizzly,

(v) further breakage in hoppers and at transfer points,

(vi) discharging from conveyor and passing through crushers,

(vii) transporting from crusher to stockpiling area,

(viii) laying down in stockpile and compacting, if required,

(ix) weathering and oxidisation in stockpile,

(x) reclaiming from stockpile,

(xi) passing through re-crusher and then on to power station.

53. This ends the responsibility so far as the mine is concerned but the following will occur within the power station:-

(i) discharging from conveyor to power station bunkers,

(ii) in feeder supplying mill,

(iii) pulverisation in coal mill.

54. The design of each link in this chain requires a knowledge of the properties of the incoming material presented to it. This problem is not quite so formidable as it seems since these breakages and the products resulting from them can be predicted mathematically. One method of doing this is by means of the theory proposed by Rosin and Rammler and which bears their names. This is described in Appendix "C" and Plate 21 shows its graphical representation. For the present it is sufficient that it makes possible, by the use of logarithmic graph paper, to obtain a straight-line plot relating to the percentage retained on a given size of screen against the screen aperture. The position of this characteristic line alters as the coal undergoes the various crushing operations but this alteration is a "modification" rather than a radical change.

55. There is, at present, insufficient information on the properties of the Hat Creek coal to predict much about its crushing characteristics, since this depends on the production of coal under something approaching full-scale mining conditions. Many useful data can, however, be expected as a result of the mining of the samples required for the power station burn tests. A sample trench would be better than a sample shaft as the method of excavation would approximate more closely to that proposed for full-scale mining, ie generally two free faces whilst in the underground mine only one free face would be generally available and very different equipment would be used.

Example of Rosin-Rammler Theory

56. There are only limited data available as to the size distribution of Hat Creek coal and, since these have been obtained from bucket-auger samples, their relevance to full-scale conditions is limited. However, as an example of the Rosin-Rammler theory, Plate 22 shows the plot of a set of size gradings carried out by Commercial Testing of Vancouver and quoted by G. Armstrong in his report of 15th June, 1976. This sample had been divided into eight sizes from 2 in to minus 200 mesh, and it can be seen that it is possible to represent the plot as a straight line with a slope of 0.64. This slope is referred to in the Rosin-Rammler theory as the "size distribution constant". For rom coals, this normally lies between 0.62 and 0.89 (see Appendix "C"). Thus, although the absolute sizes obtained from the auger sample are small their distribution may not be too different from that of the rom coal.

57. Birtley Engineering, in their report of 13th August, 1976, give size distributions for the three samples which they tested. Unfortunately, they only graded them into four sizes which do not produce a particularly revealing plot. However, the "A" and "C" samples appear to give basically similar results to the Commercial Testing sample, while the "B" sample gives a size distribution constant of about 1.0. The same report also gives details of the samples after crushing to minus 2 in. These were graded into eight sizes and the results are generally compatible with those quoted earlier.

58. If it is postulated that the rom coal will have the same size distribution constant of 0.64, then it is possible to predict its size distribution. As has already been suggested, light blasting on a 10-ft pattern could result in an absolute top size of 48 in. The formula can only handle finite percentages greater than a given size

so for the present purpose it will be assumed that this is equivalent to 1% greater than 36 in. This, then, corresponds to 20% greater than 7 in, 50% greater than 2 in, and 80% greater than three-eighths of an inch. This distribution is shown in more detail in Table VII. There is sufficient small material in this "mix" for it to be safely handled on 60-in belts without any pre-crushing. If the degree of blasting were increased to give only 1% exceeding 20 in then 50% of the material would be larger than 1 in. The complete distribution for this case is also shown in Table VII.

59. It must be emphasised again that this argument is only intended to illustrate the method and too much credence must not be put on the absolute values in view of the very limited data available at present.

CHAPTER IV

MINE PLANNING

GENERAL

1. The same mine planning principles that were described in Report No 2 have been used in this report, ie manual design and calculation (since the computer systems are still under development). The changes in the design of the openpit are marginal and due to further information on the following aspects:-

(i) geological,

(ii) geotechnical,

(iii) coal quality,

(iv) power plant coal requirements.

GEOLOGICAL ASPECTS

2. The 1976 in-fill drilling programme improved the geological knowledge of the shape and structure of the deposit and, in particular, found additional coal to the north-east. The shape of the openpit has therefore been modified accordingly.

GEOTECHNICAL ASPECTS

3. The 1976 geotechnical field work was concerned largely with the characteristics of the rocks from the point of view of slope stability since this problem is of prime importance in respect to operation and safety and also the economics of mining. GA Report No 6 strongly confirms the 16^0 overall slope adopted in Report No 2. (In some locations the safe slope would be less and in others it would be more.)

4. The main incline has been relocated further to the east to take advantage of more stable material there and it has been re-directed somewhat so that it "aims" towards the centre of the proposed pit.

5. The interbed analysis has indicated that bentonitic beds will probably not cause too much trouble in coal handling, as had been feared, neither do they appear to be so widespread. However, further work is required to check whether the results obtained from one set of tests apply across the deposit as a whole.

6. The hydrological investigation has provided information on the in-flow of water which might be expected although again insufficient work has been carried out in this area. Estimates have been made of borehole pumping costs to ensure the stability of the main access incline and pit slopes.

7. Because of the weak nature of much of the waste material, the dump designs have had to be changed and a second dumping site incorporated (see Chapter II).

COAL QUALITY

8. Chapter III deals with considerations of coal quality and indicates that this report is based on the quality for rom coal assumed by Integ/Ebasco, ie:-

Calorific value	5,950 Btu/lb
Ash	28%
Moisture	20%

The coal tonnages have, therefore, been adjusted accordingly, using the power plant heat rate of 10,443 Btu/net kWh (see Appendix "D"). It is considered that the next phase of the power plant and mine studies will be dominated by the need to resolve the problems in this area.

POWER PLANT COAL REQUIREMENTS

9. The power plant coal requirements have been modified in accordance with the power plant operating regime given by BCH (Appendix "D") and the coal quality determined as described above. Table VIII gives the power plant coal requirements over the life of the plant both at 5,950 Btu/lb and 5,500 Btu/lb, the value used in Report No 2. The difference in coal requirements overall is 27 million tons. Plate 23 shows the annual coal requirements and Plate 24 the cumulative. The 560-MW (500-MW net) generator unit heat rates used at different capacity factors are as follows:-

Capacity Factor %	Annual Heat Rate 10^{12} Btu
70	32
65	30
55	25

10. Table VIII also indicates that 1.1 million tons of coal would be mined before the first generator starts up, ie during the development phase of the mine. This coal would clearly have to be stockpiled but, in any case, a large stockpile would be required before start-up to ensure continuity of operation during the difficult expansion period. Subsequently, it is envisaged that a similar quantity would be maintained in stock throughout the operation. Whilst this would obviously be drawn down during the plant run-down period, this is ignored. In any case, losses from stock due to "carpet" loss, dust and rainfall might well account for this amount over the period 1984 to 2022, ie 38 years (0.3%).

MINE DESIGN

Pit Depth

11. No further information has been received to indicate that the coal could be worked to deeper levels and therefore the "600-ft" pit concept (ie pit floor elevation 2,400 ft) has been retained. (Incidentally, it is considered most unlikely that a decision could be taken on extension in depth until a considerable amount of actual mining experience has accrued.) Nevertheless, there are large reserves of coal down to the 1,500-ft elevation ("1,500-ft" pit) and therefore volumes and ratios have been calculated for this pit and the size indicated.

Pit Design

12. The same method of pit design has been used as in Report No 2. Starting from the rom coal requirements, the "in-situ" coal requirements have been calculated from Table II on the following basis.

13. Partings over 3 ft thick will be removed by selective mining while also attempting to remove waste down to 1 ft in thickness. On this basis, 13% of the 17% parting material will be removed leaving the remaining 4% as part of the rom coal. Similarly, it has been assumed that 11% of the 12% interlayered coal and waste will be removed as "low-grade coal" leaving 1% in the rom coal.

14. As in Report No 2, the 600-ft pit has been designed in eight stages and the expansion to the 1,500-ft pit designated Stage 9. The resulting volumes of waste and the cumulative and instantaneous stripping ratios by stages are shown in Table IX. Plate 23 shows the coal requirements, total waste and stripping ratio by years; Plate 24 shows the cumulative rom and in-situ coal tonnage and total waste by years; and Plate 25 shows the cumulative waste against cumulative tonnage of in-situ coal.

15. It should be noted that the total rom coal requirement over the whole life of the project is 348 million tons which is reached in Stage 7, ie the full development of the 600-ft pit is not required. This is in marked contrast to Report No 2 in which the 600-ft pit could not supply the full life of the power plant.

Pit Location

16. The outline of the coal deposit on 2,400-ft elevation can be effectively enclosed in an ellipse having major and minor axes of 7,000 ft and 3,500 ft in length respectively, and centred at 20,000 ft east and 79,000 ft north. The final slope of the 600-ft pit (end of Stage 8) has been projected outwards from this ellipse at an angle of 15^0 57'.

17. The outline of the coal deposit on 1,500-ft elevation was also plotted and enclosed in an ellipse with axes measuring 3,300 ft and 2,000 ft, and parallel to the axes of the 600-ft pit. The final slope of the 1,500-ft pit (end of Stage 9) was projected from this ellipse to surface at an angle of 15^0 57'.

18. Plate 26 shows the surface intercepts of Openpit No 1 as given in Report No 2 and as revised in this report for both the 600-ft and 1,500-ft pits. Plate 27 shows the surface intercepts of the stages of development, ie:-

Stage 1	-	minimum excavation required before start-up, ie-sufficient face length developed in coal and a coal stockpile of about 1 million tons
Stage 2	-	period between start-up and full output
Stage 3	-	the 600-ft pit has reached its full depth at the 2,400-ft elevation and is in the form of an inverted cone with its apex at the bottom of the access incline
Stages 4 to 7	-	arbitrary expansions of the pit for calculation purposes, derived by dividing the base ellipse into approximately equal annular areas
Stage 8	-	completion of 600-ft pit; only coal remnants left above the 2,400-ft elevation
Stage 9	-	expansion to 1,500-ft pit

19. The elliptical floor at the 2,400-ft elevation could not, of course, be maintained if this were the limiting depth as face collapse would be continuous. It is, therefore, a hypothetical concept. If a flat floor could be maintained then it would be possible to go deeper.

20. Table X gives the co-ordinates of the 600-ft pit limits at the end of the life of the 2,000-MW power plant in Stage 7 (assuming, of course, that coal is not required for other purposes), at the full expansion of the 600-ft pit (Stage 8) and also the 1,500-ft pit (Stage 9). Table XI gives the maximum vertical height of the slopes of the various pits. With regard to this table, it should be remembered that the "600-ft" depth applies to the main body of the coal itself and not to the overburden so that the maximum vertical height of slope including overburden would be 1,250 ft on the SW side. This is a formidable height in this type of rock.

21. The tables show clearly the greatly increased width and depth of Openpit No 2 (see Report No 3) as compared with Openpit No 1.

22. Table XII gives the proportion of coal zones by mining stages in terms of in-situ coal, ie Stage 1 is entirely Zone D, Stage 2 is 32% Zone D, etc. The mixing effect of mining in stages orientated in this manner is apparent.

<u>Incline Location</u>

23. As mentioned in paragraph 4 above, the access incline has been relocated slightly further to the east and its direction adjusted to allow for the change in centre of the proposed pit above the 2,400-ft elevation. This location is still, of course, subject to detailed geotechnical design.

24. The site adopted satisfies the following criteria:-

(i) minimum coal "sterilised" beneath the incline (this can be recovered on the retreat at close-down),

(ii) minimum excavation in the bad ground on the west side of Hat Creek,

(iii) minimum excavation to open up sufficient coal face before start-up.

24. The side slopes of the excavation for the access incline have been reduced from 25^0 as in Report No 2 to 22^0 as recommended in GA Report No 6. This slope angle of 22^0 only applies, however, to that part of the incline excavation which lies outside the conical pit.

25. The re-location of the access incline has resulted in an increase in plan length from 5,250 ft (Report No 2) to 6,000 ft and the incline will surface at an elevation of 2,800 ft. This increase in incline length, together with the change in excavation slope angle, will involve an additional volume of excavation outside the pit shape of about 12 million yd^3 (spread over Stages 1 to 3). Clearly any excavation within the pit confines will have to be made anyway and only the timing of this is affected. The extra cost of this excavation and the extra conveyor length are penalties to be paid for increased security and lower costs of slope stabilisation. The final location of the incline will obviously have to be given very careful consideration.

DEVELOPMENT PROGRAMME

26. Plate 28 shows prelimiary project construction schedules revised in accordance with:-

(i) four generators instead of three,

(ii) start-up deferred to April 1984, (since drafting this report this date has been changed to January, 1984, but this does not materially affect the results),

(iii) programme for next phase preliminary engineering, public hearings and then detailed design.

This programme could be shortened by various methods if necessary but it is controlled by the power plant start-up date. The items in the mine with the longest lead times will be the conveyor and stockpiling systems.

Environmental Considerations

27. In stronger rocks it might have been possible to have opened up initially to one side of the coal deposit and then, having worked it to the limit, commenced a restricted amount of back-filling on that side as the pit advances on the other side (as at Westfield). This method is not, of course, so economic as the stripping ratio is bound to be worse in the early years. However, it must be ruled out in the case of Openpit No 1 because of the low strength of most of the waste material, particularly the siltstones and claystones, which need confining to prevent instability. The presence of such dumps anywhere in the pit would be bound to increase the geotechnical problems which are already serious enough with the in-situ material. Therefore, the openpit will have to be maintained until mining ceases. Because of the enormous cost of replacing the dumped waste in the pit, back-filling could only come from other developments in the valley.

CHAPTER V

MINING OPERATIONS

GENERAL

1. This chapter describes the changes proposed in the mining operations since Report No 2 as a result of the investigations made in 1976. Where appropriate it explains the reasons for confirming the original concepts and estimates and examines the practical aspects of selective mining in line with the objectives listed in Chapter III.

PRODUCTION SCHEDULE AND MOBILE MINING EQUIPMENT REQUIREMENTS

2. Table XIII, Summary of Yearly/Stage and Cumulative Production, gives the amount of material to be mined and Table XIV, Schedule of Mobile Mining Equipment Requirements, lists the mobile equipment required to excavate and transport the different classes of material. In accordance with BCH's wishes, no further investigations have been made into the use of bucket-wheel excavators for Openpit No 1. The next stage of study work will involve further investigation into alternative mining systems including the use of bucket-wheel excavators.

DEVELOPMENT

3. Stages in the pit development are shown on Plate 27. These are generally the same as given in Report No 2 with some adjustments, and the apex of the cone will reach the 2,400-ft elevation, ie approximately 600 ft below the valley surface level at the end of Stage 3.

DIVERSION OF HAT CREEK

4. Monenco Consultants have reported on this in detail and the costs have been included under Infrastructure in Chapter VIII of this report. The diversion should be completed before any major mining work starts, otherwise temporary diversions will have to be made which would expose the initial workings to flood damage.

SURFICIALS

5. Further geological and geotechnical work has confirmed that the surficials are generally at elevations higher than the top of the conveyor incline and that no blasting will be necessary before removal by scrapers.

6. Many of the surficial materials, particularly those to the east of Hat Creek, are suitable for use in building retaining embankments for the waste dumps and, after filling and levelling the area north of the conveyor incline, it is planned to deliver these materials directly to site at the Houth Meadows and Medicine Creek waste dumps for this purpose. In 1989/90 the Medicine Creek surficial conveyor will be installed and surficials will all be dumped in that area except for material required to enlarge the retaining embankment at Houth Meadows. Some permeable materials will be needed for drainage beds in the waste dumps.

7. The quantity of surficials to be removed each year increases from 4 million to 14 million bank yd^3 over the mine life - the total quantity being 345 million bank yd^3. This is less than the quantity estimated in Report No 2, largely as a consequence of the lower coal output.

PIT WASTE

8. Pit waste will be removed by shovel and truck at the same time as the surficials as described in Report No 2. The total estimated quantity is 434 million bank yd^3 and this will be removed at a rate of from 2 million to 23 million bank yd^3 per year.

9. In Report No 6, Golder Associates have confirmed that blasting prior to excavation will not be necessary.

COAL, LOW-GRADE COAL AND SEGREGATED WASTE

10. The mining of these materials is dealt with in detail later in this chapter. Light blasting has been assumed to be necessary and they will then be loaded by shovel into trucks and thence transferred to conveyors. (Some material may be trucked out of the pit if it is suitable for construction purposes.)

11. During the period 1980-84, prior to the start up of the power station, 1.1 million tons of coal will be produced. The output will then climb from 2.7 million tpa in 1984/85 (one boiler generator unit operating) to 10.8 million tpa in 1987/88 (all four boiler units operating). It will then gradually fall to 8.4 million tpa in 2018/19 as the load factor of the station decreases, and drop sharply to 2.1 million tpa in 2021/22, the last year of operation.

12. An extra shovel has been added to the coal-loading equipment listed in Report No 2 to improve flexibility, to reduce the amount of time wasted in moving the shovels between benches and to allow for some loss of output due to selective mining.

BLASTING

13. The average uniaxial compressive strengths of the Hat Creek materials as given in GA Report No 6 are as follows:-

Material	Average Uniaxial Compressive Strength (lb/in^2)
Andesite	>3,000
Coal	1,250
Conglomerate	850
Sandstone	280
Claystone	75

This table shows all the material except the andesite to be class E, ie very low strength(3), and hence GA have given their opinion that blasting would not be required at Hat Creek. In view of this, no provision has been made for blasting of the pit waste although this was done in Report No 2. However, for the following reasons light blasting has been included in the cost of excavating the coal:-

(3) Rock Mechanics, SME Mining Engineers' Handbook

(i) selective blasting will be an essential aid to selective mining, ?

(ii) the cleat is poorly developed in the coal,

(iii) direct loading by 15-yd^3 shovel may be possible but problems would be caused when large pieces of coal were loaded on to the conveyors, and shovel wear and tear would increase.

14. In addition, hard calcareous or ironstone bands and boulders are expected to occur in the burn zone and in the Coldwater rocks and these will require blasting. To cover the extra blasting involved in this, the powder factor, although low at 0.3 lb/ton, is higher than would be necessary solely for light blasting of coal.

15. The drilling equipment proposed would be suitable for either coal or occurrences of harder beds and boulders. For the coal, 4-in holes at 10-ft centres would be a suitable drilling pattern.

16. Because blasting of the waste is not now required, there will be an overall saving compared with Report No 2 in explosives cost of $29 million over the life of the mine, which is equivalent to a saving of ¢8 per short ton in the overall production cost. In addition, there will be the saving in the ownership and operating cost of the drilling machines.

17. The explosive cost for blasting coal and other harder rock is estimated at $39 million over the life of the mine, ie ¢11 per short ton.

EQUIPMENT

18. To enable direct comparison of costs with Report No 2 to be made readily, the same types of equipment have been assumed. Detailed selection of the correct equipment will be optimised at a later stage and the size and type of equipment will be selected on the bases of suitability, cost and flexibility. The importance of the latter may mean smaller equipment than would be desirable solely on production grounds.

19. The type of truck selected is the rear-dump, but bottom-dump or even side-dump trucks would be considered.

20. Hydraulic shovels and 35-ton trucks have been added to the excavating and loading equipment for segregated waste and low-grade coal and their use is discussed in detail below when considering selective mining.

21. To ensure that hard bands in the surficials or pit waste can be removed efficiently, it has been assumed that one third of the bulldozers would be fitted with ripper attachments. To deal with trafficability problems, one fifth would be fitted with winches - thus initially half of the bulldozers would have an extra fitting.

22. The change in annual quantities of coal, surficials and waste can be seen by comparing Table III of Report No 2 with Table XIII.

23. The changes referred to above have affected the amount of mobile mining equipment as compared with Report No 2 as follows:-

(Comparisons are in equipment-years)

(i) Coal extraction and transport

Truck requirements have been reduced to 94%.

Shovel requirements have been increased by 64%.

(ii) Pit waste extraction and transport

No drilling machines have been included in this report.

Truck requirements have been reduced to 84%.

Shovel requirements have been reduced by 1%.

(iii) Surficials

Scraper requirements have been reduced to 57%.

24. The equipment requirements for segregated waste and low-grade coal have changed radically due to the importance of selective mining which became evident during the 1976 investigations.

ROADS

25. All roads on benches, particularly where the floor consists of claystones, should be surfaced with compacted ash - either bottom or fly ash -(or gravel) to ensure good trafficability. The thickness of this will vary from nil to several feet depending on the strata and detailed design of roads will be necessary at the next stage. For costing purposes an average thickness of 2 ft of ash has been taken in this report. It should be borne in mind that these roads are temporary only.

26. The ash will be taken from the ash conveyor to the pit by 35-ton trucks. Allowance has been made for a reasonable fleet of these in the costs but, in case of any temporary difficulty, they could easily be supplemented by the use of 150-ton trucks that are still serviceable but too old to be economically viable or sufficiently reliable for use in coal hauling.

27. It has been assumed that once a road has been formed it should be possible to push over the ash to the next cut by bulldozer, grader or wheeldozer. (This would, in any case, be done before loading if the bench consists of coal.) Additional material would be added to replace wastage and to allow for the increased circumference of successive cuts. Allowance has been made for this work in the capital and direct operating costs under the heading of ash handling. The recovery of ash may present problems where the bench includes a high proportion of clay and to cover this and general wastage a 50% factor has been assumed for ash utilisation.

SELECTIVE MINING

28. Chapter III explains the aspects of selective mining which are relevant to Hat Creek. This section looks into the practical problems of removing partings and describes the equipment recommended.

29. Table II summarises the examination of the core logs of 14 boreholes, each coal zone being intersected by seven boreholes. The aggregate intersections have then been used to compute the percentage of each zone and of the whole deposit comprising partings of various widths, ie:-

Thickness (ft)	% of Deposit
<1	2.6
1 to 2	0.7
2 to 3	0.7
3 to 5	1.6
5 to 10	3.3
> 10	8.2
	17.1

Waste above 20 ft thickness is included in pit waste. Low-grade coal represents 12% of the deposit on the basis of these holes. In this regard "low grade coal" is described as coal below 1 ft in thickness mixed with waste partings of similar thicknesses. (The actual grade cut offs for "waste", "low grade coal" and "coal" will be determined accurately later in the project. These values will vary throughout the life of the project as the value of a Btu changes and economic factors change.) Clearly, a more accurate parting count will become available when the computer system is operative.

30. The relative difficulty in the separation of partings increases as the partings become thinner; it also varies with the occurrence of the partings, ie whether at the bottom, middle or top of the bench, and also the orientation with respect to the face of the working bench.

31. If it is assumed that the dip of the partings is the same as that of the coal, then, as explained in Chapter II, a very approximate guide to their dip is:-

- 45% of the partings dip in the same direction as the pit slopes,
- 45% of the partings dip along the pit slopes, ie the strike of the partings is normal to the pit slope,
- 10% of the partings dip in the opposite direction to the pit slopes.

This is very satisfactory from a production point of view because it means that excavation will be relatively uncomplicated for all but 10% of the partings since it will be more difficult to separate partings dipping in the opposite direction to the face.

32. Separation of the partings normal to the pit slope can be arranged to be the same as working partings along the pit slope by working "on-end" to one side or the other. Plate 29 shows the two methods of mining. Both "face" and "on-end" mining are possible at Hat Creek due to the width of the benches which are wide enough to maintain the overall pit slope of $15^{0}\,57'$ (ie 120 ft minimum, see Plate 17). In most openpits this option is not available due to the narrowness of the benches and the steeper overall slope.

33. The various methods that are considered possible for a variety of thicknesses, positions and orientations are discussed below, and the bases for the allocation of special equipment for selective mining are then defined. In considering these methods the following assumptions have been made:-

(i) the difference in appearance between partings and coal can be recognised by operators, not for -2 m partings.

(ii) close bench drilling will be done throughout the mine life primarily for sampling and the position of partings will therefore be known in advance with some accuracy,

(iii) the coal will require blasting and the partings will not, Not so

(iv) selection will be done to minimise dilution of rom coal with waste but will not involve coal losses due to over selection.

Partings Above 5 ft in Thickness

34. It is expected that these partings could be removed by the large shovels in the usual course of mining the coal. This will be more difficult in some cases than others but when problems occur help will be available from smaller equipment, wheeldozers, etc. The large shovels would move on if excessive loss of production seemed likely.

Partings From 3 ft to 5 ft in Thickness

35. While the 15-yd^3 shovels could handle partings down to 3 ft in thickness, it is clear that this would reduce the production rate. For this reason a shovel was allocated solely for removal of segregated waste in Report No 2 and this has again been included. Plate 30 shows how various types of partings could be removed. As with the thicker partings, assistance for difficult situations could be given by the hydraulic shovels.

Partings From 3 ft to 1 ft in Thickness

36. These will all be removed by the 5-yd^3 hydraulic shovels assisted by the wheeldozers. General methods of taking out these partings are shown on Plate 31.

Partings Below 1 ft in Thickness

37. It is expected that these will not normally be removed but when they consist of bentonitic material it is possible that separation in the pit will be necessary to avoid excessive problems in materials handling both in the mine and in the power plant. Hydraulic shovels, wheeldozers and graders may all be used but it is probable that smaller equipment may be necessary, in particular front-end loaders might be preferable. Wide, shallow scraper buckets could be used. It is clear the productivity in these cases would be very low.

Low-Grade Coal

38. This can be loaded by the 15-yd^3 shovel when massive or the 5-yd^3 hydraulic shovel when in pockets. It has been assumed that only 11% of the 12% of the deposit representing low-grade coal can be separated, ie about 10% of the low-grade coal would be loaded as rom coal.

Equipment

39. A detailed study has been carried out on the productivity and costs of selective mining under conditions similar to those expected at Hat Creek with various sizes of equipment and different thicknesses, positions and orientations of partings.

40. This study indicated that the most efficient way of excavating partings greater than 10 ft in thickness would be by 15-yd^3 shovel and for sizes below this by a 5-yd^3 hydraulic excavator with interchangeable crowd shovel and backhoe attachments.

41. The production rate of the 15-yd^3 shovel is estimated to vary from 635 bank yd^3/hour for 10-ft thick inclined partings down to 347 bank yd^3/hour for 3-ft thick horizontal partings, and to drop to as low as 137 bank yd^3/hour if the horizontal parting is in mid-bench (the worst condition). The production rate of the 5-yd^3 hydraulic excavator is estimated to vary from 292 bank yd^3/hour for inclined partings 10-ft thick to 154 bank yd^3/hour for horizontal partings 1-ft thick. The comparative costs for this work (ownership and operating costs) for a 15-yd^3 shovel are estimated to vary from ¢41/bank yd^3 for inclined 10-ft thick partings to $1.92/bank yd^3 for 3-ft thick mid-point horizontal partings and for the 5-yd^3 excavator from ¢50/bank yd^3 for 10-ft thick inclined partings to ¢95/bank yd^3 for 1-ft thick horizontal partings.

42. Production figures were taken from this study and using the expected thicknesses and orientations described earlier in this chapter, taking a mean figure for the positions of the partings and estimating that 90% of the low-grade coal would be removed, additional equipment has been included under the heading "segregated waste" in Table XV.

43. For cost purposes the large shovel operating costs have been based on 4,000 hours per year and the hydraulic excavator costs on 2,500 hours per year, not 5,000 hours. These are the hours scheduled for these machines and full utilisation has not been assumed for the following reasons:-

(i) selection of waste partings and low-grade coal can be done more satisfactorily during daylight hours,

(ii) extra travelling between and along benches will be necessary.

44. On the basis of the information available, it is estimated that the equipment added in this report will be adequate to separate partings down to 1 ft in thickness but any additional selectivity on a regular basis would require additional equipment.

MINE DRAINAGE AND PUMPING

45. Report No 2 described the likely sources of water in the pit as:-

(i) drainage from surrounding areas,

(ii) seepage from the Hat Creek diversion dam,

(iii) seepage from surrounding strata,

(iv) natural precipitation.

The work done in 1976 enables these sources to be evaluated more accurately and the results of the GA tests are given in Report No 6.

46. Water from these sources can be dealt with as follows:-

(i) This can be minimised by a trench drainage system. An allowance for this work has been made in determining the numbers of bulldozers, wheel-dozers and graders shown in Table XIV

(ii) This is covered in the Monenco report and is minimised by having a seepage dam and pumping the water retained by it back up to the diversion canal.

(iii) In Table 5 of GA Report No 6 details are given of the proposed dewatering for mine slope stability. The cost for this would be $8,863,500 and this has been included in the costs of mobile mine equipment in Table XIX. The depreciation costs (pumps and ancillary equipment have been depreciated over ten years) and operating costs have been included. GA estimate that the total quantity of water pumped by these dewatering pumps would vary from 100 gpm to 250 gpm and they have estimated the total quantity of water to be handled by the mine and dewatering pumps to vary from 200 gpm at the start of the mine life to 500 gpm in the 1990s, decreasing to 250 gpm in 2010. The mine pumps must, therefore, be capable of handling up to 250 gpm from the pit on account of seepage from the coal and waste.

(iv) The maximum quantity of water to be pumped as a result of direct precipitation will vary from 150 gpm in the early years to 2,080 gpm when the pit reaches its maximum area. These figures are based on an annual precipitation equivalent to 12 in with half the precipitation taking place over a period of three months (the winter) and the remainder over the other nine months.

47. Table XIV shows the number of pumps scheduled throughout the mine life. The capacity will be more than enough to pump all the water expected from seepage and precipitation. Pumping capacity has been estimated on the basis shown below:-

Stage	Winter	Remainder of Year
1	1,000 gpm	500 gpm
2	1,500 gpm	500 gpm
Mid 4 onwards	2,000 gpm	500 gpm

48. The quantities to be pumped are thus not large in mining terms but careful planning for pumping will be necessary in the detailed design stage. Horizontal centrifugal pumps, which are suitable for pumping water containing suspended solids, would be used. The sump would be made at the low point of the pit bottom and near (but not immediately adjacent to) the base of the ramp. The pumps would be mounted on pontoons with one as standby and they would deliver water directly to the surface through pipes running up the ramp.

CHAPTER VI

SURFACE PLANT

INTRODUCTION

1. The surface layout on which the estimates used in this report are based shows considerable changes from Report No 2 for the following reasons.

2. The power plant site has now been confirmed as being near Harry Lake at an elevation of 4,500 ft, as against the mine site assumed in Report No 2. Therefore, the coal stockpile has been realigned to provide an "in-line" coal flow to the new location.

3. The latest recommendation by GA in Report No 6, Alternative B, was to create spoil dumps in both Houth Meadows and Medicine Creek, whereas Report No 2 was based on dumping most of the spoil in Houth Meadows. The spoil conveyor system has therefore been modified to deliver selectively to either dump and an additional provision has been made for the separate storage of low-grade coal and for its delivery to the power plant through a crushing unit, if required.

4. The proposal in Report No 2 to dispose of ash "dry" (ie conditioned by moistening) has been accepted and the ash disposal system from the selected power plant site has also been reappraised and a modified system included, utilising the Medicine Creek dump which is nearer to the new site.

5. The location of the power plant at 4,500-ft elevation and approximately 15,000 ft from the coal stockpile instead of at the mine site has increased the overall power requirements and conveyor costs, as has the delivery of spoil to Medicine Creek and low-grade coal to a separate stocking ground.

6. Since the various coal conveyor systems will have to operate prior to commissioning the power plant, temporary power lines from the grid system have been included in the estimates for supply of power to plant which could later be energised from the power plant direct.

COAL HANDLING

7. In Report No 2 the coal-handling system covered transport of the coal from the benches in the pit to the power plant. Included in this were whatever crushing, beneficiation and storage functions were required. The following sequence of operations was proposed:-

- coal trucks tip into hopper fitted with grizzly
- primary crushing
- conveyor transport out of pit
- secondary crushing
- coal preparation plant (if required)
- coal stockpile
- coal reclaimed from stockpile and transported to power station.

8. In this report the sequence has been modified and primary crushing is now assumed to be at a permanent location on the surface, because the type of crusher now being considered requires a large coal feed. The material in the pit is merely reduced to a size which passes through a 24-in grizzly, oversize being broken down by mechanical hammers. This has the advantage of reducing the number of primary crushers from one at each pit loading point to a single location on the surface which is fixed for the life of the mine in a more suitable working environment.

9. The system to be adopted finally can only be decided when more detailed information is available on the crushing and separation characteristics of the coal. If the arrangement now assumed does not prove feasible then the former concept of feeder-breakers in the pit would be used.

Truck/Conveyor Interface

10. While off-highway trucks are an expensive and labour-intensive method of moving material, they score on the grounds of flexibility. It is important, however, to keep their travelling distance as short as possible and to minimise uphill grades, particularly when travelling fully loaded. The ideal concept is, therefore, a truck/conveyor transfer point on each working bench, but this is not economically desirable and, in the suggested layout, loading points at three different levels are included. This will involve a certain amount of truck operation on the main incline to reach the nearest loading point. Each loading point would comprise a surge bunker fed through a grizzly with a feeder belt to deliver to the appropriate main incline conveyor. Any material above the loading stations would be transferred down the incline to the loading station. Material at elevations above the top of the conveyor incline would be delivered to the ground loading hoppers (shown on Plate 33) or direct to the dumps.

11. Any large lumps of oversize material could be broken up by a mechanical hammer to a size which would pass through the grid but where this could not easily be accomplished (eg some boulders), the lumps would be placed on one side by a grab and later removed by truck. As this operation would only be necessary infrequently, it is an acceptable exception to the policy of not taking loaded trucks up the main incline. Furthermore, such hard lumps are likely to be desirable construction material.

12. The correct design of the loading points is of extreme importance and can only be finalised after the type and size of truck has been decided.

13. The following criteria must be considered:-

(i) Sufficient hopper space must be available to accommodate a truck load of material and the design must match the selected vehicle.

(ii) Delivery to any of the three conveyors must be possible.

(iii) Three grizzlies would be necessary to enable coal, low-grade coal and waste to be handled separately. As the duties of the individual grizzlies could vary depending on which main machine conveyor was being used for which material, the grizzlies must be designed to take any material.

(iv) Arrangements must be included for breaking up oversize material on the grizzly and, where necessary, moving it to an adjacent grizzly for transport by the correct conveyor or loading it into road transport for removal from the pit.

(v) The loading point must be accessible from both sides of the pit and the conveyor gradient may have to be adjusted in places to allow the conveyors to pass underneath in tunnels to provide vehicle crossovers.

(vi) The merits of crushing in the pit before conveying, as against crushing at the surface, should be carefully considered after coal crushing tests have been carried out.

(vii) The feeders to the incline conveyors must not obstruct the incline roadways which must have unlimited height clearance.

CONVEYORS

Main Incline Conveyors

14. Report No 2 allowed for three parallel conveyors to take the coal and waste out of the pit. This basic proposal remains unchanged. The eventual length of each conveyor will now be about 6,000 ft which will be achieved when the pit reaches its maximum depth (2,400-ft base elevation). The basic design, therefore, must envisage periodic extension of the conveyor from its initial installed length of about 2,000 ft, corresponding to a floor at 2,700 ft in Stage 1, to 6,000 ft in Stage 3. Even when the maximum depth has been reached the conveyor need not extend beyond the lowest loading point which could be two benches up from the pit bottom. Trucks would then haul the bottom level production up to this point. This is acceptable as the amount of material removed from the bottom levels is comparatively small. It also has the advantage of protecting the tail end of the conveyor if the bottom level is flooded due to a flash storm. As the length and overall lift increases there will be a considerable increase in the power required to operate the belt. This is no particular problem as large and long belts are usually driven by several driving units, so that initially a reduced number would be fitted and these would be added to as the required driving power increased. The belt itself however would be designed for the maximum tension (eg steel-cord).

15. As the conveyors have to be capable of handling coal or waste, each conveyor has to be designed to take the maximum loading (ie pit waste) and, as there is a considerable difference in the tonnages of spoil and coal to be handled, there will be long periods on coal haulage where the belt is overpowered and the system will have to be designed to minimise the effects of this on the power supply system (eg power factor).

16. The availability of three conveyors means that facilities will always exist for handling coal, pit waste and surficials or low-grade coal on separate conveyors, but frequently one will be spare.

17. The spare will normally be available to guard against breakdowns and permit periodic maintenance and extension of the other belts. Alternatively, it can be used, when required, to handle waste or low-grade coal which requires segregation from the normal rom materials.

18. During development of the incline, the conveyors will be extended by a "leap-frog" action in which the two longest belts will normally be used for materials and the shortest kept spare. This will continue until it is possible to extend the shortest belt by, say, one bench length on the incline (about 600 ft). This will then be taken out of service and be extended past the other two. It will then be used for coal handling and one of the others will become the spare. This "leap-frog" process will continue as the pit deepens, until the conveyors reach their full working length.

19. Towards the end of the life of the pit, when the stripping ratios increase dramatically, it may be necessary to use two belts occasionally to handle pit waste.

Primary Crusher

20. As mentioned in paragraph 7, in Report No 2 it was proposed to install primary crushers in the pit. Deep troughed, 60-in wide belts are, however, quite capable of handling 24-in lumps so that, provided the lump size is kept below 24 in, as proposed in paragraph 8, it is possible to install the primary crusher outside the pit with the advantages previously mentioned. The object of the primary crusher would be to reduce the -24-in material to about -10 in.

21. A possible crusher for this duty is the Siebra crusher manufactured by Krupp, although a Bradford Breaker is another possibility. Both these machines have the advantage that while they crush the coal they do not crush large, hard stones but discharge them separately. They also reject, to a limited extent, large lumps of clay. They are the only types of primary crusher which have this classification characteristic as well as size-reduction capability. If the crushers are located near to the top of the main pit incline conveyors, then it is relatively easy to use a short conveyor to transfer the stone and clay rejects to the adjacent waste handling conveyor system.

22. A visit has been made to see a Siebra crusher at a brown coal mine in West Germany and a report on this visit has already been submitted to BCH. The machine was rejecting lumps of stone, hard clay and wood. Plastic clay is reported to ball up and be rejected but none was seen in the machine. It is likely that this was due, at least in part, to the visit being made towards the end of a hot, dry summer. A test has been proposed on a bulk sample of Hat Creek coal with interbedded claystone.

Secondary Crushers

23. Secondary crushers are required to reduce the -10-in material produced by the primary crushers to the size required by the power plant. This is not yet known but for costing purposes a hammer-mill has been assumed for reduction to 1.25-in size. It has been assumed that secondary crushing would take place after primary crushing, prior to delivery to the stockpile, so that only small coal would be stockpiled.

COAL STOCKPILE

24. Some form of coal stockpile is necessary for three basic reasons:-

- to provide a short term "surge" capacity so as to even out differences between the mine output and the power plant requirement,
- to guard against interruption to mine production and to ensure continuous supply to the power plant,
- to carry out a blending function so as to enable a more consistent product to be fed to the power plant.

25. In this instance, the blending function is important. For this purpose a storage of about 1 million tons of coal has been allowed for, although the quantity can only be finally settled when the quality range acceptable to the power plant is known. The subject of blending stockpiles has been discussed in Chapter III. Four piles fit the available space well and are sufficient for operational purposes.

26. One major problem in the storage of sub-bituminous coal is the possibility of spontaneous combustion. There are two basic ways of preventing this. One is to store only large lumps of coal and to maintain adequate air flow through the stack. This means that, although oxidisation will take place, the air flow will be sufficient to prevent a build-up of temperature to dangerous levels. The other approach which has been adopted is to stockpile small coal in a manner which ensures that, as far as possible, air is excluded from the stockpile. If dumping by stacker results in a tendency to heat up then compaction will have to be adopted. It is hoped to obtain more information from laboratory and field tests to assess this hazard.

27. The suggested layout of the coal stocking area is shown on Plate 32. A normal stock of 1 million tons has been proposed and the stocking area is designed around four piles, each about 2,500 ft long x 200 ft wide x 50 ft high. One such pile will contain about 0.3 million tons but there will only be the equivalent of three piles on the ground at one time. The normal position will be two complete piles standing idle, one part-pile being built and one part-pile being reclaimed.

28. The various stacking methods have already been referred to in Chapter III. It is proposed in this case to stack in windrows as this method best suits the layout, with recovery by bucket-wheel reclaimer moving centrally up the stockpile with the boom arcing over the whole width of the stockpile to give better blending.

29. In Report No 2 it was proposed to service these piles using three stackers and two reclaimers. It is now proposed to use one stacker and one reclaimer. The machines would be crawler-mounted for flexibility and would move between piles to service whichever is in use. A tripper is incorporated in each stacking conveyor to make the transfer of the spreader a simple operation. Crawler-mounted machines have been suggested rather than rail-mounted because of the greater flexibility of movement.

30. It is emphasised that there are a number of stocking, blending and reclaiming systems, each with its particular advantages. The system costed into this report is one of the simplest but the final choice will be influenced by the degree of quality variance allowed by the power station and obviously the more stringent the variance, the more expensive and complicated the blending machinery.

31. In case of a breakdown of the stocking system, spreader or reclaimer, one of the recovery conveyors is arranged to collect coal from the incoming cross conveyor and deliver to the outgoing cross conveyor direct, thereby cutting out the stocking arrangement temporarily while repairs are carried out, thus feeding direct to the power plant. It is not envisaged that this would occur often as the working conditions in the stockyard should be good, and normal maintenance should preclude breakdowns, but this provides an alternative feed to maintain the principle of duplication of all plant in the power plant coal supply system.

32. In addition to the normal storage, provision must be made for the 1 million tons of uncrushed coal produced during the initial development of the mine. This will have to be trucked as neither the main conveyors nor the stocking area equipment are likely to be operational at this stage. It is proposed to dump this immediately to the north of the permanent stocking area and, after compacting and sealing to prevent spontaneous combustion, the stock can either be gradually reclaimed or retained as an emergency stockpile. Reclamation would be by way of the "spare side" of the northern reclaiming belt and a combination of the bucket-wheel reclaimer and bulldozers. The lump size in this coal should be controlled in blasting, and breakdown would occur in stock. The re-crusher should be designed to handle any remaining lumps.

33. The estimates in this report exclude any coal beneficiation plant but in the suggested surface layout a possible site has been indicated for a plant should this be required. This is on the south side of the stocking ground and it would be fed from the crushing station. After beneficiation, treated coal could be stocked alongside the existing south-side stockyard reclamation conveyor, and recovered as and when required for blending with the main power plant supply. Rejects would be delivered to the spoil disposal conveyors.

RECRUSHING

34. After recovery from the stockpile and before delivery to the power station coal conveyors, a re-crusher has been included to reduce any frozen lumps or accretions to the size acceptable to the power plant. This unit could comprise a scalping screen and hammer mill.

TRUNK COAL CONVEYORS TO POWER PLANT

35. With the main stockpile at the mine, the only storage at the power plant will be that held in the bunkers in the boiler house. These are unlikely to contain more than, say, 12-hr requirements. Thus, even a short shutdown on the trunk conveyors could result in a loss of electrical output. To guard against this, all plant thoughout the coal transport and reclaiming system is either duplicated or provided with an alternative route. This system conforms with the recommendations of Integ/Ebasco.

SURFACE INTERCHANGE

36. At the surface, the incline conveyors deliver to an interchange station which enables materials coming from the pit to be routed appropriately. This routing function requires a sophisticated communication system and route selection facility located at a central control station. The pit waste passes direct to a conveyor feeding the Houth Meadows spoil dump. (In emergency it could be routed to the Medicine Creek dump.)

37. Surficial spoil or low-grade coal passes on to a feeder conveyor for delivery to the main conveyor for transport to the Medicine Creek area. Two ground hoppers at the surface allow spoil being transported by truck or scraper to be dumped on to the appropriate spoil disposal conveyor.

SPOIL DISPOSAL

38. The overlying and the segregated waste will come up one (or more) of the main incline conveyors. As recommended in GA Report No 6, there are to be two main dumping areas, one reserved for the clay and other difficult materials at Houth Meadows and the other for the more stable materials at Medicine Creek. However, no great harm would be done by sending small amounts of material to the "wrong" dump. This relaxation makes it possible to install only one conveyor to each dump, rather than two each of 100% capacity. Due to the long length of the conveyors this allows a substantial cost saving.

39. The pit spoil is delivered from the surface interchange on to a conveyor running on the north side of the Houth Meadows dump up to about the 3,000-ft level but eventually to the 3,750-ft level. From there the spoil is transferred to a movable conveyor on the embankment of the spoil dump for disposal by a travelling tripper and spreader. The movable conveyor is moved forward as required by a side-boom tractor.

40. Initially, the surficial waste will be removed by scrapers, discharging direct to the dump areas. When distances and the route become uneconomical for direct dumping, it is proposed to discharge into hoppers with feeder conveyors at the top of the incline. These would allow the material to be fed on to the normal waste conveyors for transport to the dump areas.

41. Spoil for the Medicine Creek dump is transported from the interchange to a main transfer point east of the pit area. This conveyor also transports the low-grade coal when required.

42. At the transfer point the spoil is fed on to a further conveyor feeding the Medicine Creek spoil area where it is disposed of through a conveyor feeding a movable conveyor on the spoil dump with tripper and spreader.

43. The details of the method of building the dumps will be worked out when the geotechnical recommendations and designs are completed but, in any case, the guidelines issued under the Coal Mines Regulations Act will be observed.

ASH DISPOSAL

44. The power plant ash is transported by conveyor from the power plant to the transfer point referred to in paragraph 41. The transfer point is so arranged that ash can be loaded direct from the ash conveyor into trucks if required for use on roads in the mine or spoil dumps, or delivered to the Medicine Creek conveyor for delivery to the spoil dump together with mine spoil. It is estimated that, at full output, about 350 tons/hour of ash will have to be disposed of and, of this, a maximum of 130 tons/hour will be required for surfacing in the mine and spoil dumps and for roads, etc. This will be taken by truck from the transfer point and the surplus will pass with the spoil to the Medicine Creek area.

LOW-GRADE COAL

45. When the Medicine Creek spoil conveyor is switched to transporting low-grade coal, the transfer point referred to earlier is arranged so that the low-grade coal can be diverted to a conveyor running parallel to the ash conveyor to the boiler house area.

46. Normally, the low-grade coal will be transferred from this conveyor to a conveyor/tripper/spreader arrangement for stockpiling. Should it be desired to feed the power plant direct with low-grade coal, the conveyor would deliver direct to a crusher and thence by a short conveyor to the boiler house. When recovering from the stockpile the same conveyor/crusher arrangement would feed the coal to the boiler house.

47. Recovery arrangements have not been included in the estimates as it has been assumed that during the life of the mine any low-grade coal required would come direct from the mine; and when the low-grade stocks are consumed after the mine is exhausted, the main stocking area plant would be free to be transferred.

CONVEYOR SCHEDULE

48. Table XV is a preliminary conveyor schedule. The total installed horsepower is 40,000. The complete conveyor system is shown diagrammatically on Plate 33.

GENERAL

49. The whole complex will be controlled from a main control centre situated near the main conveyor interchange station. This will be equipped with full

telemetry, monitoring and radio/telephone communication with all operating units and centres. All plant controls will incorporate automatic sequence starting and sequence interlocking.

50. The coal and spoil conveyors in fixed installations will have belt turnovers and deep troughing on the carrying side and troughing on the return side to minimise spillage. Belt cleaners will be fitted to all conveyors and all conveyor rollers will be designed to minimise the ingress of dirt.

51. All surface fixed conveyors will be totally enclosed in removable enclosures large enough to permit maintenance personnel to carry out inspections inside the enclosure, thus avoiding environmental problems due to dust and weather.

52. All conveyor systems will have emergency stop systems throughout their length with normal controlled stopping and pre-start warning systems. All conveyors will have adequate lighting throughout.

POWER SUPPLY

Surface Power Supply

53. The proposed electrical distribution system is shown on Plate 34. As in Report No 2, it is assumed that power would be supplied at 60 kV and distributed in the mine at 12.5 kV, and that until the power plant is commissioned the power for all conveyor drives would be supplied from the mine substation.

In-Pit Power Supply

54. Two main feeder cables installed down the incline form a 12.5-kV ring main with isolators to isolate any damaged section and allow the rest to remain operational. The shovels operating on electric power in the pit would be fed by five distribution cables from each side of the main incline feeders (ten in all) at 12.5 kV. The mine pumping units would be fed from the main feeder cable through suitable transformers.

55. In view of the length and number of pit and waste dump cables, a cable car would be necessary to handle new cables and to transfer cables without damage.

POWER CONSUMPTION

56. Power consumption will vary considerably throughout the life of the mine depending on the level of origin and ultimate disposal levels of the materials, together with the conveying length and material tonnages involved.

57. For estimating purposes, the power required for the average hourly loading, based on yearly tonnages, has been used, and the cost calculated on a flat rate of 20 mils per kWh. The estimated power costs are shown in Table XXII.

58. The estimated maximum demand of the mine covering coal conveying, stocking and delivery, spoil disposal, ash disposal and low-grade coal handling, is shown on a yearly basis on Plate 35. The figures have been estimated on the basis of 75% diversity factor and an average power factor of 0.75. The maximum demand is estimated to be about 35 MVA.

59. In calculating the maximum demand, the low-grade coal handling system has been excluded as it is assumed that when this is operating the coal conveyors will be idle and the Medicine Creek spoil disposal plant will only be handling ash.

60. The ash conveyor from the power station will only require power at start-up, being regenerative when loaded, and as it is continuously running the effect on the power loading has been neglected.

CHAPTER VII

WASTE AND ASH DISPOSAL

1. The volumes of materials to be transported and dumped are shown in Table XVI. In summary, 377 million yd^3 of surficials, 624 million yd^3 of pit and segregated waste and 97 million yd^3 of conditioned ash, totalling 1,098 million yd^3, will have to be dumped. Report No 8 deals more fully with the disposal of this material and reclamation.

2. In Report No 2, all waste material from the 600-ft pit was to be dumped in the Houth Meadows area. However, subsequent tests by GA, described in Report No 6, showed the pit waste to be very weak material and, because of this and the presence of old flow slide debris at Houth Meadows (GA Interim Report No 4), the quantity capable of being stored in Houth Meadows has been reduced. Two alternative dumping schemes were given by GA but alternative A was not recommended for feasibility purposes because of the risk that the waste would not stand at the gradient necessary. The selected alternative, B, has thus been followed in this report which uses Houth Meadows and Medicine Creek as the dumping areas. These are shown on Plate 36.

3. Table XVII shows the areas available and the volumes of space available to various elevations. These are shown graphically on Plate 37. There are also 12 million yd^3 of surficials to be dumped in the valley at the north end of the inclined ramp to form a flat and regular area for mine surface installations and the coal stockpile.

4. Building of the retaining embankments and the formation of dumps is dealt with in the Reclamation Report (No 8), but basically: -

(i) Pit and segregated waste and some weak surficials will be dumped in the Houth Meadows area.

(ii) Surficials and some pit waste will be dumped in Medicine Creek. However, the split between the two dumps will be governed by practical mining as well as geotechnical considerations.

(iii) Any power plant ash which cannot be usefully used as a construction or road-making material will be placed within the dumps.

(iv) The retaining embankments will be made from glacial sand and gravel and compacted glacial till, with the possible addition of some power plant ash.

(v) The bases of the embankments will be formed by direct dumping by scrapers with subsequent compaction as required.

(vi) The embankments will be carefully designed in accordance with sound soil mechanics principles.

(vii) The maximum slope of the embankment face will be 1 in 2 to comply with the guidelines provided by the Department of Mines under section 8 of the Coal Mines Regulation Act of 1969.

CHAPTER VIII

INFRASTRUCTURE AND CIVIL WORKS

HAT CREEK DIVERSION

Introduction

1. The present course of Hat Creek is through the centre of the proposed mining area. Therefore, as already explained in Report No 2, it is necessary to provide an alternative route for the water at present handled by the Creek. This route must remain operational throughout the working life of the mine. After this period the stream can be returned to its original course and the pit allowed to fill up, provided that, at the same time, some water is passed round the pit to maintain a minimum flow in the lower reaches of the Creek. This status must be maintained until the pit water has reached the lowest point of the rim (near the top of the incline) and is overflowing into the old stream bed. This all assumes that the pit void will not be used for disposal of waste from Openpit No 2, in which case the diversion must be maintained until tipping is completed. The whole problem associated with the reinstatement of the stream is dealt with in detail in the Reclamation Report (No 8).

Original Proposals

2. Report No 2 proposed constructing a reservoir of about 850 acre-ft capacity to the south of the pit and pumping the water round the pit by means of 100 ft^3/sec piped diversion. More recent information has shown that this proposal has insufficient capacity to cope with the duration and rate of the maximum anticipated spring flood. This matter was considered exhaustively in the Monenco report "Hat Creek Diversion Study" produced in November, 1976. The salient points of this are discussed below.

Monenco Report

3. This report was able to draw on additional information concerning the magnitude and duration of the maximum flows in the Creek. These are caused by a combination of snow melt and spring rains. From an analysis of this information they deduced that a pumped diversion of 100 ft^3/sec capacity would require some 10,000 acre-ft of reservoir storage to cope with the 100-year flood condition.

4. They considered a number of alternatives using tunnels, canals, storage or a combination of these, but their preferred solution is a canal with a maximum capacity of 800 ft^3/sec. This matches the 100-year flood. No reservoir storage is necessary, although a small dam is proposed at the canal inlet. This canal would follow the 3,200-ft contour round the east side of the pit. This contour is cut by the pit during or about year 2005 and this requires a section of the canal to be replaced by a tunnel at that stage. It is, however, financially advantageous to postpone this work for some 25 to 30 years rather than making this section in tunnel originally, especially as the final pit outline may well change from that originally proposed. The exact length and location of the diversion required can only be determined as the pit approaches its planned limit. Experience may show that it is possible to steepen pit slopes locally and it may eliminate the need to provide alternative diversion arrangements.

5. The Monenco capital cost estimate for their preferred scheme is \$6.8 million. This, however, includes a charge for some 300,000 yd^3 of fill material which could be supplied from material excavated during the development stage of

the pit. As a cost for disposal of this material has already been allowed in the pit economics, it is reasonable to treat this as "free issue material" so far as the canal scheme is concerned. The Monenco proposal is still considerably more expensive than the $1.75 million allowed in Report No 2, although this direct comparison is not completely fair since the Monenco estimate includes items such as access roads and the diversion of minor streams which were excluded from the diversion cost given in Report No 2 as they were included elsewhere in the total budget. It also includes interest during construction which was dealt with separately in Report No 2.

Present Proposals

6. The Monenco report was based on the original pit development given in Report No 2. This has now been modified as has already been described in Chapter IV. One result of this is that the 3,200-ft contour is not now cut until year 2013. The relationship of this new pit outline to the course of the canal is shown on Plate 38. Some reduction can be made to the Monenco cost estimate as this includes interest on capital during construction while this report calculates interest throughout the whole life of the project. The Monenco estimate also allows for the cost of fill material, which will in fact be available as "free issue" material from the development stage of the mine. These factors reduce the overall cost to $5.9 million while still using the design philosophy and the unit costs given in the Monenco report.

7. There are two possible modifications to the Monenco report which should be examined when the final engineering design of the pit is being considered. Their report is based on a diversion system which can handle a 100-year flood. Anything in excess of this is allowed to spill over into the pit. It would be possible to design the system to handle only a 35-year flood. This would mean accepting the probability of one spill-over during the operating life of the mine, with its attendant damage and loss of production. The 35-year flood rate is about 660 ft^3/sec (compared with the 100-year flow of 800 ft^3/sec) and the capital saving is only about $0.5 million. Since the cost of a spill-over in the early years of production could exceed $1 million, this does not look particularly attractive. However, it cannot be ruled out at this stage.

8. The other alternative, which was, in fact, mentioned in the Monenco report, is to run the canal at a slightly higher elevation. Raising it to 3,250 ft would keep it clear of the revised pit outline, thus avoiding the need for any diversion, unless it were decided to extend the pit by working deeper than 600 ft. Raising the canal in this way would involve moving the head dam slightly higher up the valley and this could give problems collecting the water from Anderson Creek.

9. However, these are both points of detail and they are unlikely to make a noticeable change to the cost of this aspect of the project.

SURFACE DRAINAGE

10. The Monenco report proposed draining the area around Finney and Alleece lakes via a 30-ft^3/sec ditch to Anderson Creek for diversion into the canal. Drainage from the remaining area in the valley bottom below the diversion ditch and up to the canal on the East Valley slope would be retained behind an earth fill dam located at the pit rim from where it would be pumped into the canal. The costs for this work were included in their estimate for the main creek diversion. All these proposals have been incorporated in this report.

ROADS

11. In Report No 2 it was proposed to divert the road running north from Upper Hat Creek round the eastern perimeter of the pit. It is now proposed to modify the route slightly so that as far as possible it is situated alongside the diversion canal, as was allowed for in the Monenco report. This produces a useful cost saving and also facilitates routine monitoring and maintenance of the canal.

12. It will also be necessary, as described in Report No 2, to upgrade the main access road to cope with the additional weight and number of vehicles requiring access to the mine. A sum of $1,278,000, spread over Stages 1 to 3, has been allowed to cover this, together with the other roadworks needed within the mining area. The main service roads will be paved but some of the minor ones need only be gravel. It is anticipated that much of the base material can be found from material excavated from the pit. In addition, from Stage 2 onwards there will be supplies of pulverised fuel ash available from the power plant and this is excellent road-making material.

SURFACE MINE BUILDINGS

13. No significant changes are proposed to the surface mine buildings allowed for in Report No 2. There are some cost changes, principally due to the use of 1976 rather than 1975 as a cost basis. The schedule of buildings required initially and their costs are shown in Table XVIII.

SERVICES

14. Services such as power and water supply, and sewage disposal are itemised in Table XXI. This section also includes two graders for surface road maintenance, a number of pick-up trucks to cover all the miscellaneous transport requirements which are not directly associated with mine production, and buses to provide employee transport.

HOUSING

15. Here again the position is not greatly different from that proposed in Report No 2, although the labour requirements have been changed slightly. During the pre-production stage a large part of the labour on site will be employed by outside contractors and construction firms. This transient labour will be housed in trailer camps or pre-fabricated camps provided by the contractors at or near the mine site.

16. It is likely, however, that the permanent mine labour force will prefer to establish roots in one of the existing local communities such as Ashcroft, Cache Creek, Clinton or Pavillion, where there are established schools and recreational facilities. These townsites all lie between 15 and 30 miles from the mine site. Some of the employees will prefer to use their own transport but sufficient buses have been included to provide a service to and from each township to correspond with the regular shift times.

17. The capital costs of the initial mine site camps and of the permanent housing in local communities are shown in Table XVIII and are also included in the schedule of infrastructure (Table XXI). They are not, however, included in the calculation of the production cost of the coal as it is assumed that the finance will not have to be provided by BCH. The temporary accommodation should be the responsibility of the various construction firms. There is not a sufficient pool of housing in the local townsites to accommodate the permanent mine staff, so new houses will have to be built irrespective of their preferred location. It is assumed, however, that these will be financed by the occupants under normal loan arrangements.

CHAPTER IX

ECONOMICS

BASIS

1. This report, being an up-date of Report No 2, adopts the same format with some minor alterations, and again covers mining Openpit No 1 down to the 2,400-ft elevation (600-ft pit) with projections to the 1,500-ft elevation (1,500-ft pit). In particular, coal handling, ash handling and reclamation costs (derived from Report No 8) have been segregated.

Basic Financial Data

2. Appendix "D" summarises the basic financial data and gives comparisons with the former version, Appendix "D" of Report No 2. Plate 39 shows the corporate overhead rates. The base date used for all economic factors is October, 1976, and classes of cost have been segregated to facilitate indexing. It is for consideration whether the cash flow streams could be stored in the BCH computer and programmed for future rapid up-dating.

3. As regards inflation calculations, it has again been assumed that capital goods and electrical energy will have the same inflation rates as labour and materials.

4. Again, the following further 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 an "historic cost" basis, hence depreciation and interest can be directly inflated.

5. The changes in the basic financial data since issuing Report No 2 can be summarised as follows:-

(i) base date changed,

(ii) inflation rate - similar to before,

(iii) 10% and 15% discount rates used instead of 15%,

(iv) power costs doubled,

(v) provincial sales tax of 7% on all purchased goods added.

Economic Factors

6. The main changes in the economic factors are approximately as follows:

Capital goods	-	+ up to 9% (based on new quotes)
Other materials	-	+ up to 15%
Labour	-	+ 8% (based on anti-inflation legislation)
Management	-	+ 8% (based on anti-inflation legislation)

CAPITAL COSTS

7. The following tables deal with the capital costs of plant, equipment and services:-

Table XIX - Schedule of Mobile Mining Equipment - Initial and Replacement Costs

Table XX - Schedule of Fixed Installations

Table XXI - Schedule of Infrastructure.

DIRECT OPERATING COSTS

8. The following tables deal with direct operating costs:-

Table XXII - Summary of Power Costs

Table XXIII - Summary of Labour and Payroll Costs

Table XXIV - Summary of Materials and Fuel Cost Excluding Electric Power

Table XXV - Summary of Direct Operating Costs.

TOTAL INVESTMENT AND CAPITAL CHARGES

9. The following tables deal with these items:-

Table XXVI - Summary of Depreciation

Table XXVII - Capital Investment, Interest During Construction, Interest and Insurance - Mine

Table XXVIII - Capital Investment, Interest and Insurance for Coal and Ash Handling.

PRODUCTION COSTS

600-ft Pit (2,000-MW Power Plant)

10. Table XXIX, Run-of-Mine Production Costs for the Mine, totals the direct operating cost, capital charges and royalty items using accounting methods and Table XXX does the same for the coal and ash handling costs (not including royalty) and also shows the totals for mine and coal and ash handling. Plate 40 shows how the production cost varies over the life of the mine, ie from about \$7/ton to \$20/ton.

11. The pronounced reduction in cost in Stage 3 is due to the effect of the Medicine Creek conveyor being brought in to convey the surficials from the pit in the year 1991/92. These costs do not include the cost of reclamation which is dealt with in Report No 8.

12. Report No 2 estimated the coal and ash handling costs to be 80¢ per ton. These costs have now been isolated and are considerably higher than before. In addition to the sales tax costs and inflation, the increase is mainly due to the proposed siting of the power plant near Harry Lake compared with the previously assumed position in the valley.

1,500-ft Pit

13. The instantaneous stripping ratio at the probable limit of the pit down to the 1,500-ft elevation has been recalculated at 29.2 bank yd^3/short ton rom. Extrapolation after Stage 7, which is the extent of the pit required for the life of the 2,000-MW power plant, and this ratio of 29.2 results in a production cost of about $38/ton, thus the additional coal between the pit at 600-ft elevation and the 1,500-ft pit limits could be mined at a production cost of between $15 and $38/ton.

DISCOUNTED CASH FLOW

14. Tables XXXI and XXXII show the cash flow of the expenses and the calculation of constant selling prices which would yield internal rates of return of 10%, alternatively 15% (specified by BCH - see Appendix "D"). The cash flow includes a capital and direct operating cost element, together with insurance and royalty. This has been done separately for the mine (Table XXXI) and for coal handling and the ash handling (Table XXXII). The results are as follows:

	Coal Prices, $/ton	
	10% discount rate	15% discount rate
Mine	6.93	7.99
Coal handling	1.36	1.66
Ash handling	0.10	0.12
Total	8.39	9.77

The totals are shown on Plate 40. If this uniform selling price at the mine (ie $7.99/ton at 15% discount rate) is compared with the corresponding price derived in Report No 2 (ie $5.55/ton) there is a difference of $2.44. This can be attributed to the following factors:-

		$/ton
(i)	The addition of 7% sales tax and inflation of 10%	1.00
(ii)	The effect on the dcf of spreading coal output over a longer period, ie maximum yearly coal output now 10.8 million tons in place of 13 million to 14 million tons	0.87
(iii)	Power costs increase from 10 to 20 mils/kWh	0.15
(iv)	Additional conveyor for surficials transport to Medicine Creek	0.42
	Total	2.44

There is also an increase of $0.98/ton in the coal and ash handling costs (ie $1.78/ton compared with 80¢/ton used in Report No 2) due to:-

		$/ton
(i)	Sales tax of 7% and inflation of 10%	0.15
(ii)	Conveyor from mine mouth to power plant at Harry Lake	0.83
	Total	0.98

15. It should be noted that although the differences are explained exactly above, there are numerous other minor changes in costs or prices. Also, the "ton" in this report is 8% higher in calorific value than the "ton" in Report No 2.

CONFIDENCE LIMITS OF ESTIMATED SELLING PRICE

16. As in Report No 2, a pessimistic view has been taken in all areas of uncertainty and the estimated production costs and selling prices can be regarded as a "maximum". Using an uncertainty of $\pm$10% about the mean, the mean and minimum figures can then be calculated. (The uncertainty is likely to be greater in areas involving coal quality.) The resultant mine mouth selling prices and ranges for 10% and 15% discount rates are given below:-

Coal Prices, $/ton

	10% discount rate	15% discount rate
Uniform selling price including coal handling and ash disposal	8.39	9.77
Coal handling and ash disposal costs	1.46	1.78
Maximum mine mouth selling price	6.93	7.99
Mean mine mouth selling price	6.24	7.19
Minimum mine mouth selling price	5.62	6.47

17. With the availability of increased information on the deposit, areas of uncertainty become less and it would now seem that the most likely areas of cost saving would be in optimisation of equipment selection and utilisation, and in less but more efficient blasting.

LIFE OF OPENPIT NO 1

18. With the change in the operating regime for the 2,000-MW power plant, the quantity of coal down to the 600-ft level is more than adequate for the 35 years' life needed.

PRODUCTION COST (INFLATED)

19. Table XXXIII shows the production costs for the mine, for coal handling and for ash handling. These have been calculated on the same basis as Tables XXIX and XXX but the cost elements have been inflated according to the rules given in Appendix "D". Royalty costs have not been inflated.

DISCOUNTED CASH FLOW (INFLATED)

20. Table XXXIV is a repetition of Tables XXXI and XXXII inflated in the same way as Table XXXIII.

21. The resultant uniform selling prices are as follows:-

Coal Prices, $/ton

	10% discount rate	15% discount rate
Mine	17.28	16.57
Coal handling	3.32	3.44
Ash handling	0.27	0.26
Total	20.87	20.27

22. As in Report No 2, these costs are more than twice the uninflated figures of $8.39 (70¢/$10^6$ Btu) and $9.77 (82¢/$10^6$ Btu) for the 10% and 15% discount rates respectively.

23. It will be noted that in the case of the uninflated costs the 15% discount price is higher than the 10% discount price whilst in the case of the inflated costs this is reversed. This is a function of the shape of the cash flow curve. The discounted coal production is not, of course, inflated.

OPPORTUNITY VALUE FOR
HAT CREEK COAL

24. Plate 45 of Report No 2 remains valid and on the basis of international oil prices at, say, $13/bbl, the opportunity value of Hat Creek coal would be about $22/ton.

BREAK-EVEN STRIPPING RATIO

25. Plate 46 of Report No 2 remains valid.

CHAPTER X

SUMMARY AND RECOMMENDATIONS

GENERAL

1. This report is an up-date of Report No 2, Preliminary Report on Hat Creek Openpit No 1, dated March, 1976. It takes into account further information obtained during 1976 from the following sources:-

(i) geological field work,

(ii) geotechnical field work,

(iii) reports from consultants working on other aspects of the project, eg Integ/Ebasco, Monenco, Acres, etc.

The economic calculations have also been up-dated to a base date of October, 1976, and modified in accordance with BCH's requirements. All the information collected in 1976 has not yet been received in final form nor processed to produce all the conclusions which can be obtained from it, but nevertheless the validity of the conceptual Openpit No 1 has been confirmed and indeed reinforced.

GEOLOGICAL

2. The main geological changes resulting from the 1976 in-fill (closer-spaced) drilling programme can be summarised as follows:-

(i) better knowledge of the structure of the deposit,

(ii) identification of four coal zones and more quality data,

(iii) increase in reserves of coal which, coupled with a reduction in the coal demand, now means that Openpit No 1 can support a 2,000-MW (net) power plant for the full 35-year life of the generators.

GEOTECHNICAL

3. The main changes resulting from the 1976 geotechnical field work carried out by Golder Associates (see Report No 6) are as follows: 1979 Report

(i) Confirmation that the general pit slope angle adopted (15° 57') cannot be increased with the present state of knowledge. Revised 16°-20°-25° circle

(ii) More knowledge of the strength and characteristics of all the rock materials, in particular confirmation of the low strength of the claystones. This has resulted in changes in the designs and locations of the waste dumps.

(iii) Better knowledge of the type and properties of the coal interbeds, coupled with the conclusion that the bentonitic materials are unlikely to cause such serious handling problems as was feared. Clay cut Trench C

(iv) The flow of groundwater is not expected to be a problem as such, although pore water pressure will seriously influence stability. No depressurisation by dewatering.

(v) The poor and unstable nature of the ground on the west side of the incline as previously sited has resulted in it being moved somewhat to the east and it is expected that groundwater pumping will be required to maintain stability.

COAL QUALITY

4. A considerable amount of coal quality data was collected in 1976 and has been incorporated in the computerised drillhole data base. Some useful processing has been undertaken but more accurate processing awaits the implementation of the full computer system. Preliminary results have given data by coal zones and mining stages. The shape of the mining stages results in mixing of the coal and reduces the variation in quality.

5. For the sake of uniformity, the assumed heating value of the coal has been increased from 5,500 Btu/lb to 5,950 Btu/lb as proposed by Integ/Ebasco. This is also justified by the assumption that more interbedded waste could be removed by selective mining, and provision has been made for additional equipment for this purpose.

6. A considerable amount of work needs to be done in this area to reconcile the frequently conflicting desiderata of the power plant and the mine and to settle remaining uncertainties in the area of coal preparation. completed

7. The decision to site the coal stockpile at the mine has resulted in more attention being given to this area. A blending stockpile is recommended to further reduce the variation in coal quality. completed

COAL HANDLING AND ASH DISPOSAL

8. The confirmation of the Harry Lake site has increased the coal handling costs as duplicate conveyors to the power plant have been added. The confirmation that the ash will be disposed of dry by conveyor coupled with the use of the Medicine Creek disposal site has enabled the ash disposal system to be modified to advantage.

WASTE DISPOSAL

9. The weak claystones will be disposed of in the Houth Meadows area, behind an embankment constructed of more stable material. Surficials will be disposed of in the Medicine Creek area. Both dumps will conform to the guidelines issued by the Provincial Government and will be designed in accordance with sound soil mechanics principles. Provision will be made for drainage and the surfaces will be contoured and prepared so as to encourage re-vegetation.

INFRASTRUCTURE AND CIVIL WORKS

10. The buildings and surface works are unchanged as compared with Report No 2. The siting of the Hat Creek diversion channel has been adjusted in the light of the Monenco recommendations and the modified shape of the pit.

ENVIRONMENTAL CONSIDERATIONS

11. No back-filling in the pit will be possible until mining operations cease and after that it would be prohibitively expensive. Hence, efforts have been directed to minimise the environmental impact of the pit and the waste dumps and reclamation procedures have been developed, see Report No 8.

ECONOMICS

12. All the economic tables have had to be re-worked to incorporate the consequences of the modifications to the conceptual Openpit No 1 (600-ft pit) discussed above. The results as compared with Report No 2 are summarised as follows:-

	Report No 2	Report No 9	
Capital investment to start-up ($\$10^6$)	134	209	
Uniform selling price (\$/ton)			
10% discount	5.63	8.39	7.25
15% discount	6.35	9.77	
Coal calorific value (Btu/lb) As received 24% H_2O	5,500	5,950	5885

COMPARISON OF OPENPITS NO 1 AND 2

13. Table XXXV is a revised summary of the comparison between Openpits No 1 and 2 and the revised Openpit No 1. The details and costs in this report and in the data for revised pit No 1 are based on the information obtained from work carried out in 1976 and hence are more accurate. Data for the Openpit No 2 has not been up-dated since Openpit No 1 was selected as the preferred pit. This preference is obviously still valid as the capital investment to start up and the uniform selling price shown for Openpit No 2 in Table XXXV would certainly increase due to inflation, sales tax and re-siting of the power plant in the same way as they have done for Openpit No 1.

RECOMMENDATIONS

Geological

14. It is recommended that the in-fill drilling programme should be completed to a maximum of 500-ft spacing between boreholes - say 95,000 ft of drilling at a cost of approximately \$3 million. ✓ completed

Geotechnical

15. GA Report No 6 gave detailed recommendations for the continuation of the geotechnical investigations, particularly as regards groundwater, slope stability, waste dump embankments and material testing. This information is required for detailed design of slopes, embankments, etc. It will be recalled that only one year's field work has been carried out of the two-year programme originally envisaged. complete

Coal Quality

16. Future work should include the following:-

(i) processing quality data already obtained,

(ii) incorporation of new data,

(iii) further liaison with the power plant consultants, ✓

(iv) completion of sample mine, ✓

(v) coal testing in all aspects, — need milling tests. -

(vi) more accurate appraisal of blending parameters. How?

Computer Systems Development

17. In view of the large volumes of data which have to be recorded and assimilated and the function of the computer systems as links between all sections of the power project, the continuing development of the computer systems is considered essential. In the first instance, these systems will provide design information and later production control information.

Detailed Design

18. The detailed design of the mine facilities can only follow the completion of the definitive concepts, although useful preliminary work can be done. Important areas are the conveyor system, the stockpiling system, the surface civil engineering works and the electrical distribution system.

APPENDIX "A"

LIST OF REPORTS PREPARED BY PD-NCB CONSULTANTS IN ASSOCIATION WITH WRIGHT ENGINEERS AND GOLDER ASSOCIATES

No	
1	Interim Report on Geological and Geotechnical Exploration at Hat Creek - November 1975
2	Preliminary Report on Hat Creek Openpit No 1 - March 1976 (incorporates Report No 1)
3	Preliminary Report on Hat Creek Openpit No 2 - March 1976
4	Hat Creek Geotechnical Study Interim Conclusions - October 1976
5	BC Hydro Hat Creek Project. Coal Resources Optimisation and Production Scheduling (Crops) System Phase II - General Methods of Approach (Interim) - December 1976
6	Hat Creek Geotechnical Study (Final) - March 1977 Revised Dec 78.
7	Hat Creek Power Project. Combined Pit Operation Study for 5,000-MW Power Plant - January 1977
8	Reclamation Study Related to Mining of Hat Creek Openpit No 1 - March 1977
9	Revised Report on Hat Creek Openpit No 1 (this report)
10	Description of Computer System (five volumes)

APPENDIX "B"

LIST OF DOCUMENTS AND DRAWINGS RECEIVED BY PD-NCB FROM 8TH JUNE, 1976, TO 5TH JANUARY, 1977

REPORTS

"Proposed Hat Creek Development Transportation Study", Swan Wooster Engineering Company Limited, June 1976

"Preliminary Considerations of Sampling Plan Design for the Hat Creek Coal Deposit", A.J. Sinclair, June 1976.

"Inter and Intra Laboratory Reproducibility Hat Creek Coal Analyses", A.J. Sinclair, July 1976.

"Interim Report on Dry Proximate Analyses of Test Holes 135 and 136", A.J. Sinclair, September 1976.

"Hat Creek Project - Detailed Site Selection Study", final draft, Integ-Ebasco, September 1976.

"Report on Hat Creek Field Work in Regards to Soils Suitable for Reclamation", J.T. Forster.

"Hat Creek Diversion Study" - Draft, Monenco Consultants Pacific Limited, November 1976.

RECORD OF COMPLETED DRILL HOLES

DDH 76-119 to DDH 76-208

BAH 76-1 to BAH 76-15

P76-1 to P76-4, P76-6, P76-7, P76-9 to P76-18, P76-18A, P76-19 to P76-21, P76-21A, P76-22 to P76-24, P76-26 to P76-28, P76-28A, P76-29, P76-30.

GEOPHYSICAL LOGS OF BOREHOLES

76-105, 76-107, 76-108, 76-110, 76-111, 76-112, 76-113, 76-115, 76-116, 76-117, 76-118, 76-119, 76-120, 76-121, 76-122, 76-123, 76-124, 76-125, 76-126, 76-127, 76-128, 76-129, 76-130, 76-132, 76-133, 76-134, 76-135, 76-136, 76-137, 76-138, 76-139, 76-141 to 76-208.

WRITTEN LOGS OF BOREHOLES

76-117 to 76-208, 76-814, 76-817.

GRAPHICAL GEOLOGICAL LOGS SCALE 1 IN TO 40 FT OF BOREHOLES

76-111 to 76-160, 76-163, 76-165, 76-167, 76-170, 76-173, 76-178, 76-180, 76-182, 76-184 to 76-186, 76-189, 76-190, 76-194, 76-195, 76-814, 76-817.

GRAPHICAL GEOLOGICAL LOGS SCALE 1 IN TO 20 FT OF BOREHOLES

76-120 to 76-208, 76-814, 76-817.

MISCELLANEOUS PLANS AND SECTIONS

No 1 deposit, location map, 1 in = 1,000 ft, showing locations of drill holes up to No 76-133.

Topographic map, No 1 deposit area with drill hole locations to No 76-129. Scale 1 in = 400 ft; contour interval = 10 ft.

Geological sections, reduced (1 in = 400 ft) showing subdivision of strata into A, B, C and D zones. Following sections:-

77,000 N	80,000 N
78,000 N	80,500 N
78,500 N	81,500 N
79,000 N	

Sketch showing relationship between survey co-ordinates and land divisions with covering letter to R.M. Woodley, BC Hydro.

No 1 deposit, fault model plan 1 in = 1,000 ft.

Sections 1 in = 200 ft scale showing coal zones A to D, DDHs to 30th September, 1976, and faulting 78,000 N, 79,000 N, 21,000 E, 22,000 E.

Hat Creek No 1 Area - 1 Mylar Topo. Map 1 in = 1,000 ft.

DCA No 1 deposit plan showing drilling progress, scale 1 in = 1,000 ft.

Preliminary plan showing contours of bedrock surface, scale 1 in = 400 ft.

Preliminary draft geological sections of No 1 coal deposit, 1 in = 400 ft, 23rd October, to 4th November, 1976.

E-W Sections	N-S Sections
76,000 N	17,000 E
77,000 N	17,500 E
77,500 N	18,000 E
78,000 N	18,500 E
78,500 N	19,000 E
79,000 N	19,500 E
79,500 N	20,000 E
80,000 N	20,500 E
80,500 N	21,000 E
81,000 N	21,500 E
81,500 N	22,000 E
82,000 N	
82,500 N	

Preliminary draft - fault and fold model projected to ground level, No 1 coal deposit, 1 in = 1,000 ft, 3rd November, 1976.

Preliminary subcrop contours of No 1 coal deposit, 1 in = 400 ft, 1st November, 1976.

Preliminary isopach of superficials, No 1 coal deposit, 1 in = 400 ft, 1st November, 1976.

Preliminary draft geological plans, No 1 coal deposit, scale 1 in = 400 ft, 10th November, 1976.

Elevations 2,400 ft
2,600 ft
2,800 ft

Preliminary total overburden isopach, No 1 coal deposit, scale 1 in = 400 ft, November 1976.

Preliminary draft subcrop geology, No 1 deposit, 1 in = 400 ft, 15th November, 1976.

Preliminary draft section 77,500 N, No 1 deposit, 1 in = 400 ft, revised 15th November, 1976.

Preliminary draft stratum contours, No 1 deposit, scale 1 in = 400 ft, 26th November, 1976.

Top of A Zone
Top of B Zone
Base of B Zone
Top of D Zone
Base of D Zone

MISCELLANEOUS CORRESPONDENCE

Letter, 3rd June, 1976, to B.A. Angel from L.T. Jory re computerisation of analytical data.

Letter, 4th June, 1976, to H.J. Goldie from L.T. Jory re sampling philosophy.

Letter, 10th June, 1976, to H.J. Goldie from L.T. Jory re history of sampling and assaying procedures.

No 1 coal deposit, ash-calorific value linear regression graphs with copy of covering letter of 15th June, 1976, to C. Guelke.

Table of copper and molybdenum values in composite samples, No 1 coal deposit with accompanying letter of 17th June, 1976, to H.J. Goldie. (Supplement to report by Dr. K. Fletcher, 2nd April, 1976).

Table showing preliminary subdivision of No 1 coal deposit into A, B, C and D zones.

Letter, 9th July, 1976, to C. Guelke from L.T. Jory re interim sampling and assaying procedures and new budget estimate for Stage 3B total assay costs.

Hat Creek Coal Development - Drilling Difficulties 1957/59, 1974, 1975, 1976.

Copy of letter concerning ash beds at Hat Creek, 26th July, 1976, from R.M. Quigley to L.T. Jory.

Copies of terms of reference for studies or assignments:-

(i) Assignment to Consultant for the Review of Mining Studies, Draft - 22nd June, 1976.

(ii) Hat Creek Utilisation Study - 14th June, 1976.

(iii) Hat Creek Thermal Power Plant Conceptual Design Study - 6th May, 1976.

(iv) Water Supply Study of the Proposed Hat Creek Development - 6th May, 1976.

(v) Hat Creek Diversion Study - Draft - 6th May, 1976.

(vi) Detailed Environmental Studies of the Proposed Hat Creek Development, Draft - 20th April, 1976.

DCA principal item work schedule.

DCA tender information for development drilling.

Copies of letters from L.T. Jory to C. Guelke dated 27th October and 29th October, 1976. DCA data on specific gravities for hole No 76-137.

DCA letter dated 10th September, 1976, referring to geographical reference to proposed pit locations.

Diary note by P.T. McCullough dated 18th June, 1976, entitled "Co-ordination of Diamond Drill Programs".

DCA data on drilling difficulties, holes 76-120 to 76-208.

Comments by DCA on PD-NCB Report No 5, 17th November, 1976.

Preliminary results of leachate test - Acres.

Letter from J.J. Fitzpatrick to S.C. Brealey dated 23rd December, 1976, detailing basic financial data.

FIELD SPECIFIC GRAVITY TESTS
FOR BOREHOLES

75-68 to 75-79, 75-79A, 75-80 to 75-83, 75-83A, 75-84 to 75-93, 75-95 to 75-103, 75-103A, 75-104, 75-106 to 75-110, 76-114 to 76-130, 76-132 to 76-184, 76-186 to 76-208, 76-814, 76-817.

COAL QUALITY AND ANALYTICAL DATA

Field slaking tests on drill cores for drill holes 75-90, 100, 101, 103, 104, 109, 110 and 76-112.

Graphic record of analyses, No 1 deposit, drill holes 74-23, 25, 26, 37A, 38, 39, 41, 43, 44, 46, 75-50, 51, 53, 106, 107, RH75-4.

No 1 coal deposit, statistical tables of proximate analysis data, 15th July 1975.

Sample record sheets (9) Commercial Test Laboratories DDH 76-135.

Sample record sheets (8) Commercial Test Laboratories DDH 76-136.

Special sample holes - samples and designated analysis holes 76-135, 76-136, revised 4th November, 1976.

Float-sink analytical data - special sample BAH 76-2 (Commercial Testing and Engineering Company).

Samples and designated analyses - DDH 76-136, sheet 5 (of 7). Shows corrected position of boundary between B and C stratigraphic zones. Replaces original sheet 5.

Proximate analysis - air dry moisture data plus copy of covering letter dated 1st September, 1976, to R.M. Woodley, BC Hydro.

DDHs 135 and 136 ash, calorific value, linear regression tables of proximate analysis data bound with covering letter.

Hat Creek development coal analysis schedules and budget.

Moisture samples of bucket auger holes.

Drill hole analytical data as follows:-

Drill Hole No	Sample Numbers	Remarks
76-120	120-1 to 120-65	Complete
76-122	122-1 to 122-6	Complete
76-125	125-1 to 125-17	Complete
76-126	126-1 to 126-23	Complete
76-127	127-1 to 127-10	Complete
76-128	128-1 to 128-21	Complete
76-129	129-1 to 129-2	Complete
76-132	132-1 to 132-7	Complete
76-133	133-1 to 133-36	Complete
76-134	134-1 to 134-22	Complete
76-135	135-1 to 135-108	Proximate, CV and
	135-113 and 135-114	sulphur including
	135-116 to 135-164	high ash
	136-168 and 135-169	
	135-171 to 135-175	
	135-180 to 135-199	
76-136	136-1 to 136-267	Complete
76-137	137-1 to 137-18	Complete
76-138	138-1 to 138-27	Complete
76-139	139-1 to 139-38	Complete
76-140	140-1 to 140-9	Complete
76-141	141-1 to 141-44	Complete
76-142	142-1 to 142-17	Complete
76-143	143-1 and 143-2	Complete
76-144	144-1 to 144-44	Complete
76-145	145-1 to 145-40	Complete
76-147	147-1 to 147-20	Complete
76-149	149-1 to 149-38	Complete
76-151	151-1 to 151-46	Complete
76-152	152-1 to 152-14	Complete
76-153	153-1 to 153-28	Complete
76-155	155-1 to 155-27	Complete
76-156	156-1 to 156-14	Complete
76-157	157-1 to 157-26	Complete

Drill Hole No	Sample Numbers			Remarks
76-160	160-1	to	160-13	Complete
76-162	162-1	to	162-22	Complete
76-163	163-1	to	163-10	Complete
76-165	165-1	to	165-15	Complete
76-173	173-1	to	173-6	Complete
76-176	176-1	to	176-6	Complete
76-177	177-1	to	177-5	Complete
76-814	814-1	to	814-9	Complete

APPENDIX "C"

ROSIN-RAMMLER THEORY OF COAL SIZE DISTRIBUTION

1. The breaking of coal is not a haphazard event in the sense that the distribution of sizes produced is entirely unpredictable. Among the many laws of size distribution that have been developed, that proposed by Rosin and Rammler in 1933 appears to most nearly represent the size distribution in broken coal. Rosin and Rammler originally applied their equation to fine coal only but, in 1936, J.G. Bennett found that the same law could be applied to the entire output of a mine.

2. Bennett represented the form of the Rosin-Rammler equation as:-

$$R = 100\, e^{-\left\{\frac{x}{\bar{x}}\right\}^{n}}$$

where:

R is the percentage retained on a sieve of opening size x

x is the screen opening

$\bar{x}$ is the absolute size constant

n is the size distribution constant

3. It is possible to reduce the equation to a straight line if log log reciprocal R is plotted against log x. This is shown in diagrammatic form on Plate 21(a). Special graph paper is available to facilitate this and an example is shown on Plate 21(b).

4. The size constant $\bar{x}$ is an indication of the average particle size as it measures the relative coarseness or fineness of the material. Its dimensions are those of length in the units used to express sieve opening width. If $\bar{x} = x$ then R becomes $100\ e^{-1}$ or 36.79%. In other words, 36.79% of the material is larger than $\bar{x}$ and 63.21% smaller. The size distribution constant n is a characteristic of the fracture pattern and is dimensionless.

5. Bennett, in his paper of 1936, showed that different coals exhibited generally similar distribution constants which lay within the following ranges:-

Rom coals	-	0.62 to 0.89
Coal fractured by hammer crusher or small slack screened from rom coal	-	about 1.0
Crushed coals smaller than 200μ	-	1 to 1.35

6. These figures were based on actual tests. Subsequent work at the British Coal Utilisation Research Association developed the theory behind these figures. This was published in 1941 in a series of three papers by Bennett, Brown and Crone.

7. It is not intended to go into the detailed mathematics behind the theory but one or two points are worthy of mention. Brown showed that a single cycle of total breakage yielded a size distribution constant of unity, two such cycles a value of 0.83 and three cycles 0.70. The exact definition of one cycle is not important in this context but the term "total fracture" implies that each and every lump in the sample is broken. This then implies that rom coal, with a distribution constant of 0.62 to 0.89, has experienced approximately two or three cycles of total breakage. This seems reasonable as by that stage it will normally have been blasted (or sheared), loaded and then passed through several transfer and transport phases, each of which will produce some breakage.

8. Obviously crushing in a jaw crusher is not total breakage since the intention is that only the large sizes should be broken. In theory all pieces smaller than the crusher setting should pass through unbroken. In practice, since the pieces are not presented to the crusher separately, some of the small material is broken. However, the effect is still to break the large pieces preferentially. This obviously increases the slope of the characteristic line and hence the value of the distribution constant. The use of a jaw crusher means that the material has a finite top size and thus the end of the characteristic line curves upwards.

9. There is also a limit on the actual minimum particle size since material cannot be broken below its ultimate grain size by mechanical forces alone. This means that though a typical characteristic is straight for the greater part of its length the two tails are curved to give an "S" shape as shown on Plate 21(b).

APPENDIX "D"

BASIC FINANCIAL DATA

This appendix up-dates Appendix "D" of Report No 2 in accordance with the letter from Mr. J. J. Fitzpatrick, Manager, Mining Department, Thermal Division, BC Hydro and Power Authority, dated 23rd December, 1976. (For comparison the Report No 2 figures are shown in brackets.)

1. Base date for economic calculations -
   October 1976 (1975)

2. Inflation Rate -

Fiscal Year (April/March)	Rate %		Index 1976/77 = 100	
1975/76	-	(Base)	-	(91)
1976/77	Base	(10)	100	(100)
1977/78	11	(10)	111	(110)
1978/79	9	(10)	121	(121)
1979/80	8	(10)	131	(133)
1980/81	7	(5)	140	(140)
1981/82	5	(5)	147	(147)
Thereafter	5	(5)	-	-

3. Discount Rate -
   Evaluations to be performed at 10% and 15% (15%)

4. Power Costs -
   Incremental energy costs 20 mils/kWh.
   To be separately identified. (10 mils/kWh)

5. 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%. (same)

6. Corporate Overhead Rate -
   A corporate overhead of 5% has been added to the uninflated direct costs of the project (see Plate 39). (same)

7. Tax and Debt Equity Ratio -
   Assumed that no tax paid by BC Hydro and that financial structure is 100% debt. (same)

8. Sales Tax -
Provincial sales tax 7% on purchased materials and equipment - not on labour costs. (zero)
No Federal sales tax is payable. (zero)

9. The Provincial Royalty -
Assumed to be 75¢ per long ton (equivalent to 67¢ per short ton). (same)

10. Power Station Operating Regime -

Capacity Factor	Period
70%	First 15 years
65%	Next 10 years
55%	Remaining 10 years

"The capacity factor is the percentage of the rated capacity that will be used - 8,760 hours per year. In establishing coal quantities required, we need to consider the gross power production required to meet our net output of 2,000 MW. Current planning is based on the use of four 560-MW units to produce the 2,000-MW net output. Net station heat rate is 10,443 Btu per net kWh."

(Report No 2 was based on three 750-MW units and the coal production was specified as "12 million short tons per year for the 35-year life of a 2,000-MW thermal generating station" in the terms of reference.)

TABLE I

1 metric tonne = 1·102311 short tons
1 cu. metre = 1·307951 cu. yds

BASIC PLANNING DATA
(Revised March 1977)

Density of in-situ coal	-	1.39 short tons/bank yd^3 = 1·65 tonne/m^3 (1·26 tonne)
Swell	-	25%
Density of in-situ waste in coal and claystone (assumed wet)	-	1.87 short tons/bank yd^3 = 2·22 tonne/m^3
Swell	-	33%
Density of rom coal	-	1.27 short tons/bank yd^3
Density of superficial deposits	-	1.56 short tons/bank yd^3
Swell	-	15%
Density of conditioned ash	-	1.2 short tons/yd^3 (loose)
Estimated in-situ waste content	-	29% ie 17% partings and 12% low grade coal
Estimated waste extraction by selective mining	-	24% ie 13% partings and 11% low grade coal
Waste remaining in rom coal	-	5%
Working days per year	-	350
Hours per shift	-	8
Teams of men (crews)	-	4
Number of producing shifts per week	-	20
Number of maintenance shifts per week	-	1

TABLE II

SUMMARY OF PARTING THICKNESSES AND FREQUENCY IN VARIOUS COAL ZONES

Parting Thickness (ft)	Zone A			Zone B			Zone C			Zone D			Total - All Zones		
	No of BH Inter-sections	Aggregate Length of BH Intersections (ft)	%	No of BH Inter-sections	Aggregate Length of BH Intersections (ft)	%	No of BH Inter-sections	Aggregate Length of BH Intersections (ft)	%	No of BH Inter-sections	Aggregate Length of BH Intersections (ft)	%	Aggregate Length of BH Intersections (ft)	% in Whole Deposit	% Removed by Mining
1	137	62.3	1.98	112	46.1	4.42	73	32.7	2.73	146	43.0	2.56	184.1	2.6	-
1 to 2	28	32.0	1.02	39	9.0	0.86	3	4.0	0.33	3	4.0	0.24	49.0	0.7	-
2 to 3	17	36.0	1.14	5	10.5	1.01	1	2.0	0.17	2	4.5	0.27	53.0	0.7	-
3 to 5	20	69.7	2.21	2	7.0	0.67	7	23.5	1.96	3	11.5	0.69	111.7	1.6	1.6
5 to 10	22	159.0	5.05	1	5.0	0.48	3	22.5	1.88	4	44.0	2.62	230.5	3.3	3.3
10	17	324.0	10.29	3	55.5	5.32	8	171.0	14.25	1	28.0	1.67	578.5	8.2	8.2
Sub-total - partings	241	683.0	21.69	162	133.1	12.76	95	255.7	21.32	159	135.0	8.05	1,206.8	17.1	13.1
Interlayered coal and waste	67	420.1	13.34	19	70.5	6.75	23	343.9	28.66	21	14.8	0.88	849.3	12.0	11.0
Total - partings, interlayered coal and waste	308	1,103.1	35.03	181	203.6	19.51	118	599.6	49.98	180	149.8	8.93	2,056.1	29.1	24.1

Notes: 1. Based on the following borehole intersections:-

Zone A - 76-120, 136, 144, 183, 191, 196, 200
Zone B - 76-120, 136, 144, 183, 190, 191, 196
Zone C - 76-120, 137, 157, 190, 191, 196, 200
Zone D - 76-125, 137, 157, 190, 191, 196, 200

2. Boreholes 76-191 to 208 were sampled more precisely than earlier boreholes.

3. Interlayered coal consists of multiple thin beds probably classified as "low-grade coal".

TABLE III

COMPARISON OF STATISTICAL PARAMETERS
FOR TEST HOLES 135 AND 136 FOR
DRY PROXIMATE ANALYSES

(Deposit No 1)

Zone	Variable	Dry Basis						20% Moisture	
		DDH 135			DDH 136			DDH 135	DDH 136
		No	Mean Value	Standard Deviation	No	Mean Value	Standard Deviation	Mean Value	Mean Value
A	Btu/lb	54	6,415	2,068	54	6,227	2,176	5,132	4,982
	Ash, %	54	42.92	14.00	56	45.68	16.41	34.34	36.54
	FC, %	54	27.46	9.54	54	26.15	10.01	21.97	20.92
	VM, %	54	29.63	5.08	54	29.41	5.78	23.70	23.53
	S total %	54	0.680	0.244	54	0.671	0.383	0.544	0.537
B	Btu/lb	24	7,679	1,229	26	7,639	1,645	6,143	6,111
	Ash, %	24	34.84	8.35	26	40.34	17.72	27.87	32.27
	FC, %	24	33.78	5.69	26	33.78	7.45	27.02	27.02
	VM, %	24	31.37	3.24	26	31.48	3.88	25.10	25.18
	S total %	24	0.792	0.190	26	0.817	0.256	0.634	0.654
C	Btu/lb	22	4,111	1,567	15	4,924	1,731	3,289	3,939
	Ash, %	22	58.42	10.46	15	51.83	10.48	46.74	41.46
	FC, %	22	17.31	7.38	15	21.59	7.53	13.85	17.27
	VM, %	22	24.27	4.49	15	26.58	3.58	19.42	21.26
	S total %	22	0.377	0.163	15	0.402	0.192	0.302	0.322
D	Btu/lb	25	9,211	1,371	29	9,665	1,010	7,369	7,732
	Ash, %	25	25.99	8.65	29	22.17	6.36	20.79	17.74
	FC, %	25	41.10	6.86	29	42.44	5.06	32.88	33.95
	VM, %	25	32.91	2.45	29	35.39	1.81	26.33	28.31
	S total %	25	0.231	0.067	29	0.296	0.061	0.185	0.237

Source: Report by Dr A J Sinclair, 20th September 1976

(20% moisture figures added by PD-NCB)

TABLE IV

WEIGHTED AVERAGES OF ALL 5-FT TO 40-FT SAMPLES BY STRATIGRAPHIC ZONE - PRE-1976

(Deposit No 1)

Zone	Btu/lb	Ash %	Fixed Carbon %	Volatiles %	Total Sulphur %
Dry Basis					
A	6,273	43.4	26.3	30.3	0.70
B	7,337	36.2	30.9	32.9	0.72
C	4,699	53.2	20.4	26.4	0.40
D	9,236	24.1	39.6	36.3	0.31
Combined	6,993	38.7	29.6	31.7	0.50
20% Moisture Basis					
A	5,018	34.7	21.0	24.3	0.56
B	5,870	29.0	24.7	26.3	0.58
C	3,759	42.6	16.3	21.1	0.32
D	7,389	19.3	31.7	29.0	0.25
Combined	5,594	30.9	23.7	25.4	0.40
Comparison with Report No 2	5,500	32	-	-	-

Source: "An Evaluation of Pre-1976 Proximate Analyses, No 1 Deposit Hat Creek" by Dr A J Sinclair dated 18th August, 1976

TABLE V

ESTIMATED MEAN COAL QUALITY BY STAGES (OPENPIT NO 1)

(at 20% moisture)

Stage	Ash %	Volatiles %	Fixed Carbon %	Gross Btu/lb	Total Sulphur %
3	30.1	27.7	22.2	5,539	0.44
4	31.4	26.5	22.1	5,441	0.45
5	29.2	27.4	23.4	5,719	0.29
6	26.4	27.8	25.8	6,283	0.27
7	28.2	25.9	25.9	5,993	0.34
8	33.6	25.2	21.2	5,137	0.58
combined	29.7	27.1	23.2	5,680	0.38

Notes:
1. Computed from drill hole intersections.
2. 30 drill holes included.
3. Weighted by sample length.
4. Samples included 1-199 and 201-299 series.
5. Based on computer print-out dated 7th October 1976 from BCH drill hole data base.
6. Stages 1 and 2 (development) omitted.

TABLE VI

SUMMARY OF DIAMOND DRILL HOLE FIELD SAMPLING PROCEDURES

Reference letter	10th June, 1976			4th June, 1976	9th July, 1976	8th August, 1976
DDH No	25-1 to 25-7	57-8 to 59-22	74-23 to 76-116	76-117 to 76-119	76-120 to 76-190	76-191 to 76-208
Length of samples	Unknown	Up to 100 ft; sometimes pieces of core combined into one sample	10 to 50 ft averaging 25 to 30 ft. Never less than 10 ft but controlled by geology	20 ft maximum	20 ft maximum	20 ft maximum with breaks at significant geological boundaries
Definitions for sampling purposes:						
Coal			More than 10% carbonaceous	As before	More than 10% carbonaceous	More than 35% carbonaceous visually
High-ash coal			More than 10% carbonaceous sampled as coal	As before	More than 10% carbonaceous sampled as coal	From 10% to 35% carbonaceous visually
Waste			Less than 10% carbonaceous	As before	Less than 10% carbonaceous	Less than 10% carbonaceous visually
Coal sampling					1. Sample separately if more than 3 ft thick and more than 3 ft from other coal 2. Include in adjacent coal sample if more than 3 ft thick and less than 3 ft from other coal 3. No sample if less than 3 ft thick and more than 3 ft from other coal	1. Sample separately if more than 3 ft thick and more than 10 ft from other coal 2. Sample separately if more than 1 ft thick and 1 to 10 ft from other coal 3. High-ash coal 1 to 3 ft thick and 1 to 10 ft from other coal - sample at Field Geologist's discretion
Thin interbedded coal and partings					Combine beds less than 3 ft thick into 1 sample up to 20 ft long	Combine coal beds less than 1 ft thick into 1 sample up to 20 ft long
Thin high-ash interbed coal sampling			A few samples less than 10 ft in length	As before		Combine high-ash coal beds less than 3 ft thick into 1 sample up to 20 ft long
Partings (waste) sampling			If less than 10 ft thick either no sample, or included in adjacent coal	As before	1. More than 10 ft thick - no sample 2. 3 to 10 ft thick - sample separately 3. Less than 3 ft thick - include in adjacent coal sample	1. More than 6 ft thick - sample separately 3 ft at top and bottom of adjacent coal 2. 1 to 6 ft thick - sample separately 3. Less than 1 ft thick - include in adjacent coal sample

Notes: 1. Reference letters from DCA to BCH

2. All thicknesses measured along core axis

TABLE VII

TENTATIVE SIZE DISTRIBUTION OF RUN-OF-MINE COAL

(i) Material blasted to give a maximum of 1% greater than 36 in

Size Range	Weight within Range %	Cumulative Weight %
+16 in	6.5	6.5
16 x 8 in	11.5	18.0
8 x 4 in	15.0	33.0
4 x 2 in	17.0	50.0
2 x 1 in	14.0	64.0
1 x ½ in	12.0	76.0
½ x ¼ in	8.0	84.0
¼ in x 28 mesh	12.3	96.3
28 x 48 mesh	1.3	97.6
48 x 100 mesh	0.8	98.4
-100 mesh	1.6	100.0

(ii) Material blasted to give a maximum of 1% greater than 20 in

Size Range	Weight within Range %	Cumulative Weight %
+16 in	2.0	2.0
16 x 8 in	6.0	8.0
8 x 4 in	12.0	20.0
4 x 2 in	16.0	36.0
2 x 1 in	16.0	52.0
1 x ½ in	14.0	66.0
½ x ¼ in	11.0	77.0
¼ in x 28 mesh	17.2	94.2
28 x 48 mesh	2.1	96.3
48 x 100 mesh	1.3	97.6
-100 mesh	2.4	100.0

TABLE VIII

SUMMARY OF POWER PLANT COAL REQUIREMENTS

Item	Unit	Stage 1							Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
		1978-79	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-2006	2006-13	2013-18	2018-22
Annual/stage power station heat requirement:	10^{12} Btu																
No 1 Unit		-	-	-	-	-	-	-	32	32	32	96	192	402	190	125	25
No 2 Unit		-	-	-	-	-	-	-	-	32	32	64	192	404	195	125	50
No 3 Unit		-	-	-	-	-	-	-	-	-	32	32	192	406	200	125	75
No 4 Unit		-	-	-	-	-	-	-	-	-	-	-	192	408	205	125	100
Total		-	-	-	-	-	-	-	32	64	96	192	768	1,620	790	500	250
Coal requirement at 5,950 Btu/lb:	10^6 ton																
Annual/stage		-	-	-	-	-	-	-	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42.0	21.0
Cumulative		-	-	-	-	-	-	-	2.7	8.1	16.2	16.2	81.0	217.5	283.9	325.9	346.9
Cumulative pit production	10^6 ton	-	-	0.1	0.3	0.7	1.1	1.1	3.8	9.2	17.3	17.3	82.1	218.6	285.0	327.0	348.0
Stockpile (by difference)	10^6 ton	-	-	0.1	0.3	0.7	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1	1.1
Alternative coal requirement based on 5,500 Btu/lb:	10^6 ton																
Annual/stage		-	-	-	-	-	-	-	2.9	5.8	8.7	17.4	69.6	147.1	71.8	45.5	22.7
Cumulative		-	-	-	-	-	-	-	2.9	8.7	17.4	17.4	87.0	234.1	305.9	351.4	374.1

Assumptions:

1. No 1 unit on full load 1st April, 1984 and annual additions thereafter.
2. Four units each 560 MW gross capacity, 500 MW (each) net capacity.
3. Net station heat rate 10,443 Btu per net kWh (BCH figure).
4. BCH capacity factors for each unit:-

   15 years at 70% (equivalent to 32 x 10^{12} Btu/yr)

   10 years at 65% (equivalent to 30 x 10^{12} Btu/yr)

   10 years at 55% (equivalent to 25 x 10^{12} Btu/yr)

TABLE IX

OPEN PIT NO 1 VOLUMES, TONNAGES AND RATIOS

A. Referred to In-situ Coal

Pit Stage	Pit Floor Elevation ft	Surficials		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	2,700	16	16	4	4	20	20	2	2	10.0	10.4
2	2,500	9	25	5	9	14	34	25	27	1.3	1.2
3	2,400	36	61	30	39	66	100	81	108	0.9	0.9
4	2,400	107	168	116	155	223	323	182	290	1.1	1.4
5	2,400	73	241	93	248	166	489	95	385	1.3	2.4
6	2,400	68	309	108	356	176	665	53	438	1.5	5.6
7	2,400	69	378	119	475	188	853	35	473	1.8	9.1
8	2,400	74	452	134	609	208	1,061	23	496	2.1	17.9
9	1,500	-	-	-	-	3,421	4,482	382	878	5.1	22.1

B. Referred to ROM Coal

Pit Stage	Pit Floor Elevation ft	Surficials		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	2,700	16	16	4	4	20	20	2	2	10.0	13.8
2	2,500	9	25	9	13	18	38	19	21	1.8	1.8
3	2,400	36	61	42	55	78	116	62	83	1.4	1.4
4	2,400	107	168	142	197	249	365	138	221	1.7	2.1
5	2,400	73	241	106	303	179	544	72	293	1.9	3.4
6	2,400	68	309	116	419	184	728	40	333	2.2	7.5
7	2,400	69	378	124	543	193	921	27	360	2.6	12.2
8	2,400	74	452	137	680	211	1,132	17	377	3.0	23.8
9	1,500	-	-	-	-	3,475	4,607	290	667	6.9	29.2

Notes:

1. Specific gravities used:

   In-pit waste 1.87 st/byd^3
   In-situ coal 1.39 st/byd^3
   ROM coal 1.27 st/byd^3
   Low-grade coal 1.52 st/byd^3
   Segregated waste 1.69 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 24% of the in-situ coal by weight, ie 13% partings and 11% low grade.
6. ROM coal 5,950 Btu/lb.
7. Project requires 348 million st ROM.

TABLE X

CO-ORDINATES OF PIT LIMITS

Pit Stage	Co-ordinates - North (ft)			Co-ordinates - East (ft)		
	Max	Min	Difference	Max	Min	Difference
No 1 Deposit						
7 (35 yr)	86,000	73,000	13,000	25,000	14,000	11,000
8 (600-ft pit)	86,000	72,500	13,500	25,500	12,500	13,000
9 (1,500-ft pit)	86,000	68,000	18,000	29,500	9,500	20,000
No 2 Deposit						
35 yr	68,500	46,500	22,000	28,500	17,000	11,500
8 (600-ft pit)	68,500	41,500	27,000	30,000	17,000	13,000
9 (1,500-ft pit)	68,500	35,500	33,000	33,000	12,000	21,000

("Pit limit" includes ramp excavation)

TABLE XI

MAXIMUM VERTICAL HEIGHT OF PIT SLOPES

Pit Stage	Pit Bottom Elevation (ft)	Pit Top Elevation (Highest Point) (ft)	Maximum Vertical Height (ft)	Direction of Highest Point from Pit Centre
No 1 Deposit				
7 (35 yr approx)	2,400	3,650	1,250	SW
8 (600-ft pit)	2,400	4,000	1,600	SW
9 (1,500-ft pit)	1,500	4,600	3,100	SW
No 2 Deposit				
35 yr	2,900	4,400	1,500	SE
8 (600-ft pit)	2,900	5,000	2,100	SE
9 (1,500-ft pit)	2,000	5,000	3,000	SE

TABLE XII

PROPORTIONS OF COAL ZONES BY MINING STAGES

(10^6 tons)

Mining Stage	Zone A		Zone B		Zone C		Zone D		Total Tons
	%	Tons	%	Tons	%	Tons	%	Tons	
1	-	-	-	-	-	-	100	2	2
2	24	6	20	5	24	6	32	8	25
3	40	32	22	18	21	17	17	14	81
4	34	62	7	13	24	44	35	63	182
5	31	29	6	6	18	17	45	43	95
6	36	19	7	4	15	8	42	22	53
7	34	12	14	5	15	5	37	13	35
8	30	7	18	4	22	5	30	7	23
Totals	34	167	11	55	20	102	35	172	496

Notes: 1. In terms of in-situ coal

2. Project terminates in Stage 7

TABLE XIII

SUMMARY OF YEARLY/STAGE AND CUMULATIVE PRODUCTION

Item	Unit	Stage 1					Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
		1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
In-situ coal production	10^6 tons														
Yearly/Stage		0.1	0.3	0.5	0.5	1.4	3.6	7.1	10.7	21.4	85.2	179.6	87.5	55.1	27.6
Cumulative		0.1	0.4	0.9	1.4	1.4	5.0	12.1	22.8	22.8	108.0	287.6	375.1	430.2	457.8
ROM coal production (5,950 Btu/lb)	10^6 tons														
Yearly/Stage		0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42.0	21
Cumulative		0.1	0.3	0.7	1.1	1.1	3.8	9.2	17.3	17.3	82.1	218.6	285.0	327.0	348
Segregated waste	10^6 tons														
Yearly/Stage		-	0.1	0.1	0.1	0.3	0.5	0.9	1.4	2.8	10.8	23.0	11.4	7.1	3.6
Cumulative		-	0.1	0.2	0.3	0.3	0.8	1.7	3.1	3.1	13.9	36.9	48.3	55.4	59
Low-grade coal	10^6 tons														
Yearly/Stage		-	-	0.1	0.1	0.2	0.4	0.8	1.2	2.4	9.6	20.1	9.7	6	3
Cumulative		-	-	0.1	0.2	0.2	0.6	1.4	2.6	2.6	12.2	32.3	42.0	48	51
Surficial waste	10^6 yd^3														
Yearly/Stage		4	4	5	5	18	4	4	4	12	39	101	74	65	36
Cumulative		4	8	13	18	18	22	26	30	30	69	170	244	309	345
Pit waste	10^6 yd^3														
Yearly/Stage		-	-	2	2	4	2	2	2	6	36	112	90	109	77
Cumulative		-	-	2	4	4	6	8	10	10	46	158	248	357	434
Total waste (excluding segregated waste)	10^6 yd^3														
Yearly/Stage		4	4	7	7	22	6	6	6	18	75	213	164	174	113
Cumulative		4	8	15	22	22	28	34	40	40	115	328	492	666	779
Yearly/Stage stripping ratio (in-situ coal basis)		40	13.3	14	14	15.7	1.7	0.8	0.6	0.8	0.9	1.2	1.9	3.2	4.1
Instantaneous stripping ratio (in-situ coal basis)		-	-	-	-	-	3.5	2.2	1.7	1.2	0.9	1.4	2.4	5.6	9.1
Pit waste plus segregated waste	10^6 tons														
Yearly/Stage		-	0.1	3.8	3.8	7.7	4.2	4.6	5.1	13.9	78.0	232.5	179.6	210.9	147.6
Cumulative		-	0.1	3.9	7.7	7.7	11.9	16.5	21.6	21.6	99.6	332.1	511.7	722.6	870.2
Yearly/Stage stripping ratio (ROM coal basis)		40	20	17.5	17.5	20	2.2	1.1	0.7	1.1	1.2	1.6	2.5	4.1	5.4
Instantaneous stripping ratio (ROM coal basis)		-	-	-	-	-	4.0	2.6	2.3	1.8	1.4	2.1	3.4	7.5	12.2
Total material removed	10^6 byd^3														
Yearly/Stage		4.1	4.2	7.4	7.4	23.1	8.6	11.1	13.7	33.4	136.2	342.4	227.0	213.6	132.9
Cumulative		4.1	8.3	15.7	23.1	23.1	31.7	42.8	56.5	56.5	192.7	535.1	762.1	975.7	1,108.6

TABLE XIV

SCHEDULE OF MOBILE MINING EQUIPMENT REQUIREMENTS

Item	Stage 1				Stage 2			Stage 3						Stage 4												
	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	2000-01	2001-02	2002-03	2003-04	2004-05	2005-06
Coal																										
Drills and compressors	1	1	1	1	2	2	2	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Shovels	1	1	1	1	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Trucks	1	1	1	1	2	3	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
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
Wheeldozers	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
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
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
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
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
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
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
Sump pumps	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	4	4	4	4	4	4
Pre-drainage	See Golder's Report No 6																									
Segregated Waste and Low-grade Coal																										
Shovels	Use coal shovels				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
Hydraulic shovels	-	-	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Small trucks	-	-	2	2	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6
Pit Waste																										
Shovels	-	-	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3
Trucks	-	-	2	2	2	2	2	4	4	4	4	4	4	5	5	6	6	6	6	6	6	8	8	8	9	9
Bulldozers	-	-	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	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
Water tankers	-	-	1	1	1	1	1	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Diesel tankers	-	-	1	1	1	1	1	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3
Maintenance vehicles	-	-	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	3	3	3	3	3	3	3
Graders	-	-	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	3
Pick-up trucks - 1 ton	-	-	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
Surficials																										
Scrapers	4	4	4	4	4	4	4	5	7	7	8	8	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Pushers/Bulldozers	2	2	2	2	2	2	2	2	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Water tankers	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
Diesel tankers	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2
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
Graders	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
Compactors	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

TABLE XIV

(continued)

Item	Stage 5							Stage 6					Stage 7			
	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	2020-21	2021-22
Coal																
Drills and compressors	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Shovels	3	3	3	3	3	3	3	3	3	3	3	3	2	2	2	1
Trucks	6	6	6	6	6	6	6	7	7	7	7	7	7	5	4	2
Bulldozers	1	1	1	1	1	1	1	2	2	2	2	2	2	1	1	1
Wheeldozers	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Water tankers	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
Maintenance vehicles	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Graders	1	1	1	1	1	1	1	2	2	2	2	2	2	2	1	1
Pick-up trucks - 1 ton	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Explosives trucks	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Sump pumps	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Pre-drainage	See Golder's Report No 6															
Segregated Waste and Low-grade Coal																
Shovels	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Trucks	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Hydraulic shovels	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2	1
Small trucks	8	8	8	8	8	8	8	8	8	8	8	8	8	6	6	3
Pit Waste																
Shovels	3	3	3	3	3	3	3	5	5	5	5	5	5	3	3	3
Trucks	11	11	11	11	11	11	12	21	21	22	22	24	24	20	19	18
Bulldozers	3	3	3	3	3	3	3	4	4	4	4	4	4	4	3	3
Wheeldozers	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water tankers	3	3	3	3	3	3	3	4	4	4	4	4	4	3	2	2
Diesel tankers	3	3	3	3	3	3	3	4	4	4	4	4	4	3	2	2
Maintenance vehicles	3	3	3	3	3	3	3	4	4	4	4	4	4	3	2	2
Graders	3	3	3	3	3	3	3	4	4	4	4	4	4	3	2	2
Pick-up trucks - 1 ton	4	4	4	4	4	4	4	5	5	5	5	5	5	4	3	3
Surficials																
Scrapers	5	5	6	6	6	6	6	7	7	7	7	7	7	6	5	4
Pushers/Bulldozers	2	2	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Water tankers	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Diesel tankers	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Maintenance vehicles	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
Graders	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2
Compactors	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2

TABLE XV

CONVEYOR SCHEDULE

From	Elevation (ft)	To	Elevation at Discharge (ft)	Difference in Level (ft)	No of Conveyors	Length (ft)	Capacity (tons/hr)	Belt Width (in)	Belt Speed (ft/min)	Approximate hp per Conveyor	Material
Bottom of incline	2,460	Surface interchange	2,800	340	3	6,000	4,480	60	1,000	3,150	Spoil or coal
Interchange to Houth Meadows dump	2,750	First section spoil dam	3,000	250	1	3,000	4,480	60	1,000	2,000	Pit waste and segregated waste
North end of spoil dam (movable)	2,750	South end of spoil dam	2,790 (spreader)	40	1	4,000	4,480	60	1,000	1,300	Pit waste and segregated waste
Interchange	2,750	Ash feeder interchange	3,800	1,050	1	11,000	3,920	60	1,000	7,300	Surficial waste or low-grade coal
Ash feeder interchange	3,800	Medicine Creek spoil dump	3,800	Nil	1	3,800	4,260	60	1,000	1,000	Surficials and ash
Medicine Creek dump north side of dam	3,800	North side of spoil area	4,000	200	1	4,000	4,260	60	1,000	1,950	Surficials and ash
Medicine Creek (spreader conveyor)	3,800	South side of dam (movable)	3,840 (spreader)	40	1	8,000	4,260	60	1,000	2,250	Surficials and ash
Coal stock recrusher	2,750	Power station	4,550	1,800	2	15,000	1,520	60	1,000	5,200	Coal
Power station	4,500	Ash feeder interchange	3,800	-700	1	8,000	450	36	800	300	Ash
Ash feeder interchange	3,800	Crusher near power station	4,500	700	1	7,000	2,800	60	600	3,150	Low-grade coal
Low-grade coal crusher	4,500	Power station	4,600	100	1	1,000	2,800	60	600	470	Low-grade coal
Low-grade coal conveyor tripper	4,100	South side of low-grade coal dump/stacking conveyor	4,120	20	1	3,000	2,800	60	600	420	Low-grade coal

Note: This table is not a specification but indicates the basis on which power requirements have been estimated at full production.

TABLE XVI

VOLUMES OF SPOIL SENT TO
WASTE DUMPS BY MINING STAGES

Mining Stage	Surficials				Pit and Segregated Waste				Conditioned Ash		Total to Dumps	
	10^6 bank yd^3		10^6 loose yd^3		10^6 bank yd^3		10^6 loose yd^3		10^6 loose yd^3		10^6 loose yd^3	
	zero swell	15% swell	stage	cum	pit waste	seg waste	stage total	cum	stage	cum	stage	cum
1	6	12	20	20	4	0	5	5	0	0	25	25
2	12	0	12	32	6	2	11	16	5	5	28	53
3	32	7	40	72	36	6	56	72	17	22	113	166
4	72	29	105	177	112	14	168	240	38	60	311	477
5	8	66	84	261	90	7	129	369	20	80	233	710
6	0	65	75	336	109	4	150	519	11	91	236	946
7 (part)	0	36	41	377	77	2	105	624	6	97	152	1,098
Total	130	215	377	-	434	35	624	-	97	-	1,098	-

Notes:
1. Bank volumes from Table XIII.
2. 130 million bank yd^3 of surficial material used in embankment construction with zero swell (see Report No 6). Remaining surficials at 15% swell.
3. Pit and segregated waste at 33% swell (see Report No 6).
4. Dry ash 28% of rom coal by weight (on average).
5. Ash conditioned to 15% moisture.
6. Loose density of conditioned ash approximately 1.2 tons/yd^3.

TABLE XVII

SUMMARY OF SPOIL SPACE BY ELEVATION

(10^6 yd^3)

Elevation	North Valley		Houth Meadows		Medicine Creek		Total	
	Interval	Cum	Interval	Cum	Interval	Cum	Interval	Cum
2,700	1	1	-	-	-	-	1	1
2,800	8	9	-	-	-	-	8	9
2,900	3	12	8	8	-	-	11	20
3,000	-	12	35	43	-	-	35	55
3,100	-	12	59	102	-	-	59	114
3,200	-	12	79	181	-	-	79	193
3,300	-	12	104	285	1	1	105	298
3,400	-	12	113	398	9	10	122	420
3,500	-	12	95	493	26	36	121	541
3,600	-	12	67	560	51	87	118	659
3,700	-	12	39	599	81	168	120	779
3,800	-	12	12	611	103	271	115	894
3,900	-	12	-	611	105	376	105	999
4,000	-	12	-	611	99	475	99	1,098

TABLE XVIII

SURFACE MINE BUILDINGS AND HOUSING COSTS
(INITIAL REQUIREMENTS)

Description	$ 10^3
Surface Mine Buildings	
Administration office	309
Change house	255
Maintenance bays, workshops and warehouse with equipment and tools	3,914
Core sheds	32
Powder magazines	50
Total	4,560
Single Work Force and Senior Staff Camp	
11 40-men bunkhouses	935
11 recreational units	210
Kitchen-diner	80
First aid	6
Company store	22
Sewage treatment for both camps (half of full price)	25
All services (electrical, sewer, gas, water)	55
Power generator for both camps (half of full price)	81
Single work force camp	1,414
Senior staff camp	460
Total	1,874
Permanent Housing	
217 detached houses	8,023
54 town houses	1,516
6 apartment blocks	1,470
Total	11,009

TABLE XIX

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
	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Coal														
Drills and compressors	102	-	102	-	204	204	-	204	408	1,020	2,550	1,530	918	816
Shovels	1,610	-	-	-	1,610	1,610	-	-	1,610	3,220	6,440	3,220	3,220	1,610
Trucks	607	-	607	-	1,214	1,214	607	2,428	4,249	10,926	23,066	13,354	9,712	4,856
Bulldozers	254	-	254	-	508	254	-	254	508	762	1,524	1,016	508	508
Wheeldozers	170	-	170	-	340	340	-	340	680	1,020	2,040	1,360	680	680
Water tankers	240	-	-	-	240	240	-	-	240	480	720	480	240	240
Diesel tankers	34	-	-	-	34	34	-	-	34	68	102	68	34	34
Maintenance vehicles	30	-	-	-	30	30	-	-	30	60	90	60	30	30
Graders	190	-	190	-	380	190	-	190	380	570	1,140	760	950	570
Pick-up trucks	12	-	-	-	12	12	-	-	12	24	36	36	18	12
Explosives trucks	40	-	-	-	40	40	-	-	40	80	120	160	80	40
Sump pumps	66	-	-	-	66	-	-	-	-	132	198	198	66	132
Pre-drainage pumps	-	348	610	-	958	1,058	-	-	1,058	1,283	3,462	3,546	541	236
Sub-totals	3,355	348	1,933	-	5,636	5,226	607	3,416	9,249	19,645	41,488	25,788	16,997	9,764
Segregated Waste and Low-grade Coal														
Shovels	-	-	-	-	-	1,610	-	-	1,610	-	3,220	-	1,610	-
Trucks	-	-	-	-	-	607	-	607	1,214	1,821	3,642	4,856	2,428	1,821
Hydraulic shovels	-	-	386	-	386	386	-	-	386	386	1,158	772	386	-
Small trucks	-	-	414	-	414	828	-	414	1,242	2,070	3,726	3,312	1,656	828
Sub-totals	-	-	800	-	800	3,431	-	1,021	4,452	4,277	11,746	8,940	6,080	2,649
Pit Waste														
Shovels	-	-	1,610	-	1,610	-	-	-	-	3,220	4,830	4,830	3,220	-
Trucks	-	-	1,214	-	1,214	1,214	-	1,214	2,428	7,284	27,922	23,673	37,027	20,031
Bulldozers	-	-	254	-	254	254	-	254	508	762	3,048	2,794	2,540	1,524
Wheeldozers	-	-	170	-	170	170	-	170	340	510	1,020	680	340	340
Water tankers	-	-	240	-	240	-	-	240	240	720	2,400	1,200	1,440	480
Diesel tankers	-	-	34	-	34	-	-	34	34	102	340	170	204	34
Maintenance vehicles	-	-	30	-	30	-	-	30	30	30	270	150	180	30
Graders	-	-	380	-	380	380	-	380	760	1,140	3,040	2,090	1,900	760
Pick-up trucks	-	-	12	-	12	-	-	12	12	24	54	48	36	18
Sub-totals	-	-	3,944	-	3,944	2,018	-	2,334	4,352	13,792	42,924	35,635	46,887	23,217
Surficials														
Scrapers	1,592	-	1,592	-	3,184	1,592	-	1,592	3,184	7,562	9,950	8,756	6,368	3,980
Pushers/bulldozers	508	-	508	-	1,016	508	-	508	1,016	2,032	3,048	2,794	1,524	1,270
Water tankers	240	-	-	-	240	480	-	-	480	960	1,440	1,440	720	480
Diesel tankers	34	-	-	-	34	34	-	-	34	68	238	102	102	34
Maintenance vehicles	30	-	-	-	30	30	-	-	30	60	90	120	60	30
Graders	190	-	190	-	380	380	-	380	760	1,140	2,280	2,280	1,140	950
Compactors	178	-	178	-	356	356	-	356	712	1,068	2,136	2,136	1,068	890
Sub-totals	2,772	-	2,468	-	5,240	3,380	-	2,836	6,216	12,890	19,182	17,628	10,982	7,634
Initial spares	613	35	474	61	1,183	759	61	121	941	856	399	832	396	-
Totals	6,740	383	9,619	61	16,803	14,814	668	9,728	25,210	51,460	115,739	88,823	81,342	43,264

TABLE XX

SCHEDULE OF FIXED INSTALLATIONS AND CAPITAL COSTS

($ 10^3)

Item	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-2006	2006-13	2013-18	2018-22
Coal Handling															
Incline conveyors	-	-	5,054	500	500	6,054	-	-	6,620	6,620	8,735	20,234	10,815	7,268	4,827
Surface interchange	-	-	2,825	-	-	2,825	-	-	32	32	32	-	-	-	-
Mine loading points	-	-	-	235	-	235	-	-	-	-	219	219	219	-	-
Electrics	496	65	1,050	110	73	1,794	248	10	110	368	921	2,330	1,541	563	315
Communications and control	-	200	-	500	200	900	-	-	-	-	700	1,100	700	200	-
Low-grade Coal															
Conveyors	-	-	-	-	8,062	8,062	1,546	-	-	1,546	1,608	14,938	-	-	-
Spreader/tripper	-	-	-	-	5,305	5,305	-	-	-	-	-	5,305	1,608	9,180	1,608
Electrics	-	-	-	108	204	312	52	-	6	58	161	758	161	256	114
Bulldozers	-	-	-	-	115	115	-	-	-	-	230	345	230	115	115
Waste Disposal															
Houth Meadows conveyors	-	-	4,215	-	-	4,215	128	128	944	1,200	4,128	6,468	5,836	1,531	534
Spreader/tripper	-	-	5,069	-	-	5,069	-	-	-	-	-	5,069	-	-	-
Electrics	-	-	176	-	-	176	6	-	89	95	133	280	222	101	57
Medicine Creek conveyors	-	-	-	-	14,221	14,221	22	22	22	66	9,088	28,767	3,034	13,738	2,902
Spreader/tripper	-	-	-	-	-	-	-	-	-	-	5,069	-	5,069	-	-
Electrics	-	-	541	-	-	541	-	-	119	119	669	1,442	788	419	294
Bulldozers	-	-	-	-	-	-	-	-	-	-	910	3,640	910	1,820	-
Initial spares	25	16	1,508	154	2,149	3,852	402	23	551	976	1,586	50	10	-	-
Sub-totals	521	281	20,438	1,607	30,829	53,676	2,404	183	8,493	11,080	34,189	90,945	31,143	35,191	10,766
Coal Stockpiling															
Stockyard conveyors	-	-	-	-	15,134	15,134	-	-	-	-	3,063	21,132	3,063	15,006	3,063
Stacking/recovery plant	-	-	-	-	7,618	7,618	-	-	-	-	-	15,236	-	7,618	-
Crushing plant	-	-	-	-	2,318	2,318	-	-	-	-	-	4,636	-	2,318	-
Coal conveyor to power station	-	-	-	-	28,494	28,494	-	-	-	-	4,740	37,824	4,740	27,268	4,740
Electrics	-	-	-	-	807	807	-	-	12	12	24	1,257	236	12	12
Initial spares	-	-	-	-	4,530	4,530	-	-	-	-	-	-	-	-	-
Sub-totals	-	-	-	-	58,901	58,901	-	-	12	12	7,827	80,085	8,039	52,222	7,815
Ash Disposal															
Conveyors	-	-	-	-	1,612	1,612	-	-	-	-	224	2,060	224	1,284	224
Transfer point	-	-	-	-	257	257	-	-	-	-	-	257	-	-	-
Trucks	-	-	-	-	621	621	-	-	-	-	621	1,863	621	621	621
Electrics	-	-	-	-	33	33	-	-	-	-	-	66	-	33	-
Initial spares	-	-	-	-	312	312	-	-	-	-	-	-	-	-	-
Sub-totals	-	-	-	-	2,835	2,835	-	-	-	-	845	4,246	845	1,938	845
Stage totals	-	-	-	-	-	115,412	-	-	-	11,092	42,861	175,276	40,027	89,351	19,426

TABLE XXI

SCHEDULE OF INFRASTRUCTURE

($ 10^3)

Unit	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-2006	2006-13	2013-18	2018-22
Infrastructure															
Hat Creek and road diversion	5,879	-	-	-	-	5,879	-	-	-	-	-	-	3,024	-	-
Administration buildings	-	-	103	103	103	309	-	-	-	-	-	159	150	-	-
Change house	-	-	-	-	-	-	-	100	155	255	191	191	-	-	-
Workshop and stores	-	900	1,000	1,000	1,014	3,914	-	-	-	-	-	-	3,081	191	-
Core sheds	-	6	6	6	6	24	-	4	4	8	18	16	24	24	8
Magazines	-	-	10	-	10	20	-	10	20	30	-	-	-	-	-
Roads	50	100	100	100	148	498	170	170	185	525	255	-	-	-	-
Sewage	25	25	-	-	-	50	-	-	-	-	50	50	50	50	-
Power and water supply	-	-	-	400	390	790	-	-	-	-	-	-	-	-	-
Buses	132	-	-	-	132	264	22	-	-	22	308	550	374	264	132
Pick-up trucks	-	60	-	-	-	60	90	-	-	90	270	630	560	280	210
Graders	-	380	-	-	-	380	380	-	-	380	760	1,140	760	380	-
Allowance for initial spares	50	100	-	-	95	245	-	-	-	-	-	105	-	-	-
Total	6,136	1,571	1,219	1,609	1,898	12,433	662	284	364	1,310	1,852	2,841	8,023	1,189	350
Employee Housing															
Trailer camps	400	400	400	400	274	1,874	-	-	-	-	-	-	-	-	-
Permanent houses	-	-	-	-	-	-	4,000	4,000	-	8,000	3,009	3,450	1,798	1,604	-
Total	400	400	400	400	274	1,874	4,000	4,000	-	8,000	3,009	3,450	1,798	1,604	-
Land purchase	-	-	-	1,080	1,080	2,160	-	-	-	-	-	1,368	-	-	-

TABLE XXII

SUMMARY OF POWER COSTS (BASED ON $0.02 PER kWh)
($ 10^3)

Item	Stage 1					Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-2006	2006-13	2013-18	2018-22
Mobile Mining Equipment														
Shovels	41	41	123	123	328	246	246	246	738	1,968	4,464	2,870	2,460	1,353
Mine pumps	-	28	28	28	84	28	28	28	84	198	471	273	195	156
Sub-totals	41	69	151	151	412	274	274	274	822	2,166	4,935	3,143	2,655	1,509
Fixed Installations														
Conveyors (coal)	-	-	11	11	22	56	120	180	356	1,428	3,013	1,487	952	486
Conveyors (surficials)	-	-	60	60	120	60	60	60	180	660	1,660	1,060	832	471
Conveyors (pit waste)	-	-	60	60	120	60	60	60	180	705	1,960	1,360	1,190	768
In-pit loading points	-	-	12	12	24	12	12	18	42	182	468	252	180	144
Interchange	-	-	3	3	6	5	7	12	24	123	293	169	126	96
Lighting	-	-	2	4	6	5	5	5	15	30	65	35	25	20
Spoil conveyor (Houth Meadows)	-	-	39	39	78	39	39	39	117	640	2,215	2,035	2,518	1,320
Spreader conveyor (Houth Meadows)	-	-	30	30	60	30	30	30	90	315	783	626	627	126
Spreader (Houth Meadows)	-	-	37	37	74	37	37	37	111	468	1,195	823	698	370
Spoil conveyor (Medicine Creek)	-	-	200	200	400	200	200	200	600	1,905	4,820	3,446	3,143	1,715
Spoil/ash conveyor (Medicine Creek)	-	-	26	26	52	33	33	33	99	339	846	624	512	297
Dump conveyor (Medicine Creek)	-	-	6	6	12	7	7	7	21	293	1,281	1,085	884	578
Spreader conveyor (Medicine Creek)	-	-	20	20	40	20	20	20	60	212	840	940	1,095	870
Spreader (Medicine Creek)	-	-	30	30	60	30	30	30	90	225	570	364	360	255
Low-grade coal conveyor	-	-	4	4	8	6	8	13	27	91	290	189	135	95
Low-grade coal spreader conveyor	-	-	1	1	2	3	3	4	10	30	65	35	25	20
Low-grade coal spreader	-	-	1	1	2	4	6	9	19	68	136	66	45	24
Sub-totals	-	-	542	544	1,086	607	677	757	2,041	7,714	20,500	14,596	13,347	7,655
Infrastructure														
Dam pumping	-	9	9	9	27	9	9	9	27	54	117	63	45	36
Buildings	20	30	40	55	145	55	55	55	165	330	715	385	275	220
Services	10	20	30	36	96	36	36	36	108	216	468	252	180	144
Sub-totals	30	59	79	100	268	100	100	100	300	600	1,300	700	500	400
Totals	71	128	772	795	1,766	981	1,051	1,131	3,163	10,480	26,735	18,439	16,502	9,564
Production, 10^6 tons	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42.0	21.0
Average cost, $/ton	0.7	0.64	1.93	2.0	1.60	0.36	0.19	0.14	0.19	0.16	0.20	0.28	0.40	0.46
Average cost, ¢/10^6 Btu	6	5	16	17	13	3	2	1	2	1	2	3	4	4
Coal Handling														
Interchange	-	-	-	3	3	3	7	8	18	72	156	78	50	23
Stocking and recovery	-	-	-	97	97	150	300	450	900	3,462	7,501	3,721	2,355	1,145
Coal conveyor to power station	-	-	-	-	-	170	330	500	1,000	3,930	8,515	4,180	2,600	1,330
Total	-	-	-	100	100	323	637	958	1,918	7,464	16,172	7,979	5,005	2,498

TABLE XXIII

SUMMARY OF LABOUR 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
				1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
	$	$	$															
Mobile Mining Equipment																		
Equipment operators	7.45	9.00	18,000	-	396	396	666	666	2,124	918	1,440	1,818	4,176	20,160	49,302	34,236	29,160	19,908
Maintenance personnel	7.80	9.40	19,000	-	342	342	570	570	1,824	798	1,216	1,520	3,534	17,024	41,572	28,956	24,624	16,796
Labourers	7.00	8.45	17,000	-	68	68	119	119	374	153	238	306	697	3,417	8,330	5,831	4,964	3,366
Overtime	-	-	-	-	40	40	136	136	352	94	145	364	603	4,058	9,921	6,904	5,875	4,000
Sub-totals	-	-	-	-	846	846	1,491	1,491	4,674	1,963	3,039	4,008	9,010	44,659	109,125	75,927	64,623	44,070
Fixed Installations																		
Equipment operators	7.45	9.00	18,000	-	252	252	252	252	1,008	504	576	648	1,728	5,940	15,912	8,568	6,120	4,896
Maintenance personnel	7.80	9.40	19,000	-	95	95	95	95	380	190	247	285	722	2,660	7,163	3,857	2,755	2,204
Labourers	7.00	8.45	17,000	-	153	153	153	153	612	170	170	170	510	1,020	2,210	1,190	850	680
Overtime	-	-	-	-	25	25	25	25	100	86	50	55	191	480	1,261	679	485	388
Sub-totals	-	-	-	-	525	525	525	525	2,100	950	1,043	1,158	3,151	10,100	26,546	14,294	10,210	8,168
Infrastructure																		
Equipment operators	7.45	9.00	18,000	144	72	72	72	72	432	72	72	72	216	792	2,664	1,872	1,440	1,152
Maintenance personnel	7.80	9.40	19,000	-	38	38	38	38	152	38	38	38	114	418	1,406	988	760	608
Labourers	7.00	8.45	17,000	-	170	170	170	170	680	170	170	170	510	1,870	6,290	4,420	3,400	2,720
Overtime	-	-	-	-	13	13	13	13	52	13	13	13	39	153	518	364	280	224
Sub-totals	-	-	-	144	293	293	293	293	1,316	293	293	293	879	3,233	10,878	7,644	5,880	4,704
Totals	-	-	-	144	1,664	1,664	2,309	2,309	8,090	3,206	4,375	5,459	13,040	57,992	146,549	97,865	80,713	56,942
Production, 10^6 tons	-	-	-	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42.0	21.0
Average cost, $/ton	-	-	-	-	-	-	-	-	-	1.19	0.81	0.67	0.80	0.89	1.07	1.47	1.92	2.71
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	-	-	-	10	7	6	7	7	9	12	16	23
Coal Handling																		
Operators	7.45	9.00	18,000	-	72	72	72	216	432	324	324	324	972	1,944	4,212	2,268	1,620	1,296
Maintenance personnel	7.80	9.40	19,000	-	38	38	38	114	228	171	171	171	513	1,026	2,223	1,197	855	684
Labourers	7.00	8.45	17,000	-	34	34	34	102	204	170	170	170	510	1,020	2,210	1,190	850	680
Overtime	-	-	-	-	7	7	7	22	43	33	33	33	99	198	429	231	165	132
Totals	-	-	-	-	151	151	151	454	907	698	698	698	2,094	4,188	9,074	4,886	3,490	2,792
Average cost, $/ton	-	-	-	-	-	-	-	-	-	0.26	0.13	0.09	0.13	0.06	0.07	0.07	0.08	0.13
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	-	-	-	2	1	1	1	1	1	1	1	1
Ash Handling																		
Equipment operators	7.45	9.00	18,000	-	-	-	-	-	-	90	90	90	270	1,080	2,340	1,260	900	720
Maintenance personnel	7.80	9.40	19,000	-	-	-	-	-	-	81	81	81	243	972	2,106	1,134	810	648
Labourers	7.00	8.45	17,000	-	-	-	-	-	-	34	34	34	102	306	663	357	255	204
Overtime	-	-	-	-	-	-	-	-	-	10	10	10	30	120	260	140	100	80
Totals	-	-	-	-	-	-	-	-	-	215	215	215	645	2,478	5,369	2,891	2,065	1,652
Average cost, $/ton	-	-	-	-	-	-	-	-	-	0.08	0.04	0.03	0.04	0.04	0.04	0.04	0.05	0.08
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	-	-	-	1	-	-	-	-	-	-	-	1
Total average cost, $/ton	-	-	-	-	-	-	-	-	-	1.53	0.98	0.79	0.97	0.99	1.18	1.58	2.05	2.92
Total average cost, ¢/10^6 Btu	-	-	-	-	-	-	-	-	-	13	8	7	8	8	10	13	17	25
Average annual employees	-	-	-	6	96	96	126	142	94	214	277	324	272	545	627	763	871	776

TABLE XXIV

SUMMARY OF MATERIALS AND FUEL COST EXCLUDING ELECTRIC POWER
($ 10^3)

Item	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-2006	2006-13	2013-18	2018-22
Mobile mining equipment	-	1,027	1,057	1,977	2,430	6,491	3,643	4,298	5,008	12,949	45,351	100,454	72,204	70,110	45,629
Explosives (by contract)	-	9	26	43	43	121	310	611	920	1,841	7,326	15,445	7,527	4,739	2,374
Exploratory drilling (by contract)	-	486	486	486	486	1,944	486	486	486	1,458	2,916	6,318	3,402	2,430	-
Sub-totals	-	1,522	1,569	2,506	2,959	8,556	4,439	5,395	6,414	16,248	55,593	122,217	83,133	77,279	48,003
Fixed installations	-	5	5	727	727	1,464	1,071	1,071	1,071	3,213	7,782	19,799	10,661	7,615	6,092
Infrastructure	97	218	218	218	218	969	283	283	283	849	1,878	6,138	3,136	2,270	1,468
Engineering and administration	20	20	20	20	20	100	37	37	37	111	264	572	385	275	220
Totals	117	1,765	1,812	3,471	3,924	11,089	5,830	6,786	7,805	20,421	65,517	148,726	97,315	87,439	55,783
Production, 10^6 tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42	21
Average cost, $/ton	-	-	-	-	-	-	2.16	1.26	0.96	1.26	1.01	1.09	1.47	2.08	2.66
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	18	11	8	11	8	9	12	17	22
Coal handling	-	-	-	-	350	350	700	1,000	1,300	3,000	7,938	17,199	9,261	6,615	3,223
Average cost, $/ton	-	-	-	-	-	-	0.26	0.19	0.16	0.19	0.12	0.13	0.14	0.16	0.15
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	2	2	1	2	1	1	1	1	1
Ash handling	-	-	-	-	50	50	50	50	50	150	300	650	350	250	200
Average cost, $/ton	-	-	-	-	-	-	0.02	0.01	0.01	0.01	-	-	0.01	0.01	0.01
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Total average cost, $/ton	-	-	-	-	-	-	2.44	1.46	1.13	1.46	1.13	1.22	1.62	2.25	2.82
Total average cost, ¢/10^6 Btu	-	-	-	-	-	-	20	13	9	13	9	10	13	18	23

TABLE XXV

SUMMARY OF DIRECT OPERATING COSTS
($ 10^3)

Category	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Mobile Mining Equipment															
Labour	-	846	846	1,491	1,491	4,674	1,963	3,039	4,008	9,010	44,659	109,125	75,927	64,623	44,070
Materials, fuel and miscellaneous	-	1,522	1,569	2,506	2,959	8,556	4,439	5,395	6,414	16,248	55,593	122,217	83,133	77,279	48,003
Electric power	-	41	69	151	151	412	274	274	274	822	2,166	4,935	3,143	2,655	1,509
Sub-totals	-	2,409	2,484	4,148	4,601	13,642	6,676	8,708	10,696	26,080	102,418	236,277	162,203	144,557	93,582
Fixed Installations															
Labour	-	525	525	525	525	2,100	950	1,043	1,158	3,151	10,100	26,546	14,294	10,210	8,168
Materials and fuel	-	5	5	727	727	1,464	1,071	1,071	1,071	3,213	7,782	19,799	10,661	7,615	6,092
Electric power	-	-	-	542	544	1,086	607	677	757	2,041	7,714	20,500	14,596	13,347	7,655
Sub-totals	-	530	530	1,794	1,796	4,650	2,628	2,791	2,986	8,405	25,596	66,845	39,551	31,172	21,915
Infrastructure															
Labour	144	293	293	293	293	1,316	293	293	293	879	3,233	10,878	7,644	5,880	4,704
Materials and fuel	97	218	218	218	218	969	283	283	283	849	1,878	6,138	3,136	2,270	1,468
Electric power	-	30	59	79	100	268	100	100	100	300	600	1,300	700	500	400
Sub-totals	241	541	570	590	611	2,553	676	676	676	2,028	5,711	18,316	11,480	8,650	6,572
Engineering and Administration															
Salaries	596	635	863	976	1,122	4,192	1,320	1,595	1,595	4,510	10,734	25,041	15,645	11,175	8,940
Materials	20	20	20	20	20	100	37	37	37	111	264	572	385	275	220
Sub-totals	616	655	883	996	1,142	4,292	1,357	1,632	1,632	4,621	10,998	25,613	16,030	11,450	9,160
Consultants fees	550	550	550	550	550	2,750	275	275	275	825	1,650	3,575	1,925	1,375	1,100
Totals	1,407	4,685	5,017	8,078	8,700	27,887	11,612	14,082	16,265	41,959	146,373	350,626	231,189	197,204	132,329
Average cost, $/ton	-	-	-	-	-	-	4.30	2.61	2.01	2.59	2.26	2.57	3.48	4.70	6.30
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	36	22	17	22	19	22	29	39	53
Coal Handling															
Labour	-	151	151	151	454	907	698	698	698	2,094	4,188	9,074	4,886	3,490	2,792
Materials and fuel	-	-	-	-	350	350	700	1,000	1,300	3,000	7,938	17,199	9,261	6,615	3,223
Electric power	-	-	-	-	100	100	323	637	958	1,918	7,464	16,172	7,979	5,005	2,498
Sub-totals	-	151	151	151	904	1,357	1,721	2,335	2,956	7,012	19,590	42,445	22,126	15,110	8,513
Average cost, $/ton	-	-	-	-	-	-	0.64	0.43	0.36	0.43	0.30	0.31	0.33	0.36	0.41
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	5	4	3	4	3	3	3	3	3
Ash Handling															
Labour	-	-	-	-	-	-	215	215	215	645	2,478	5,369	2,891	2,065	1,652
Materials and fuel	-	-	-	-	50	50	50	50	50	150	300	650	350	250	200
Electric power	Nil - (Regenerative)														
Sub-totals	-	-	-	-	50	50	265	265	265	795	2,778	6,019	3,241	2,315	1,852
Average cost, $/ton	-	-	-	-	-	-	0.10	0.05	0.03	0.05	0.04	0.04	0.05	0.06	0.09
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	1	-	-	-	-	-	-	1	1
Production, 10^6 tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	4.2	2.1
Total average cost, $/ton	-	-	-	-	-	-	5.04	3.09	2.4	3.07	2.6	2.92	3.86	5.12	6.80
Total average cost, ¢/10^6 Btu	-	-	-	-	-	-	42	26	20	26	22	25	32	43	57
Total volume of material removed, 10^6/byd^3	-	4.1	4.2	7.4	7.4	23.1	8.6	11.1	13.7	33.4	136.2	342.4	227.0	213.6	132.9
Average cost, $/byd^3	-	-	-	-	-	-	1.58	1.50	1.42	1.49	1.24	1.16	1.13	1.00	1.07

TABLE XXVI

SUMMARY OF DEPRECIATION
($ 10^3)

Item	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Mobile mining equipment	-	2,232	2,239	3,643	3,643	11,757	5,226	5,530	6,138	16,894	48,663	113,721	80,825	82,360	53,343
Fixed installations	-	12	1,158	1,273	3,462	5,905	3,837	3,879	4,697	12,413	35,779	91,405	49,378	35,270	28,216
Infrastructure	184	294	294	294	294	1,360	460	460	460	1,380	2,886	6,950	5,982	5,486	4,440
Other capitalised costs	60	60	120	120	480	840	1,000	1,290	1,500	3,790	9,600	20,800	11,200	8,000	6,400
Total depreciation	244	2,598	3,811	5,330	7,879	19,862	10,523	11,159	12,795	34,477	96,928	232,876	147,385	131,116	92,399
Cumulative depreciation	244	2,842	6,653	11,983	19,862	19,862	30,385	41,544	54,339	54,339	151,267	384,143	531,528	662,644	755,043
Production, 10^6 tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42.0	21.0
Average cost, $/ton	-	-	-	-	-	-	3.90	2.07	1.58	2.13	1.50	1.71	2.22	3.12	4.40
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	33	17	13	18	13	14	19	26	37
Coal handling	-	-	-	-	4,920	4,920	4,920	4,920	4,920	14,760	29,520	63,960	34,440	24,600	19,680
Average cost, $/ton	-	-	-	-	-	-	1.82	0.91	0.61	0.91	0.46	0.47	0.52	0.59	0.94
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	15	8	5	8	4	4	4	5	8
Ash handling	-	-	-	-	189	189	189	189	189	567	1,134	2,457	1,323	945	756
Average cost, $/ton	-	-	-	-	-	-	0.07	0.04	0.02	0.04	0.02	0.02	0.02	0.02	0.04
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	1	-	-	-	-	-	-	-	-
Total average cost, $/ton	-	-	-	-	-	-	5.79	3.02	2.21	3.08	1.98	2.20	2.76	-	5.38
Total average cost, ¢/10^6 Btu	-	-	-	-	-	-	49	25	18	26	17	18	23	-	45

TABLE XXVII

CAPITAL INVESTMENT, INTEREST DURING CONSTRUCTION, INTEREST AND INSURANCE - MINE

($\$10^3$)

Item	Stage 1						Stage 2	Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-87	1987-93	1993-06	2006-13	2013-18	2018-22
Mobile mining equipment	-	6,740	383	9,619	61	16,803	25,210	51,460	115,739	88,823	81,342	43,264
Fixed installations	521	281	20,438	1,607	30,829	53,676	11,080	34,189	90,945	31,143	35,191	10,766
Infrastructure	6,136	1,571	1,219	1,609	1,898	12,433	1,310	1,852	2,841	8,023	1,189	350
Sub-totals	6,657	8,592	22,040	12,835	32,788	82,912	37,600	87,501	209,525	127,989	117,722	54,380
Direct operating costs to start-up	1,407	4,685	5,017	8,078	8,700	27,887	-	-	-	-	-	-
Insurance costs to start-up	133	172	441	257	656	1,659	-	-	-	-	-	-
Working capital	750	750	750	750	750	3,750	2,250	-	-	-	-	-
Total capital costs	8,947	14,199	28,248	21,920	42,894	116,208	39,850	87,501	209,525	127,989	117,722	54,380
Corporate overhead	1,162	1,162	1,162	1,162	1,162	5,810	-	-	-	-	-	-
Total capital costs including corporate overhead	10,109	15,361	29,410	23,082	44,056	122,018	39,850	87,501	209,525	127,989	117,722	54,380
Cumulative capital costs including corporate overhead	10,109	25,470	54,880	77,962	122,018	122,018	161,868	249,369	458,894	586,883	704,605	758,985
Interest on cumulative capital costs up to beginning of year (10%)	-	1,061	2,780	6,147	9,185	19,173	-	-	-	-	-	-
Interest on capital cost during year (5%)	505	768	1,471	1,154	2,203	6,101	-	-	-	-	-	-
Total interest during construction	505	1,829	4,251	7,301	11,388	25,274	-	-	-	-	-	-
Total investment	10,614	17,190	33,661	30,383	55,444	147,292	39,850	87,501	209,525	127,989	117,722	54,380
Cumulative investment	10,614	27,804	61,465	91,848	147,292	147,292	187,142	274,643	484,168	612,157	729,879	784,259
Depreciation	244	2,598	3,811	5,330	7,879	19,862	34,477	96,928	232,876	147,385	131,116	92,399
Cumulative depreciation	244	2,842	6,653	11,983	19,862	19,862	54,339	151,267	384,143	531,528	662,644	755,043
Outstanding investment at year or stage end	10,370	24,962	54,812	79,865	127,430	127,430	132,803	123,376	100,025	80,629	67,235	29,216
Average outstanding investment	5,185	17,666	39,887	67,339	103,648	46,745	130,639	128,282	109,792	89,057	82,491	50,320
Interest on outstanding investment (10% pa)	518	1,767	3,989	6,734	10,365	23,373	39,192	76,969	142,729	62,340	41,245	20,128
Insurance (2% pa)	104	353	798	1,347	2,073	4,675	7,838	15,394	28,546	12,468	8,249	4,026

TABLE XXVIII

CAPITAL INVESTMENT, INTEREST AND INSURANCE FOR COAL AND ASH HANDLING
($ 10^3)

Item	Stage 1		Stage 2	Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1983-84	Total	1984-87	1987-93	1993-06	2006-13	2013-18	2018-22
Coal Handling								
Total investment	58,901	58,901	12	7,827	80,085	8,039	52,222	7,815
Cumulative investment	58,901	58,901	58,913	66,740	146,825	154,864	207,086	214,901
Depreciation	4,920	4,920	14,760	29,520	63,960	34,440	24,600	19,680
Cumulative depreciation	4,920	4,920	19,680	49,200	113,160	147,600	172,200	191,880
Outstanding investment at year or stage end	53,981	53,981	39,233	17,540	33,665	7,264	34,886	23,021
Average outstanding investment	26,991	26,991	46,603	29,033	25,086	20,377	31,575	29,931
Interest on outstanding investment (10% pa)	2,699	2,699	13,981	17,420	32,611	14,264	15,788	11,972
Insurance (2% pa)	540	540	2,796	3,484	6,522	2,853	3,158	2,394
Ash Handling								
Total investment	2,835	2,835	-	845	4,246	845	1,938	845
Cumulative investment	2,835	2,835	2,835	3,680	7,926	8,771	10,709	11,554
Depreciation	189	189	567	1,134	2,457	1,323	945	756
Cumulative depreciation	189	189	756	1,890	4,347	5,670	6,615	7,371
Outstanding investment at year or stage end	2,646	2,646	2,079	1,790	3,579	3,101	4,094	4,183
Average outstanding investment	1,323	1,323	2,363	2,146	2,787	3,461	4,373	4,456
Interest on outstanding investment (10% pa)	132	132	708	1,288	3,623	2,423	2,187	1,782
Insurance (2% pa)	26	26	142	258	725	485	437	356

TABLE XXIX

ROM PRODUCTION COSTS FOR MINE
($ 10^3)

Item	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Coal production, 10^6 ROM tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42	21
Direct operating cost	1,407	4,685	5,017	8,078	8,700	27,887	11,612	14,082	16,265	41,959	146,373	350,626	231,189	197,204	132,329
Depreciation	244	2,598	3,811	5,330	7,879	19,862	10,523	11,159	12,795	34,477	96,928	232,876	147,385	131,116	92,399
Interest on outstanding investment	518	1,767	3,989	6,734	10,365	23,373	13,149	13,090	12,953	39,192	76,969	142,729	62,340	41,245	20,128
Insurance	104	353	798	1,347	2,073	4,675	2,629	2,618	2,591	7,838	15,394	28,546	12,468	8,249	4,026
Royalty (75¢/long ton, 67¢/short ton	-	-	-	-	-	-	1,809	3,618	5,427	10,854	43,416	91,455	44,488	28,140	14,070
Total cost/year or stage	2,273	9,403	13,615	21,489	29,017	75,797	39,722	44,567	50,031	134,320	379,080	846,232	497,870	405,954	262,952
Average cost, $/ton	-	-	-	-	-	-	14.71	8.25	6.17	8.29	5.85	6.20	7.50	9.66	12.52
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	124	69	52	70	49	52	63	81	105

TABLE XXX

ROM PRODUCTION COSTS INCLUSIVE OF COAL AND ASH HANDLING
($ 10^3)

Item	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Coal production, 10^6 ROM tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42	21
Mine															
Total cost/year or stage	2,273	9,403	13,615	21,489	29,017	75,797	39,722	44,567	50,031	134,320	379,080	846,232	497,870	405,954	262,952
Average cost, $/ton	-	-	-	-	-	-	14.71	8.25	6.17	8.29	5.85	6.20	7.50	9.66	12.52
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	124	69	52	70	49	52	63	81	105
Coal Handling															
Direct operating cost	-	151	151	151	904	1,357	1,721	2,335	2,956	7,012	19,590	42,445	22,126	15,110	8,513
Depreciation	-	-	-	-	4,920	4,920	4,920	4,920	4,920	14,760	29,520	63,960	34,440	24,600	19,680
Interest on outstanding investment	-	-	-	-	2,699	2,699	5,152	4,660	4,169	13,981	17,420	32,611	14,264	15,788	11,972
Insurance	-	-	-	-	540	540	1,030	932	834	2,796	3,484	6,522	2,853	3,158	2,394
Royalty	Included in Mining Costs														
Total cost/year or stage	-	151	151	151	9,063	9,516	12,823	12,847	12,879	38,549	70,014	145,538	73,683	58,656	42,559
Average cost, $/ton	-	-	-	-	-	-	4.75	2.38	1.59	2.38	1.08	1.07	1.11	1.40	2.02
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	40	20	13	20	9	9	9	12	17
Ash Handling															
Direct operating cost	-	-	-	-	50	50	265	265	265	795	2,778	6,019	3,241	2,315	1,852
Depreciation	-	-	-	-	189	189	189	189	189	567	1,134	2,457	1,323	945	756
Interest on outstanding investment	-	-	-	-	132	132	255	236	217	708	1,288	3,623	2,423	2,187	1,782
Insurance	-	-	-	-	26	26	51	47	44	142	258	725	485	437	356
Total cost/year or stage	-	-	-	-	397	397	760	737	715	2,212	5,458	12,824	7,472	5,884	4,746
Average cost, $/ton	-	-	-	-	-	-	0.28	0.13	0.09	0.14	0.08	0.09	0.11	0.14	0.23
Average cost, ¢/10^6 Btu	-	-	-	-	-	-	2	1	1	1	1	1	1	1	2
Total production cost/year or stage	2,273	9,554	13,766	21,640	38,477	85,710	53,305	58,151	63,625	175,081	454,552	1,004,594	579,025	470,494	310,257
Total average cost, $/ton	-	-	-	-	-	-	19.74	10.76	7.85	10.81	7.01	7.36	8.72	11.20	14.77
Total average cost, ¢/10^6 Btu	-	-	-	-	-	-	166	90	66	91	59	62	73	94	124

TABLE XXXI

CASH FLOW (EXPENSES) AND UNIFORM SELLING PRICE - MINE
($ 10^3)

Category	Stage 1						Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Capital investment	10,614	17,190	33,661	30,383	55,444	147,292	18,630	1,885	19,335	39,850	87,501	209,525	127,989	117,722	54,380
Direct operating costs	-	-	-	-	-	-	11,612	14,082	16,265	41,959	146,373	350,626	231,189	197,204	132,329
Insurance	-	-	-	-	-	-	2,629	2,618	2,591	7,838	15,394	28,546	12,468	8,249	4,026
Royalty (75 ¢/long ton, 67 ¢/short ton)	-	-	-	-	-	-	1,809	3,618	5,427	10,854	43,416	91,455	44,488	28,140	14,070
Cash flow expenses	10,614	17,190	33,661	30,383	55,444	147,292	34,680	22,203	43,618	100,501	292,684	680,152	416,134	351,315	204,805
Discounted cash flow at 15%	9,230	12,996	22,132	17,370	27,561	89,289	14,992	8,346	14,259	37,597	59,897	40,228	5,617	2,063	644
Discounted cash flow at 10%	9,649	14,199	25,279	20,752	34,431	104,310	19,560	11,390	20,370	51,320	98,655	96,264	21,910	10,525	4,019
Coal production, 10^6 tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42	21
Discounted coal production at 15%, 10^6 tons	-	0.08	0.13	0.23	0.20	0.64	1.17	2.03	2.65	5.85	13.37	8.38	0.92	0.24	0.07
Discounted coal production at 10%, 10^6 tons	-	0.08	0.15	0.27	0.25	0.75	1.52	2.77	3.78	8.07	21.95	19.83	3.56	1.24	0.43

Uniform Selling Price

	$/ton	¢/10^6 Btu
15% discount rate	7.99	67
10% discount rate	6.93	58

TABLE XXXII

CASH FLOW (EXPENSES) AND UNIFORM SELLING PRICE - COAL AND ASH HANDLING

($ 10^3)

Category	Stage 1					Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Coal Handling														
Capital investment	-	-	-	58,901	58,901	-	-	12	12	7,827	80,085	8,039	52,222	7,815
Direct operating costs	151	151	151	904	1,357	1,721	2,335	2,956	7,012	19,590	42,445	22,126	15,110	8,513
Insurance	-	-	-	540	540	1,030	932	834	2,796	3,484	6,522	2,853	3,158	2,394
Royalty (not included)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cash flow expenses	151	151	151	60,345	60,798	2,751	3,267	3,802	9,820	30,901	129,052	33,018	70,490	18,722
Discounted cash flow at 15%	114	99	86	29,997	30,296	1,189	1,228	1,243	3,660	6,469	7,641	448	444	60
Discounted cash flow at 10%	125	113	103	37,474	37,815	1,552	1,676	1,776	5,004	10,586	18,299	1,752	2,207	372
Ash Handling														
Capital investment	-	-	-	2,835	2,835	-	-	-	-	845	4,246	845	1,938	845
Direct operating costs	-	-	-	50	50	265	265	265	795	2,778	6,019	3,241	2,315	1,852
Insurance	-	-	-	26	26	51	47	44	142	258	725	485	437	356
Royalty (not included)	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Cash flow expenses	-	-	-	2,911	2,911	316	312	309	937	3,881	10,990	4,571	4,690	3,053
Discounted cash flow at 15%	-	-	-	1,447	1,447	137	117	101	355	835	674	64	31	10
Discounted cash flow at 10%	-	-	-	1,808	1,808	178	160	144	482	1,354	1,594	246	151	61

Coal Handling Cost

	$/ton	¢/10^6 Btu
15% discount rate	1.66	14
10% discount rate	1.36	11

Ash Handling Cost

	$/ton	¢/10^6 Btu
15% discount rate	0.12	1
10% discount rate	0.10	1

TABLE XXXIII

ROM COAL PRODUCTION COST (INFLATED) - MINE, COAL AND ASH HANDLING
($ 10^3)

Category	Stage 1	Stage 2				Stage 3	Stage 4	Stage 5	Stage 6	Stage 7
	1978-84	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Coal production, 10^6 ROM tons	-	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42	21
Mean inflation factor (1976-77 = 100)	-	170	178	187	178	223	359	578	773	962
Direct operating costs	-	11,612	14,082	16,265	41,959	146,373	350,626	231,189	197,204	132,329
Depreciation	-	10,523	11,159	12,795	34,477	96,928	232,876	147,385	131,116	92,399
Interest on outstanding investment	-	13,149	13,090	12,953	39,192	76,969	142,729	62,340	41,245	20,128
Insurance	-	2,629	2,618	2,591	7,838	15,394	28,546	12,468	8,249	4,026
Sub-totals - uninflated	-	37,913	40,949	44,604	123,466	335,664	754,777	453,382	377,814	248,882
- inflated	-	64,452	72,889	83,409	220,750	750,348	2,723,975	2,622,605	2,921,324	2,377,520
Royalty - uninflated (75 ¢/long ton, 67 ¢/short ton)	-	1,809	3,618	5,427	10,854	43,416	91,455	44,488	28,140	14,070
Total cost/year or stage	-	66,261	76,507	88,836	231,604	793,764	2,815,430	2,667,093	2,949,464	2,391,590
Average cost, $/ton	-	24.54	14.17	10.97	14.30	12.25	20.63	40.17	70.23	113.89
Average cost, ¢/10^6 Btu	-	206	119	92	120	103	173	338	590	957
Coal Handling										
Production cost	-	12,823	12,847	12,879	38,549	70,014	145,538	73,683	58,656	42,559
Inflated production cost	-	21,799	22,868	24,084	68,751	155,561	529,101	422,527	457,078	407,402
Average cost, $/ton	-	8.07	4.23	2.97	4.24	2.40	3.88	6.36	10.88	19.40
Average cost ¢/10^6 Btu	-	68	36	25	36	20	33	53	91	163
Ash Handling										
Production cost	-	760	737	715	2,212	5,458	12,824	7,472	5,884	4,746
Inflated production cost	-	1,292	1,312	1,337	3,941	12,171	46,604	43,175	45,496	45,639
Average cost, $/ton	-	0.48	0.24	0.16	0.24	0.19	0.34	0.65	1.08	2.17
Average cost, ¢/10^6 Btu	-	4	2	1	2	2	3	5	9	18

TABLE XXXIV

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
	1979-80	1980-81	1981-82	1982-83	1983-84	Total	1984-85	1985-86	1986-87	Total	1987-93	1993-06	2006-13	2013-18	2018-22
Capital investment	10,614	17,190	33,661	30,383	55,444	147,292	18,630	1,885	19,335	39,850	87,501	209,525	127,989	117,722	54,380
Mean inflation factor	131	140	147	154	162	143	170	178	187	178	223	359	578	773	962
Direct operating costs	-	-	-	-	-	-	11,612	14,082	16,265	41,959	146,373	350,626	231,189	197,204	132,329
Insurance	-	-	-	-	-	-	2,629	2,618	2,591	7,838	15,394	28,546	12,468	8,249	4,026
Sub-totals - uninflated	10,614	17,190	33,661	30,383	55,444	147,292	32,871	18,585	38,191	89,647	249,268	588,697	371,646	323,175	190,735
- inflated	13,904	24,066	49,482	46,790	89,819	224,061	55,881	33,081	71,417	160,379	557,473	2,133,586	2,155,402	2,485,057	1,816,537
Royalty - uninflated	-	-	-	-	-	-	1,809	3,618	5,427	10,854	43,416	91,455	44,488	28,140	14,070
Cash flow expenses - inflated	13,904	24,066	49,482	46,790	89,819	224,061	57,690	36,699	76,844	171,233	600,889	2,225,041	2,199,890	2,513,197	1,830,607
Discounted cash flow at 15%	12,091	18,194	32,534	26,750	44,649	134,218	24,939	13,795	25,120	63,854	120,834	120,252	28,829	14,570	5,690
Discounted cash flow at 10%	12,640	19,879	37,161	31,958	55,778	157,416	32,537	18,827	35,886	87,250	200,135	295,885	113,537	74,607	35,678
Coal production, 10^6 tons	-	0.1	0.2	0.4	0.4	1.1	2.7	5.4	8.1	16.2	64.8	136.5	66.4	42	21
Discounted coal production at 15%, 10^6 tons	-	0.08	0.13	0.23	0.20	0.64	1.17	2.03	2.65	5.85	13.37	8.38	0.92	0.24	0.07
Discounted coal production at 10%, 10^6 tons	-	0.08	0.15	0.27	0.25	0.75	1.52	2.77	3.78	8.07	21.95	19.83	3.56	1.24	0.43
Coal Handling															
Cash flow expenses	-	151	151	151	60,345	60,798	2,751	3,267	3,802	9,820	30,901	129,052	33,018	70,490	18,722
Cash flow expenses - inflated	-	211	222	233	97,759	98,425	4,677	5,815	7,110	17,602	68,367	464,963	189,982	525,575	176,656
Discounted cash flow at 15%	-	160	146	133	48,596	49,035	2,022	2,186	2,324	6,532	14,103	25,286	2,528	3,295	559
Discounted cash flow at 10%	-	174	167	159	60,708	61,208	2,638	2,983	3,320	8,941	23,187	62,182	9,941	16,409	3,497
Ash Handling															
Cash flow expenses	-	-	-	-	2,911	2,911	316	312	309	937	3,881	10,990	4,571	4,690	3,053
Cash flow expenses - inflated	-	-	-	-	4,716	4,716	537	555	578	1,670	8,524	39,301	26,168	34,844	28,785
Discounted cash flow at 15%	-	-	-	-	2,344	2,344	232	209	189	630	1,803	2,197	353	223	92
Discounted cash flow at 10%	-	-	-	-	2,929	2,929	303	285	270	858	2,939	5,345	1,383	1,112	573

Uniform Selling Price

	\$/short ton	¢ 10^6 Btu
15% discount rate	16.57	139
10% discount rate	17.28	145

Coal Handling Cost

	\$/short ton	¢ 10^6 Btu
15% discount rate	3.44	29
10% discount rate	3.32	28

Ash Handling Cost

	\$/short ton	¢ 10^6 Btu
15% discount rate	0.26	2
10% discount rate	0.27	2

TABLE XXXV

COMPARISON OF OPENPITS NO 1 AND 2

	Openpit No 1 (Report No 2) March 1976	Openpit No 2 (Report No 3) June 1976	Openpit No 1 (Report No 9) March 1977	Unit
Minimum cover:	0	25	0	ft
Maximum vertical height of pit slope:				
Life of 2,000-MW power plant	-	-	1,250	ft
600-ft pit	1,150	2,100	1,600	ft
1,500-ft pit	2,500	3,000	3,100	ft
Elevation of pit floor:				
Life of 2,000-MW power plant	-	-	2,400	ft
600-ft pit	2,400	2,900	2,400	ft
1,500-ft pit	1,500	2,000	1,500	ft
Area of excavation:				
Life of 2,000-MW power plant	-	-	2,000	acres
600-ft pit	2,000	4,000	2,300	acres
1,500-ft pit	5,000	10,000	5,100	acres
Approximate maximum area of disturbance:				
600-ft pit	8,000	20,000	8,000	acres
Rom coal reserves within:				
Life of 2,000-MW power plant	-	-	360	10^6 short tons
600-ft pit	385	664	377	10^6 short tons
1,500-ft pit	775	3,397	667	10^6 short tons
Total waste rock within:				
Life of 2,000-MW power plant	-	-	921	10^6 byd^3
600-ft pit	885	2,176	1,132	10^6 byd^3
1,500-ft pit	3,647	10,653	4,607	10^6 byd^3
Overall stripping ratio:				
Life of 2,000-MW power plant	-	-	2.6	byd^3/short ton rom
600-ft pit	2.3	3.3	3.0	byd^3/short ton rom
1,500-ft pit	4.7	3.1	6.9	byd^3/short ton rom
Instantaneous stripping ratio at pit limits:				
Life of 2,000-MW power plant	-	-	12.2	byd^3/short ton rom
600-ft pit	7.7	11.0	23.8	byd^3/short ton rom
1,500-ft pit	13.7	15.4	29.2	byd^3/short ton rom
Capital investment to start-up (2,000-MW power plant)	134	291	209	$ 10^6
Uniform selling price (max) (Life of 2,000-MW power plant)				
10% discount	5.63	9.10	8.39	$/short ton rom
15% discount	6.35	11.17	9.77	$/short ton rom
On thermal basis:				
10% discount	51	83	70	¢/10^6 Btu
15% discount	58	102	82	¢/10^6 Btu