PD-NCB Consultants Limited

In association with

Wright Engineers Ltd & Golder Associates

DRAFT

Report No. 3

Preliminary Report on Hat Creek Openpit No. 2

Volume I

to

British Columbia Hydro and Power Authority

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CHAPTER I

INTRODUCTION

1. This report deals with a conceptual mine, "Openpit No 2", situated in Area 2 of the Hat Creek coal deposits. As for Openpit No 1, two phases are considered, ie down to the 2,900-ft level (600-ft pit) and subsequently down to the 2,000-ft level (1,500-ft pit). As a result of the higher elevation of this part of the valley, the floors of these conceptual pits are 500 ft higher in elevation than in Openpit No 1. It must be emphasised that these levels have not been selected on any firm basis. However, it is considered that the 600-ft pit is technically feasible whereas the 1,500-ft pit will require far more knowledge than is available at present to prove the concept. Unlike Openpit No 1, considerable resources of coal lie deeper than the bottom of the 1,500-ft pit.

TERMS OF REFERENCE

2. Report No 1 (which is included as Appendix "A" in Report No 2), gives the full terms of reference. Openpit No 1 is dealt with in Report No 2. This study is intended for comparison with Openpit No 1. However, it must be assumed that both pits will eventually be worked and therefore the conceptual design of Openpit No 2 takes into account that of Openpit No 1. In particular, this principle is applied to spoil disposal, ie spoil will not be dumped within the surface intercepts of either of the 1,500-ft conceptual pits.

FORMAT

3. This report follows as closely as possible the format of Report No 2 so as to facilitate direct comparison as far as possible. There are clearly many elements common to both openpits and these are not repeated, attention being directed to differences. In order to avoid possible confusion, appendices,

tables and plates have been numbered consecutively with those in Report No 2, except where revised, when a suffix R has been added to the original letter or number.

PROGRESS TO DATE

4. The draft of Report No 2 (Openpit No 1) was presented in Vancouver in March 1976 both to BC Hydro and the Provincial Department of Mines and Petroleum. As a result, a number of corrections and additions have been made and the final version was completed in June 1976.

5. On 18th March, 1976, Messrs Brealey and Alexander visited the Hat Creek valley to examine the mine and spoil disposal sites and particularly to observe thaw conditions. The drilling programme in Area 2 had been completed. Discussions took place on the recommendations for further investigation and test work and it is understood that a further drilling programme and geotechnical investigation may soon be approved.

6. The documentary information received since 23 February, 1976 is listed in Appendix "F". This includes corrections to the inclined boreholes and also the borehole water levels recorded by DCA.

PROGRESS REPORTS

7. Monthly Progress Reports No 6, 7 and 8 were submitted on 29th March, 28th April and 3rd June, 1976.

BASIC DATA

8. Table I has been revised as Table IR to include additional assumptions regarding marble and volcanics.

ACKNOWLEDGEMENTS

9. The continued interest and encouragement of BC Hydro is acknowledged with thanks as is also the reception accorded

by the British Columbia Department of Mines and Petroleum Committee under the chairmanship of the Deputy Minister. The continued co-operation of Dolmage Campbell Associates is also acknowledged with thanks.

CHAPTER II

GEOLOGICAL AND GEOTECHNICAL ASSESSMENT

INTRODUCTION

1. The borehole logs for Area 2 have been re-examined to enable the preparation of new plans and sections for conceptual-mine-design purposes. All available drilling information has been included and sections have been drawn up in the light of recent discussions with DCA and BCH. Use has also been made of earlier surface outcrop mapping of marble and the volcanic rocks.

2. The drill hole data all dates from 1975 or 1976 but because of the low drilling density compared with Area 1, the structural interpretation still includes large elements of conjecture. Plates 47 to 63 are the pertinent geological plans and sections: the amended legend of symbols and abbreviations as used in Report No 2 is shown on Plate 15R.

3. In contrast to Area 1, this deposit is long and narrow with boundary faults appearing to be an even greater constraint to the limits of the deposit than in the north.

STRUCTURE

4. The structure is essentially that of a horst-faulted anticline as shown on the plan on top of coal contours (Plate 63).

Faulting

5. As mentioned above, there are appreciable gaps in prospecting, drilling having been concentrated on EW lines at about 2,000-ft intervals. In the absence of frequent marker horizons, faults have been inferred on the basis of:

- non-systematic changes in coal roof elevation,
- disappearance of coal-bearing strata,
- sheared, broken or gouge-type materials in core.

On these grounds, little evidence is available to calculate the orientation of certain fault planes and a zero hade* (vertical fault plane) has sometimes been assumed. The principal faults appear to be:-

(i) Fault A - A vertical fault plane assumed with a down-throw to the west, trending NNW-SSE and comprising the western boundary of the coal deposit.

(ii) Fault Y - A vertical fault plane assumed with a possible down-throw to the east, trending NNW-SSE and converging towards Fault A in the north. This fault is inferred to constitute the eastern boundary of the coal deposit mainly at the deeper levels of the 1,500-ft (2,000-ft level) pit.

(iii) Fault X - A normal fault, down-throwing to the east and trending NNW-SSE. The hade is shown as approximately 30° and the fault plane effectively acts as the boundary of the coal deposit along much of the eastern side of the shallower 600-ft (2,900-ft level) pit. To the south, this fault lies east of the anticlinal axis but it runs along or crosses the fold axis to the north.

(iv) Fault J - An inferred normal fault down-throwing to the west and trending NW-SE between faults X and Y.

An additional normal fault down-throwing east may be present along the western boundary of Area 2, lying just east of fault A.

* angle of inclination measured from the vertical

6. The effect of this faulting is to form a horst between fault A on the west and faults X and Y on the east, a feature somewhat emphasised by the apparent anticlinal form of the deposit. In this respect, the area has some similarities with the horst lying between the Mag fault and fault H in Area 1. It can be seen from Plate 66 that the main faults in Area 2 when extended north lie to the west and south west of Area 1.

7. It is quite possible to interpret the existing data so that additional faults of various sizes are incorporated and that the hades and directions of fault displacements are changed. It is also possible to infer cross or oblique faults in Area 2 which run sub-parallel to Dry Lake, Trig and Finney faults. Whilst these possibilities must be recognised, there is little point in attempting complex structural solutions with the present density of drilling data.

Folding

8. As in the northern deposit, extensive areas of horizontal or gently inclined strata are not anticipated. Principal features are:-

(i) The basic structure appears to be an anticlinal horst, so strata dips to the west on the western side of the deposit and somewhat less markedly to the east on the eastern side. This structure is based on levels at the top of coal.

(ii) Dips on the western flank are steep, locally in excess of 60°. The dips appear to flatten to angles of 10° to 30° towards the axis of the anticline which locally coincides with sub-superficial outcrops. Further east towards and beyond fault X, dips are towards the east at angles of 15° to 30°.

(iii) At the southern and northern limits of Area 2, the crest of the anticlinal axis is thought to plunge beneath deeper cover. A small depression in the crest of the anticlinal axis also occurs between the two incrop areas.

9. Appreciable variations in the inclination of bedding traces have been recorded within individual boreholes. It is, therefore, possible that soft sediment structures such as slumping or compaction faulting are present in addition to possible diastrophic faulting referred to in para 5 and 6 above.

10. The west-east sections illustrate the above structures showing conjectural and inferred fault positions, and the inclination of the top of coal. Whilst it is possible to explain the disposition of coal in boreholes by faulting and folding, the possibility of rapid variations in strata, such as thinning or changes in sediment character, must not be overlooked.

MATERIALS

11. The overburden is divided as in Report No 2 into superficials and waste.

Superficials

12. This includes the drift deposits which, as in Area 1, comprise glacial tills and moraines and subsequent outwash materials, lake deposits and soliflucted debris. The superficials are usually thicker than in Area 1, ranging from less than 50 ft to over 250 ft. To the south east there are small rockhead outcrops, mainly of volcanic rocks, but the coal is nowhere exposed as further north. Logs of boreholes show the superficials to consist of moderately thick units (20 to 50 ft) of sands, gravels or clay with boulders. No attempt has been made to examine the spatial distribution of these different engineering soils. Patterns are apparent in the distribution of surface materials so that mudslides and alluvial deposits can be distinguished; similar patterns of till, moraine and outwash material are to be expected in the thicker, unexposed superficials. *Isopachytes of total superficial thicknesses have been prepared (Plate 59) but by virtue of the paucity of drilling are somewhat conjectural. The following trends are apparent:-

(i) Thinner superficials are present beneath Hat Creek, on the steeper slopes associated with the volcanic rocks and at some higher levels above 4,000 ft.

(ii) Thick superficials lie just east of Hat Creek and west of the volcanic rocks.

13. Sub-superficial contours intersect coal near the anticlinal axis to give two elongated areas of incrop as shown on Plate 61 in the northern and central sections of Area 2. The more northerly of these areas is the most shallow and hence the most appropriate access point on the basis of pre-stripping requirements.

Waste

14. The in situ contiguous strata overlying the coal appears to be similar to that in Area 1, ie low or very low strength siltstones and claystones, although one borehole reports medium strength sandstone near the roof of the coal. Outside the coal area, but within the excavated slopes, the sedimentary strata are similarly siltstones and claystones with a lower proportion of sandstones and conglomerates than in Area 1.

15. Volcanic rocks are present on the east side of the deposit and locally give rise to a terrace or bench-like feature, near or just below the 4,000-ft level. Tuffs and breccias are present probably with some clay-rich horizons; some of the samples collected from outcrops are of medium strength.

16. The volcanic rocks overlie both siltstones, claystones and coal. One borehole shows signs of coal burning which may be related to the volcanism, although only a very small part of the actual coal area is covered by the tuffs and breccias, which predominantly overlie the potential eastern slopes. Contours of the assumed base of volcanic rocks and their disposition with respect to the inferred faults are shown on Plate 62.

17. In the south eastern corner of the conceptual mine there is a prominent ridge of marble, considerably older and more indurated than the coal-bearing strata. The approximate position of the contact between the marble and the younger rocks is shown on Plates 61 and 62.

18. Plate 60 shows the isopachytes of total overburden (superficials, waste and volcanics).

Coal

19. The top of coal contours and subdrift outcrop positions are shown on Plate 63. A detailed study has not been made of variations in coal quality (notably the ash contents). Given the present drilling density only very general trends are apparent and these may not be substantiated by more detailed infill drilling. Some deterioration of coal towards the west is shown by a number of boreholes: the ash content of certain horizons appears to increase as do the size and number of inter-bedded claystone and siltstone partings. There also may be a similar southward deterioration. Individual boreholes often show a moderately low ash content (10% to 25%) near the top of the coal, a high ash (40% to 50%) mixed coal/mudstone central section and some reduction of ash content (15% to 40%) in the lower parts of the coal.

20. The internal correlation of the coal remains difficult: some correlation is possible on the basis of impersistent partings such as resin or tuff bands, but widespread diagnostic features observable in the field have not yet been recorded. Palynological studies may have some useful application on the large scale in assessing the position of major faults or significant sedimentary variations. No attempt has been made to show either variations in coal quality or actual correlation on plans or sections.

ADJACENT AREAS

21. Plate 66 shows the limits of 600-ft (2,900-ft level) and 1,500-ft (2,000-ft level) pits in both Areas 1 and 2. Spoil dumps must be located outside these areas and at present there is little information on the geology of the surrounding parts of the valley. The areas considered for dumps are:-

(i) South of Area 2

Between 20,000N and 35,000N coal is present in three of the seven boreholes drilled in this vicinity, suggesting a southward extension of faulted anticlinal structure, albeit with the top of coal at a lower level than in Area 2. With the exception of one borehole, the coal appears to be in mixed and interbedded shaly units and in all cases is relatively deep, circa 400 to 800 ft. Moreover, the valley narrows to the south and the coal lies beneath higher terrain, particularly on the east side of the valley, further increasing the potential stripping requirements and making it less attractive for openpit mining, and therefore more suitable for waste disposal.

(ii) West of Area 2

Coal has not been intersected in the few boreholes in this area; indications from geophysical investigations are that thick coal is either not present or at considerable depth in a down-faulted trough. Mudslides are apparently absent hereabouts and this location could be considered for waste dumps.

(iii) North-west of Area 2

Nothing significant is known of the sub-drift geology of this area. Conjectural extensions of the structure from further south suggests that coal, if present, is likely to be deep. Waste could be dumped here as was proposed in Report No 2.

(iv) North and north-east of Area 2

A narrow (1,200 to 1,500 ft) strip of land separates the two 1,500-ft pits, there being a 7,000 ft gap in drilling between Areas 1 and 2. It again seems possible that any coal in this area is deep, but the controlling structures and sedimentary variations are not yet understood. An area for dumping along and east of Ambusten Creek would appear to lie outside the main potential coal areas and to be away from the principal mudslides.

(v) East of Area 2

Coal is seemingly absent or beneath a volcanic and sedimentary cover of several thousand feet. The increasing elevation and steepening slopes on this side of the valley render the area unsuitable for much waste disposal.

GEOTECHNICAL IMPLICATIONS OF THE GEOLOGY

22. The mining implications regarding the above findings are:-

(i) A 600-ft pit would primarily remove the upper coals, probably of moderate quality. A 1,500-ft pit might include a higher proportion of silty mixed coal.

(ii) The segregation of waste during mining will present similar problems to those likely in Area 1.

(iii) Larger gaps are present in prospecting than in Area 1. Structures and estimates of volumes cannot, therefore, be used in conceptual planning with the same degree of confidence.

(iv) As with Area 1, the diggability and trafficability of pit materials cannot be fully assessed on available information.

23. A range of potential slope failures is likely within the pit. The low strength claystones and siltstones are present as in Area 1 and a conservative slope of 15^{o} to 16^{o} has again been used in the conceptual layout for the excavations in both coal, stratified overburden and superficials.

24. Observations on core by Golder Associates Ltd (GA) suggest that the claystones and siltstones can be expected to behave as engineering soils in slopes of significant height, ie circular-type failures might be anticipated. Simple field

tests indicate compressive strengths of the order of 2,000 to 3,000 psi for the overlying strata.

25. Bench stability with steeper slopes is likely to be controlled by discontinuities such as faulting or bedding separation surfaces. In this respect the anticlinal structure may be considered more favourable than in Area 1 since the dip of the coal and stratified overburden should, in most places, be cut by the pit slope at an angle of approximately 90^{o}.

26. GA have presented a revised distribution of mudslides in Area 2. Some of the areas first considered to be mudslides are now thought to be alluvial fans, and the mudslide/mudflow boundaries have been redefined, slightly enlarged and limited to the eastern side of Hat Creek (Plate 64). The two principal slide areas are near Fish Hook Lake and opposite McDonald Creek. The latter lies entirely within the 600-ft pit, whilst the former extends into the 1,500-ft curtilage.

27. In the mining proposals for Area 1, all pit slopes progress gradually outwards except those near the ramp. In Area 2 the proposals are for progress towards the south with smaller lateral expansions. Long-term deterioration of the north-south slopes, therefore, becomes an important consideration especially as time-dependent movements could prejudice bench conveyor systems or haulage. If Area 2 is to be worked, this matter must receive due consideration.

28. No preliminary observations have been made by GA on the likely strength of the volcanic rocks. Samples collected in the field are of moderate strength but some core shows signs of breakdown on exposure, commonly found with such materials. On the basis that the volcanics may include both very weak and relatively competent rocks, and that it does not appear to have been incorporated in major faulting, an operating slope angle of 25^{o} has been used for excavations in volcanics when assessing volumes etc. Considerably more data are required both to validate this angle and to assess diggability etc.

29. A slope angle of 45^{o} has been assumed for the marble where encountered in the south east of Area 2. On the basis of nearby natural slopes, this appears to be a reasonable assumption, but further investigation would be necessary if Area 2 were to be worked south of 50,000N.

30. In most respects GA consider that the slopes could not be excavated to steeper angles than those proposed in Area 1. Similarly, their findings presented in Report No 1 and summarised in para 16 and 17 of Chapter II, Report No 2 are generally upheld.

31. Observations on water in Area 2 are limited. Packer tests on borehole 76-118 showed the coal to be impermeable. Rest water levels in open boreholes in Area 2 indicate water within 10 or 20 ft of the surface. Several boreholes had collapsed at, or below, rockhead and these rest water levels, therefore, probably reflect conditions in the superficials. More investigations of deep ground water conditions are required.

<u>FURTHER INVESTIGATIONS</u>

32. Even a cursory inspection of Plate 64 showing drilling progress to February 1976 reveals a low density of drilling in Area 2. 52 boreholes have been drilled within the limit of the 600-ft pit; five extra boreholes within the 1,500-ft pit. This represents one borehole for every 330,000 yd^2 (70 acres) in the 600-ft pit and less than half that density for the deeper pit. On the basis of potential in-situ tonnage per foot of borehole in coal, the figures are:-

600-ft pit	- approx 90,000 tons/ft of borehole in coal
1,500-ft pit	- approx 160,000 tons/ft of borehole in coal
Typical coal stripping operation	- 10,000 tons/ft of borehole in coal

33. Should Area 2 be seriously considered for development, further drilling is essential to remove doubts regarding the structure, nature of constituent rocks and superficials and geotechnical matters, especially groundwater conditions in the consolidated strata. The spacing of boreholes along existing section lines is satisfactory in most cases. The section lines are, however, widely spaced and the existing drilling concentrates on the coal area with little drilling in the pit slopes. The following additional drilling would improve the confidence of geological and geotechnical predictions in a similar fashion to that suggested in Area 1:-

Proposed Drilling	Footage
(i) Additional 300-ft boreholes at 1,000-ft centres along current WE sections mainly within the pit slopes	10,000 ft
(ii) Additional 600-ft boreholes along WE section lines at 600-ft intervals NS and WE	70,000 ft
(iii) Additional 300-ft boreholes within the pit slopes on the section lines of (ii) above, with some further allowance for the siting of spoil dumps	20,000 ft

34. This extra drilling replaces that proposed in Chapter II of Report No 1 and reduces to 15,000 tons the potential proved per foot of borehole in coal. As mentioned previously, future infill drilling need not be all core drilling; much use should be made of the infill programme for geotechnical purposes. Such a sizeable prospecting programme could be spread over several years and adjusted to optimum slopes and depths of working as these become apparent.

CHAPTER III

MINE PLANNING

GENERAL

1. The same considerations regarding valid mine planning apply as in the case of Openpit No 1. Area 2 is a much larger deposit than Area 1 and the intensity of exploration to date is even less. However, the rock types are similar. The extent of the volcanics is greater and the pit would widen out into the marble on the E side of the valley.

Structure

2. The structural features of the Area 2 deposit are discussed in Chapter II and illustrated on Plates 15R and 47 to 64 although the locations of the faults are somewhat conjectural. The limits of the deposit to the N and S are not known although coal is shown in some boreholes. In the north it is deep and may well be contiguous at depth with the Area 1 deposit. In the south, care will have to be exercised that mineable coal is not covered with spoil dumps. The deposit is narrower, longer and deeper than Area 1 and the thickness of overburden is greater.

3. As in the case of Openpit No 1, the stratigraphy of the coal itself is not determined and the amount and configuration of intercalated waste is not known. Therefore, the same assumptions have been made as for Openpit No 1.

Coal Quality

4. Coal quality aspects are dealt with in Chapter V and, owing to the lack of data, similar assumptions have been made as in the case of Openpit No 1. However, a check analysis has been carried out and some washability results for one sample only plotted. Taking these results as typical, the effect of coal preparation on coal and waste production has been calculated.

Coal Production

5. The same assumptions have been made as in the case of Openpit No 1 as regards the quantity and quality of the rom coal production, ie:-

Annual rom coal production	-	13,100,000 short tons
Ash content	-	32% (includes waste dilution)
Moisture content	-	20%
Calorific value	-	5,500 Btu/lb

Physical and Chemical Properties

6. The same situation applies as for Openpit No 1.

Groundwater

7. The results of logging the standby water level in some of the boreholes have been received and the conclusions which can be drawn are discussed in Chapter II.

TYPE OF MINE

Underground Mining

8. The preliminary appraisal of the possibilities of underground mining apply as for Openpit No 1 but the coal is deeper and hence the costs would be expected to be greater. However, it is evident that there are considerable resources of deep coal (many of the deep boreholes in fact terminated in coal) and therefore there may be a greater incentive to develop a feasible underground mining method. The remainder of this report, however, deals with a conceptual surface mine.

Factors Controlling the Design of a Surface Mine

9. In general, the same geotechnical factors apply as in the case of Openpit No 1. However, the effect of the presence of substantial deposits of volcanics and marble on the east side of the valley is discussed in Chapter II. Since these rocks are stronger than the claystones, greater angles of slope can be accepted and therefore the following have been adopted:-

Volcanics	-	25^{o}
Marble	-	45^{o}
Other rocks	-	15^{o} 57'

10. The conclusions which have been drawn from consideration of the control factors are generally similar to those applying to Openpit No 1. However, the more elongated shape of the deposit considerably influences the pit design. It may well be possible to commence backfilling before the pit is completely worked out. Also, the much larger quantity of superficials and the long, straight faces which could be formed make it possible to reconsider the use of a bucket-wheel excavator and conveyor system for the excavation and disposal of this material as an alternative to the scheme adopted for Openpit No 1, ie scraper operation (see Chapter IV).

Main Incline

11. The reasons for the adoption of a main incline equipped with conveyors in Openpit No 1 remain valid for Openpit No 2. Again, the north end of the deposit is the most favourable location since the cover is least at this point and therefore excavation is minimised. The amount of coal underlying it is, however, greater than in the case of Openpit No 1, but again much of this coal could ultimately be recovered. This location is also favourable as regards those power plant sites which are at the northern end of the valley. If a site at the southern end of the valley were to be selected, this location would be reviewed but even then it seems likely that it would be retained as opening up at the southern end of the deposit would be much more expensive due to the thicker overburden.

12. The direction of the incline has been selected so that it is pointed along the axis of the deposit thereby possibly avoiding the necessity for conveyor transfer points at the bottom (due to change of direction). However, the direction could be adjusted to suit the surface layout if necessary, eg if the Harry Lake power plant site were selected.

Depth Limitation - Reserves

13. The same policy regarding the depth of the pit has been adopted as in the case of Openpit No 1, ie a 600-ft pit has been postulated for detailed examination and a 1,500-ft pit projected without, however, any commitment as regards its technical and economic feasibility. Owing to the higher topography, the floors of these pits are at higher elevations than in the case of Openpit No 1, ie 2,900 ft for the 600-ft pit and 2,000 ft for the 1,500-ft pit.

14. In this case the reserves of coal within the confines of the fully developed 600-ft pit are more than adequate for the 35-year life of the power plant, ie 664 million short tons (Table XXIX). The 1,500-ft pit is estimated to contain 3,397 million short tons (Table XXIX) and coal is known to extend at least 450 ft below that level as well as laterally.

Pit Design

15. A manual method of designing the pit similar to that used for Openpit No 1 has been adopted.

16. The steps taken in establishing the design shown on Plate 66 are as follows:-

(i) Direct the access incline approximately along the central axis of the deposit, ie towards the centre of gravity.

(ii) Roughly equalise the waste excavation on the east and west sides of the initial pit.

(iii) Draw a conical-shaped pit centred on the incline to a floor elevation of 3,000 ft (Stage 1).

(iv) Extend the pit down the incline and sideways to the full depth of the 600-ft pit, ie to floor elevation 2,900 ft (Stage 2).

(v) At this point alternative approaches are possible ie:-

Scheme A - Widen the pit to include most of the coal above the 2,900-ft elevation and then extend it to the south of the deposit.

Scheme B - Maintain a narrow pit, extend it to the south and then widen out on the east and west sides to the limit..

These two schemes are illustrated on Plate 65. Scheme A is economically less favourable than Scheme B as it involves the removal of more waste rock at an earlier date. Also, as it progresses to the south, static benches would be left behind and these would be vulnerable to long-term slope failure. This could be mitigated only by abandoning the northern access incline (after say 20 years) and developing another incline further to the south (as shown on Plate 65). However, this could be turned to advantage as the abandoned part of the pit could be utilised for spoil or ash disposal, at the expense, of course, of abandoning the deeper coal.

Scheme B enables the removal of some of the massive waste rock on the east and west sides to be deferred and therefore the cash flow would be more favourable overall although the economic cut-off would be earlier. The southern half of the pit could be slowly widened and the long north-south faces would be kept active thereby avoiding long-term slope failure. In other words, the faces would be cleaned up from time to time. Again it might prove advantageous to abandon the northern access incline and to open another further to the south when the bulk of the excavation is in that area.

(vi) The development of the access incline occurs in Stage 1 and considerably greater quantities of overburden have to be removed than in the case of Openpit No 1. Therefore, the stripping ratio is greater. The coal excavated during this stage would again be stockpiled.

(vii) The same method of calculating the instantaneous stripping ratio has been used as in Report No 2 for Openpit No 1 (Chapter III, para 23). Table XXIX gives the volumes of different types of waste rock and the stripping ratios based on both in-situ and rom quality coal (compare Table II in Report No 2). Again the sum total of all the volumes is, of course, the total volume of the pit up to the stage in question, that of the coal being the mineable reserves. The "stage stripping ratio" has been used for the economic calculations which have, therefore, been averaged over the stage. The instantaneous stripping ratio occurs at the end of the stage and would be the value to be used for the calculation of the economic cut-off, ie the last incremental cut on each bench. (In an entirely symmetrical operation which expands outwards uniformly, the stage ratio would clearly be of a value between the instantaneous ratios at the beginning and the end of the stage because the instantaneous ratio would increase uniformly cut by cut. However, this assymetrical design this is not the case.)

Nine stages are shown for Openpit No 2 (Plate 66), the first six providing sufficient coal for the power static Stages 7 and 8 show the the further development of the 600-ft pit and Stage 9 shows the 1,500-ft pit. It will be noted that the total reserves of coal within this pit are approximately 3,397 million tons rom compared with 775 million tons for Openpit No 1 (Table II).

Plate 67 shows the cumulative volumes of waste plotted against the cumulative tonnage of in-situ coal mined out. The corresponding curves for Openpit No 1 are shown for comparison.

(viii) Table XXX, Schedule of Production, has been derived from the schedule of coal production required by the power station (three 750-MW generators) in the same way as for Openpit No 1 and Plates 68 and 69 show the yearly and cumulative coal and waste production requirements and the yearly stripping ratio (relative to rom coal). The corresponding curves for Openpit No 1 are shown for comparison.

Development Programme

17. Assuming the same timetable for the power station, the same construction schedule for the power station and development schedule for the mine apply as for Openpit No 1 (see Plate 23, Report No 2).

ENVIRONMENTAL ASPECTS

18. As regards environmental aspects, the same considerations apply as for Openpit No 1 but the volumes of waste for disposal are considerably greater, particularly if an attempt is to be made to recover most of the coal reserves. Consequently, the dumps will occupy a greater area. However, unlike Openpit No 1, there is a possibility of backfilling in the pit before mining operations cease altogether. Clearly, if Openpit No 1 is worked out first, then that volume would be available for dumping (and vice versa). Openpit No 2 is further up the valley which is also wider at this point and so the north end of the valley would be largely unaffected whereas a pit at the north end of the valley is bound to have a major effect on the valley as a whole.

CHAPTER IV

MINING OPERATIONS

INTRODUCTION

1. No 2 deposit lies to the south of Openpit No 1 and is different in shape having a greater length along the north-south axis than width over the east-west axis. This has resulted in the different method of opening up and working the coal in six arbitrarily chosen stages along the length of the deposit in a southerly direction.

2. Because of the thickness of overburden, the development rate at the start has had to be greater and because of the shape of the deposit and topography of the valley a constant annual rate of waste removal of 27 million bank yd^3 of total waste is deemed necessary.

3. In order to allow direct comparison of the two deposits, the machinery used and the type of mining are the same although the quantities and the distances involved in overburden and coal removal are different.

4. The difference in shape makes Openpit No 2 more suitable for a bucket-wheel excavator system to remove superficials and details of this type of operation are considered at the end of this chapter.

5. As the mining operations are similar, this chapter will only examine areas of difference.

6. Production schedules are detailed in Table XXX. Equipment required in shown in Table XXXI.

DEVELOPMENT

7. Stage 1 of the operation is completed before full production starts in Stage 2. The 2,400-ft elevation, which is approximately 600 ft below the surface level, is reached at the end of Stage 2.

DIVERSION OF HAT CREEK

8. The river is dammed at the southern end of the deposit and is channelled along the western side of the valley. The topography permits natural drainage so that pumping from the reservoir behind the dam is not needed.

SUPERFICIALS

9. These will be removed as described in the report on Openpit No 1 except that the scrapers will require to climb grades of up to 15%. This is reflected in the large number of scrapers needed throughout the removal of superficials.

10. From 1993 onwards, the superficials will be transported to the south of the deposit by disposal conveyors and scrapers will deliver the superficials to the conveyors. The arrangement for this conveyor would be similar to that shown on Plate 71 in connection with the bucket-wheel excavator. The loading point would be outside the area of the proposed 35-year pit.

VOLCANICS

11. From 1992 onwards volcanic rocks occur in the area of operations to the south east. These will be blasted and removed by shovel and trucks.

DRILLING AND BLASTING

12. The coal and pit waste will be drilled and blasted as in Openpit No 1.

13. The volcanics will be drilled using blast-hole drills with approximately 10-in diameter holes at an interval of 6 yd. For calculation purposes, a drilling rate of 30 ft/hour has been estimated for this type of drill in hard volcanic rocks. This compares with 210 ft/hour for the drilling rate in the softer waste and coal with 4-in diameter holes and crawler rigs. A powder factor of 0.6 lb/ton has been used for volcanics.

TRANSPORT

14. Plate 70 shows the mean haulage distance for removal of the four types of material for 1979/80 until 2019/20. It shows the reduction in distance for superficials removal by introducing the south conveyor.

WASTE DISPOSAL

15. This is dealt with in detail in Chapter VI.

MUD FLOWS

16. Plate 64 shows the mud flows in the vicinity of No 2 deposit. These mud flows are to the east of the deposit and the total volume assuming a thickness of 48 ft is 81 million bank yd^3 of which 37 million bankyd^3 are within the area of the planned pit.

DRAINAGE AND PUMPING

17. The increased size of the pit increases the quantity of water to be pumped as a result of the annual precipitation to approximately 2,000 imperial gallons/minute over the year after Stage 5 has been reached. The installation of adequate pumping facilities has been included in all estimates of equipment required.

EQUIPMENT

18. Table XXXI details the equipment required taking the actual working period for a machine as 5,000 hours/year. The capital and replacement costs for all equipment are summarised for stages and start up years in Table XXXII.

19. The replacement period for machinery is as given in the report on Openpit No 1 to which should be added:-

(i) Every 20 years - Bucket wheel excavator (alternative superficial removal scheme)

(ii) Every 10 years - Blast-hole drills

EQUIPMENT COSTS

20. In addition to Table XXXII, the allocation of equipment costs by activity is shown in Table XXXIII. Table XXXIV is a schedule of typical equipment (the manufacturers' names given do not imply any preference over other makes).

BUCKET-WHEEL EXCAVATOR

21. The use of a bucket-wheel excavator system has been considered as an alternative to scrapers for the removal of superficials.

Mining Method

22. The system of excavation involves stripping the superficials in blocks 2,250 ft wide across the pit starting from the centre and working outwards. Plate 71 shows the areas blocked out in sequence and Plate 72 shows the estimated cross sectional areas of superficials on the sections indicated.

23. Two faces would operate simultaneously working from the centre outwards on lines parallel to the estimated strike lines of the base of the superficials to produce as far as possible a face of even height. Excavation would commence at the estimated volumetric centre so that excavation of the two sides of the block would be completed in similar times.

Transport of Waste

24. Waste would be removed by a belt-conveyor system comprising two movable conveyors (one in each bench moving up with the excavator) delivering on to two cross conveyors sited on the unworked portion of the pit. These, in turn, would deliver on to a belt-conveyor system to transport the spoil to the southern end of the area for disposal by means of a boom stacker. A diagrammatic sketch of the layout is shown on Plate 73.

25. As each block is excavated, the cross conveyors move forward and the waste conveyor is shortened by the width of the block (2,250 ft). The conveyor made available can then be used

to extend the other end of the conveyor as the stacker completes the spoil benches and moves forward.

Restraints

26. Successful removal of superficials is dependent on the number and size of boulders that are encountered. Bucket-wheel excavators cannot handle very hard material or lumps too large to pass through the buckets and presumably too large to be carried on the associated conveyor system. Occasional boulders can be blasted or removed by shovel but large numbers would interfere with the operation to an extent which would render it uneconomic.

Extraction Rates

27. It is estimated that an extraction rate for superficials of 20 million bank yd^3 a year would enable the mining programme to be followed. The time schedule is shown on Table XXXV.

ECONOMICS

28. The cost of removal is estimated in Table XXXVI. Two costs are shown, one for removal and dumping as a complete operation and the other for comparison with scraper operations in excavating the removal to the pit perimeter only as both systems rely on the same conveyor system from the perimeter to the dumping ground.

29. The approximate direct operating cost for scrapers is 61¢/bank yd^3 to the dump or 46¢/bank yd^3 to the conveyor.

30. Table XXXVII shows a DCF calculation (at 15%) which compares superficials removal by scrapers and by bucket-wheel excavators. This shows that bucket-wheel excavators would be 43¢/short ton cheaper than scrapers for this work.

ADVANTAGES AND DISADVANTAGES OF BUCKET-WHEEL EXCAVATORS

31. Table XXXVII shows that the initial capital cost of the BWE system to start-up of production would be approximately $55 million compared with $34 million for the scraper scheme. However, operating and maintenance costs for the BWE system would be less than for the scrapers. In other words the BWE system loads the costs at the front end.

32. Apart from the reduced operating cost, one of the main advantages of the BWE system would be the reduction in labour requirements. In 1980, for example, when the scrapers are scheduled to remove the same quantity as the BWE, 187 men would be needed for the scraper operation compared with 52 for the BWE system.

CHAPTER V

SURFACE PLANT AND COAL PREPARATION

INTRODUCTION

1. In the absence of a firm decision on the location of the power station, the surface plant and coal preparation have been considered to be the same for Openpit No 2 as detailed for Openpit No 1.

MINE POWER SUPPLY

2. Due to the elongated shape of the mine, power requirements for transport of materials are higher than for Openpit No 1. The main incline conveyors are longer than for Openpit No 1 at the same stage and from 1983, when Stage 2 reaches the pit bottom level, the main incline conveyors require more power than in Openpit No 1 to cater for the increased depth and length. As the mine progresses, extra conveyors are required on the floor of the pit and in addition the waste disposal system for superficials on the south side requires extra power because of the conveying distance.

3. There is no requirement for pumping for the Hat Creek diversion, as the water can flow by gravity round the proposed Openpit No 2.

4. The mining method calls for electric drills to drill the volcanics from 1992 onwards as an additional item.

5. The estimated ultimate loading would be about 30 MVA.

6. Plate 74 shows in diagrammatic form the proposed ultimate HV circuit for Openpit No 2. The HV circuit layout and power requirements would not be affected to any great extent if the mine access were moved to a position further along the pit at a later date (see Chapter III), the only major difference being the possibility of the use of overhead lines for feeding the conveyors on the surface in lieu of the cables used when in the pit.

STOCKPILING AND RECOVERY

7. The same layout as proposed for Openpit No 1 has been included. This will require re-assessment when the power station site is finalised.

COAL PREPARATION

8. This has been assumed to be as for Openpit No 1. However, the results of tests carried out at the NCB Yorkshire Laboratory, UK, have been compared with earlier test results and the conclusions are given below

Coal Washability Characteristics

9. "Washability" of coal is assessed in the laboratory by float and sink analysis in liquids of different specific gravities. It is, strictly speaking, a measure of the susceptibility of the coal to gravity separation. Heavy medium processes closely approximate to pure gravity separation whilst other processes are more influenced by other factors.

10. The washability characteristics of a typical Hat Creek coal are shown on Plate 75. These are based on two samples taken from borehole 75-74 which is located in the centre of the southern part of the No 2 coal deposit (co-ordinates 48,000' N and 24,428' E) (see Plates 51 and 64). The first sample was taken from footages 1,678 to 1,844 and was analysed by Loring Laboratories Ltd in October 1975. (Report No 10464, dated 1st October, 1975 and No 10635, dated 20th October, 1975 (Appendix "G").) This was just one of a large number of samples analysed by them. The second was taken from footages 1,678 to 1,710 and was analysed by the NCB Yorkshire Laboratory, UK in March 1976 (Appendix "H"). These samples should not be regarded as statistically "representative" but are "typical" of the Hat Creek coal. They are, however, considerably better than the rom quality assumed for this report which can be accounted for by the absence of any allowance for dilution with waste rock during mining.

11. In view of the number of variables involved, coal washability characteristics can be plotted in a number of ways. Plate 75 shows five such plots, all of which have been calculated on a dry basis, ie:-

(i) Cumulative Floats (Yield) v Specific Gravity of Floats

This plot shows the yield of below-gravity material (ie coal) which would be obtained when washing in a bath of liquid maintained at that particular specific gravity. The "gradual" shape of the curve indicates that the coal is difficult to wash, ie it contains substantial amounts of "middlings". An easily-washed coal is characterised by a sharp bend in this curve, ie at one point on it a small change in specific gravity results in a large change in yield. (For perfect washability the curve would be L-shaped.) The plot shows that a somewhat higher yield (cumulative floats) was obtained in the NCB analysis and this can be explained by the fact that this sample was crushed to -¼ in whereas the Loring sample was crushed to -½ in, ie better separation of the heavier and lighter fractions has been achieved.

(ii) Cumulative Ash in Floats v Specific Gravity of Floats

This plot shows the gravity at which the coal would have to be washed to obtain a given ash content in the washed product, eg at 1.6 the ash content would be 15%.

(iii) Cumulative Floats (Yield) v Cumulative Ash in Floats

This plot shows the yield which would, theoretically, be obtained for a given ash content, eg for 15% ash the yield would be about 70%.

(iv) <u>Cumulative Floats (Yield) v Cumulative CV of Floats</u>

This plot shows the yield which would be obtained for a given CV.

(v) <u>Cumulative Ash in Floats v Cumulative CV of Floats</u>

This plot relates the ash content to the CV of the product, eg at 15% ash the CV would be 10,500 Btu/lb (dry basis).

12. The good correspondence of the NCB and Loring curves (despite the difference in size), serves as a necessary check on the laboratory methods used.

13. Plate 75 shows the laboratory results of separating the material at a range of specific gravities and almost perfect separation can be assumed. Some coal is, however, lost at the lower gravities because it is intimately associated with ash in some of the particles. In a commercial plant, however, the loss of coal would be greater because of the imperfect separation and the extent of this can only be determined by pilot plant testing of the processes available. Therefore, all the conclusions drawn from these results are optimistic.

14. Plate 76(a) shows the sample adjusted for a notional 20% moisture and the positions A, B and C of the various coal qualities assumed. Good correspondence can be seen.

15. The object of washing the coal is, of course, to increase its calorific value by reducing the ash content and the washability tests give an indication of the results which could, theoretically, be achieved and also the amount of material which would be rejected by the washery. These losses must be compensated by mining more coal and clearly the disposal of the washery rejects is a major problem in itself. This is off-set by the reduction in fly ash. (Incidentally, the enforcement of strict particulate emission standards may, in the case of high-ash coals, make some form of beneficiation unavoidable.)

16. Using the washability test results (Loring), the coal qualities and quantities at different points in the system, ie in-situ, rom (washery feed) and washed (boiler feed), and the rejected tonnages have been calculated. It must be emphasised that these figures err on the optimistic side for the reasons given above.

<u>Birtley Assessment</u>

17. Birtley Engineering (Canada) Ltd carried out a preliminary assessment of the washability of some borehole samples of Hat Creek coal and reported in August 1975 and a comparison of the results with those given above indicates that the washability curves are similar but show somewhat higher yields. It is felt that this report is somewhat optimistic and may underestimate the difficulties of coal preparation, the coal losses resulting and the costs for the following reasons:-

(i) No allowance has been made for dilution of the rom coal with waste.

(ii) No reference is made to the presence of claystones which are known to cause considerable difficulty in a similar setting at Centralia, Washington, USA

(iii) De-sliming would result in loss of fine coal and considerable difficulty would be encountered in slimes treatment and disposal.

(iv) As a result of the above, the flow sheet suggested appears to be too simple.

<u>Coal Requirements for Different Degrees of Washing</u>

18. Chapter III gives the coal qualities and quantities assumed in this study. Using the same heat input to the power station, ie 144 x 10^{12} Btu per annum, the quantities required of coals of different calorific values can be approximately calculated. Table XXXVIII gives the results for boiler feed, washery feed (ie rom coal) and in-situ coal using the Loring test results and also the rom coal assumed as the basis of this

report. Plate 76(b) illustrates the effect of various degrees of washing on the Loring sample. Since these are expressed on a dry basis, all figures have been adjusted to a notional 20% moisture content. The effect of the moisture and ash on the boiler thermal efficiency has been ignored.

Waste Production

19. Table XXXIX summarises the resulting waste quantities, including pit rejects (segregated waste), washery rejects and boiler ash and it can be seen that washing to 15% ash would increase the total waste by 33% and to 10% by 128%. This table is based on the Loring above-average sample and the actual results could well be 20% worse.

20. These waste totals do not include overburden, the quantities of which are proportional to the in-situ coal production. Also there will be a more rapid depletion of reserves and a further penalty due to earlier advance into higher stripping ratios.

Moisture Content

21. All this analysis has been based on an assumed moisture content of 20%. The in-situ moisture content of both coal and waste is at present unknown. Unless core samples are hermetically sealed as soon as they are recovered, they will inevitably experience loss of moisture prior to testing. Even immediate sealing is not without its pitfalls as contact with the drilling mud affects the moisture content of the core. However, it is the best that can be done until pitting and bulk sampling is carried out. It is, therefore, strongly recommended that when the infill drilling is carried out, selected core samples should be set aside for this purpose and when bulk samples are being procured samples should be placed immediately in sealed drums for moisture determination.

Miscellaneous Characteristics

22. The tests carried out on the coal illustrate a number of other factors which have a bearing on its combustion

properties. The sulphur content is less than 1%, which is fortunate as it is mainly in the organic form and cannot, therefore, be removed by washing. The arsenic content is also low, a feature which is common when the pyritic sulphur level is low. As might be expected, the coal has no coking properties. The ash analysis shows high concentrations of silica and alumina. This corresponds with the observed high ash fusion temperature (initial deformation being over 2,500^{o} F. The silica ratio is high (Loring 77%, NCB 88%). This will result in a very viscous slag and therefore the fuel is more suitable for firing in the pulverised form than in a cyclone furnace arranged for liquid slag tapping. The Hardgrove grindability index (Loring 51, NCB - not measured) is average and should present no particular problems, particularly as the high volatility will make extremely fine pulverisation unnecessary. On this basis a target of 65% through 200 mesh should give acceptable levels of carbon in dust and grit.

Trace Elements

23. Coal samples taken from borehole 74-25 (Area 1 deposit) were analysed for trace elements by Mr. K. Fletcher, who reported on 2nd April, 1976. He concluded that the only trace elements which could cause environmental problems were copper and molybdenum, the former occurring in two samples in concentrations "comparable to those in many porphyry copper deposits", and the latter in concentrations "within the range associated with molybdenosis in cattle". The combustion process will clearly bring about further concentration in the ash. Therefore, if these values are widespread, consideration will need to be given to the burial of this toxic material (eg Montana may stipulate 8 ft of cover). Also the waste rocks should be tested to make sure that dangerous trace element concentrations do not occur in these.

SCHEDULE OF EQUIPMENT

24. Table XL summarises the Schedule of Equipment - Fixed Installations.

CHAPTER VI

WASTE AND ASH DISPOSAL

MATERIALS AND QUANTITY

1. From Openpit No 2 there are five types of waste to be dumped in the surrounding areas. These are:-

(i) superficials
(ii) pit waste
(iii) volcanics
(iv) segregated waste separated visually from the coal
(v) ash from the power station.

The quantities produced at each stage of the working of the deposit are shown in Table XLI. Specifically 1,660 million yd^3 of space are required up to the end of the 35-year pit (between Stages 5 and 6), 3,155 million yd^3 up to the end of the 600-ft pit (Stage 8) and 15,808 million yd^3 are needed up to the end of the 1,500-ft pit (Stage 9). Again these quantities may require to be modified after experience of working the deposit.

DUMPS

2. In the southern portion of the valley, there are four suitable dump areas designated No 3, 4, 5 and 6 dumps. These are shown on Plate 77 and the total space available to selected elevations is shown on Table XLII.

3. The comments in the report on Openpit No 1 apply also to the dumping for Openpit No 2 with the exception that the dump elevation in dumps 5 and 6 has been increased to 4,500 ft. This is possible because of the general increase in altitude of the valley in a southerly direction.

DISPOSAL AREAS

4. Space for dumps is considered under these different conditions:-

(i) For the 35-year pit

The areas suitable for disposal are shown on Plate 78 with details of volumes given in Table XLIII. The volumes to various elevations are:-

	10^6 yd^3
Dump No 4 (4,000-ft elevation)	1,080
Dump No 5 (4,100-ft elevation)	671
	1,751

(Volume required 1,660 million yd^3)

More than half of the superficials will be delivered to dump 5 by the south conveyor.

(ii) For the 600-ft pit

The areas suitable for disposal are shown on Plate 79 with details of volumes given in Table XLIV. The volumes available are:-

	10^6 yd^3
Dump No 4 (4,000-ft elevation)	1,080
Dump No 5 (4,350-ft elevation)	1,935
Dump No 6 (4,350-ft elevation)	402
	3,417

(Volume required 3,155 million yd^3)

(iii) For the 1,500-ft pit

The volume required in this case is estimated to be 15,808 million yd^3. The total space available in the vicinity of Openpit No 2 is 4,867 million yd^3. It is clear that waste disposal would be a major problem and it would be necessary to dump to higher elevations or to use areas in the north of the valley or outside the valley for this purpose.

5. In all three cases the extra space provided if backfilling were undertaken has not been included.

WASTE TRANSPORT

6. In general, arrangements for disposal of waste would be the same as for Openpit No 1 except that from 1993 onwards superficials would be delivered by the south conveyor to dump 5.

CHAPTER VII

INFRASTRUCTURE AND CIVIL WORKS

HAT CREEK DIVERSION

Object

1. As in Openpit No 1, Hat Creek and its tributaries must be diverted. The 600-ft boundary of Openpit No 2 has been taken as the drainage limit and two systems have been considered.

Diversion Alternatives

2. Two alternatives, each employing a gravity flow canal along the west side of Openpit No 2, have been considered. This western canal would divert flows from the headwaters of Hat Creek and from the creeks entering the area of Openpit No 2 from the western side (ie Crater, Phil, Parke, Lake, McDonald, McCormick, Anderson, Chipuin). These drain the largest portion of the catchment from which run-off would flow to Openpit No 2.

3. The first alternative would be to provide a flood regulating pond at the upper end of the canal to limit the flow from Hat Creek headwaters to 100 ft^3/sec.

4. The other alternative would be to construct the canal of such a size as to pass unregulated peak flows based on 50-year statistics.

5. On the eastern side of Openpit No 2 a drainage ditch would divert flows from the headwaters of White Rock Creek and adjacent creeks into Cashmere and Ambusten Creeks and a second ditch would intercept flows which would otherwise enter Hat Creek downstream of the new canal entrance and divert these into the canal.

6. Due to the topography, no other major diversion works would be required on the eastern side.

Data

7. Cost estimates have been based on the unit prices assumed for Openpit No 1 and pondage volumes have been estimated from the 10-year records (1961 to 1970) of Station No 08LF061 situated near Upper Hat Creek. The synthetic hydrograph developed for Openpit No 1 studies was also used.

Hydrology

8. The catchment areas contributing run off to the main components of the diversion system would be as follows:-

Area	Component	Catchment Area (square miles)
Hat Creek headwaters	Regulating pond	55.0
Western creeks	Western canal	31.0
Headwaters of White Rock and adjacent creek	Eastern ditch	2.3
Upstream of Openpit No 2	Southern ditch	1.0

Peak Flows

9. Based on these catchment areas, and the peak 50-year flow of 1,200 ft^3/sec previously estimated for Hat Creek upstream of Openpit No 1, it was estimated that 50-year, unregulated flows would be approximately as follows:-

Location	50-year Peak Flow (ft^3/sec)
Western canal intake	470
Western canal outlet	730
Eastern ditch outlet	20
Southern ditch outlet	10

10. With regulation of the flow from the headwaters of Hat Creek into the western canal to 100 ft^3/sec, the peak flow at the outlet of the western canal would be reduced to 360 ft^3/sec.

Pondage Requirements

11. The pondage volume required to regulate the flow from the headwaters of Hat Creek to a maximum of 100 ft^3/sec has been estimated from the records of the daily flows that occurred in Hat Creek during June of 1964 (the highest of the 10-year record). During that year, a pond capacity of 860 acre-ft would have been required. A requirement of 1,000 acre-ft has been assumed for the preliminary designs and cost estimates.

Selected System

12. The selected system indicated to be the more economical is that incorporating a flow-regulating pond with the eastern and southern ditches.

13. The ponding dam would be based near elevation 3,650 ft; its crest elevation would be 3,685 ft approximately and its crest length about 1,100 ft. An emergency spillway (sill elevation 3,675 ft) would be provided at the dam to pass flows exceeding the regulating capacity of the pond. Spilled water would be conveyed into the bottom of Openpit No 2 by a culvert as described previously for Openpit No 1.

14. A 4-ft diameter outlet culvert would be provided through the dam to control flows entering the head of the western canal. Culvert discharges would be controlled by two sluice gates operating on the upstream face of the dam.

15. The western canal would be approximately 30,000 ft in length and would be concrete-lined, with bottom widths varying from 3 ft at the upstream end to 6 ft at the downstream end and with depths in the range of 4 to 6 ft. The maximum flow velocity would be 8 ft/sec; gradients would range from 0.2% to 0.3%. The canal would discharge into a natural depression located just downstream of Openpit No 2 and of Anderson Creek. An amount

has been included in the cost estimates for stabilising this depression to prevent erosion from the canal discharges.

16. The general route of the western canal and the eastern ditches is shown on Plate 64 and the estimated cost of the creek and road diversion is shown in Table XLV.

ROAD DIVERSION

17. The most logical and economic relocation for the road would be on the canal bench on the downhill side of the canal, thereby avoiding culverts and providing one road for both public use and canal maintenance.

Surface Mine Buildings

18. This item has been assumed to be identical to Openpit No 1 (see Chapter VII, Report No 2).

Road Construction and Improvement

19. This item has been assumed to be identical to Openpit No 1 (see Chapter VII, Report No 2).

Services

20. This item has been assumed to be identical to Openpit No 1 (see Chapter VII, Report No 2).

Housing

21. This item has been assumed to be identical to Openpit No 1 (see Chapter VII, Report No 2).

SCHEDULE OF EQUIPMENT

22. Table XLVI summarises the Schedule of Equipment - Infrastructure.

CHAPTER VIII

ECONOMICS

BASIS

1. Exactly the same basis has been used to derive the economic results for Openpit No 2 as for Openpit No 1 (Report No 2) and the same sequence of economic tables has been adopted to facilitate comparison. The same basic financial data have been used and the main tables are expressed in 1975 Canadian dollars with inflated costs also given.

CAPITAL COSTS

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

Table XXXI - Mobile Mining Equipment Requirements
Table XL - Schedule of Equipment - Fixed Installations
Table XLVI - Schedule of Equipment - Infrastructure.

3. If a bucket-wheel excavator system were used for the removal of superficial waste, then the capital cost to start-up would be increased by about \$20 million (see Table XXXVII).

4. Again it should be noted that many of the fixed installations, eg coal stockpiling and reclaiming and also ash handling, are strictly not part of the mine but could be considered to be part of the power plant. The capital cost of this equipment is again about \$34 million.

DIRECT OPERATING COSTS

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

Table XLVII - Summary of Electrical Energy Costs
Table XLVIII - Labour Schedule and Payroll Costs
Table XLIX - Materials and Fuel Cost Summary
Table L - Direct Operating Cost Summary

Electrical Energy

6. If a bucket-wheel excavator system were used, the electrical power consumption, and hence the electrical energy costs, would increase and this is allowed for in Table XXXVI.

Labour

7. Appendix "I" gives the labour requirements for Openpit No 2 and it will be noted that the total labour force is estimated at 726 compared with 662 for Openpit No 1 - a result of the greater volumes of material to be moved.

Managerial, Technical and Administrative Staff

8. It has been assumed that the difference in staffing between Openpit No 1 and 2 would be negligible and hence Table XIX (Report No 2) applies also in this case.

TOTAL INVESTMENT AND CAPITAL CHARGES

9. The following tables deal with these items:-

Table LI - Depreciation Summary

Table LII - Capital Investment, Interest during Construction, Interest and Insurance

PRODUCTION COST (1975 PRICES) - 600-FT PIT

10. Table LIII, Coal Production Cost (rom) shows the development of the production cost in the same way as Table XXIV for Openpit No 1. Again the coal handling and ash disposal element would be about 80¢/ton rom. It will be noted that, after the initial development period, the cost remains fairly steady at about $7.50 until the pit begins to widen out in Stage 6 when it increases rapidly. This is a consequence of selecting Scheme B, ie developing a long, narrow pit before widening out.

11. The use of a bucket-wheel excavator system for the removal of superficials could result in a saving of about 43¢/ton (Table XXXVII).

PRODUCTION COST (1975 PRICES)
- 1,500-FT PIT

12. As mentioned in Chapter III, para 13, a pit down to the 2,000-ft elevation (1,500-ft pit) has been postulated. The instantaneous stripping ratio at the probable limit of that pit is 15.4 bank yd^3/short ton rom. Again the approximate production cost at the probable limit has been extrapolated as for Openpit No 1 (see Plate 43, Report No 2). This results in a production cost of about $17/ton as compared with $10/ton at the limit of the 600-ft pit.

DISCOUNTED CASH FLOW
(1975 PRICES)

13. Table LIV shows the cash flow of the expenses and the calculation of the uniform selling prices which would yield an internal rate of return of 15% and 10%. Exactly the same method of calculation has been used as in Report No 2. The uniform selling prices which result are:-

	$ per ton	¢ per 10^6 Btu
15% discount factor	11.38	104
10% discount factor	9.22	84

Plate 80 shows these uniform selling prices compared with the production cost. The price calculated on 10% discount factor can be regarded as the equivalent of the production cost which includes 10% interest because it has been assumed that all the capital is borrowed. Plate 80 also shows the same curves for Openpit No 1 for comparison.

CONFIDENCE LIMITS OF ESTIMATED
SELLING PRICE

14. The question of the level of confidence which can be attached to these selling prices is discussed in Chapter VIII, Report No 2 and the comparable "maximum", "mean" and "minimum" mine-mouth selling prices have been calculated after deduction of 80¢/ton for coal handling and ash disposal costs, ie:-

Coal Prices, \$/ton

Discount rate	10%	15%
Uniform selling price including coal handling and ash disposal costs	9.22	11.38
Coal handling and ash disposal costs	0.80	0.80
"Maximum" mine-mouth selling price	8.42	10.58
"Mean" mine-mouth selling price	7.58	9.52
"Minimum" mine-mouth selling price	6.82	8.57

Again, probable areas of cost saving include steeper slopes, less blasting and earlier economic cut-off (depending on availability of reserves). In addition, the use of a bucket-wheel excavator system instead of scrapers for the removal of superficials would result in a saving of about 43¢/ton.

LIFE OF OPENPIT NO 2

15. As mentioned in Chapter III, para 14, unlike Openpit No 1 the reserves of coal in the 600-ft pit are more than adequate for 35 years of power plant operation, in fact the pit would only reach Stage 6 during this period. The economic estimates have, therefore, been made on this basis rather than on a 30-year basis as in the case of Openpit No 1.

Production Cost (Inflated)

16. The same procedure has been followed as in Report No 2 and Table LV gives the resulting inflated production costs. According to this calculation, the production cost would increase from \$16.31 per short ton rom in Stage 3 (the lowest value) to \$75.45 in Stage 6 (years 2019 to 2020).

Discounted Cash Flow (Inflated)

17. The same procedure has been followed as in Report No 2 and Table LVI gives the resulting inflated uniform selling price at 15% discount factor, ie \$22.19 per ton (202¢/10^6 Btu) which

is about twice the uninflated figure. At 10% discount the corresponding prices are \$21.59 per ton and 196¢/10^6 Btu. The small difference in the prices at 15% and 10% discount is due to the higher discount off-setting the inflation to a greater extent.

OPPORTUNITY VALUE OF HAT CREEK COAL

18. The concept of the opportunity value of Hat Creek coal was discussed in Report No 2 (Openpit No 1). This was considered in the context of an international crude oil price of \$11/bbl and the then ruling internal Canadian price of \$8/bbl, though it was predicted that it would only be a matter of time before the internal price rose to the international level.

19. Such a rise is now imminent. It has been reported in "Petroleum Economist" (June 1976) that the federal government is to raise the controlled price to \$9.05/bbl as from 1st July, 1976 and to make a further increase to \$9.75/bbl with effect from January 1977. This will obviously further improve the opportunity value of Hat Creek coal vis-a-vis oil. The relevant values can be obtained from Plate 45 (Report No 2).

20. The same announcement gives details of parallel increases in natural gas prices. The "city gate" price at Toronto will go up from \$1.25/10^6 Btu to \$1.40/10^6 Btu and then to \$1.50/10^6 Btu.

Break-even Stripping Ratio

21. The break-even stripping ratio for a given coal value can again be determined from Plate 46 (Report No 2).

CHAPTER IX

SUMMARY AND CONCLUSIONS

GENERAL

1. Systematic geological investigation of the coal deposits in the Hat Creek valley since 1957 has resulted in the identification of two main deposits (which may be contiguous in depth), designated Area 1 in the north of the valley (downstream) and Area 2 in the south of the valley (upstream). The object of the current studies is to develop conceptual mines (designated Openpit No 1 and 2 respectively) in these areas and to compare them so that decisions can be taken as follows:-

(i) Whether either or both can be exploited economically for electric power generation (2,000-MW plant) in the first instance or for other uses.

(ii) Which pit should be developed first.

(iii) Whether both pits would need to be developed and, if so, the phasing of this development.

2. To this end, separate studies have been made on the two conceptual pits, the first (Report No 2), dealing with Openpit No 1, being completed in May 1976 and the second being the subject of this report. In order to make valid comparisons and in view of the shortage of data, in certain areas the two studies are based on the same assumptions, follow parallel logic and adopt a similar format. The mine design and economic calculations have been carried out to the same level of confidence. In both cases, practical systems have been selected without detailed optimisation. Unless attention has been drawn to divergencies between the two pits, it may be assumed that the same considerations apply.

GEOLOGICAL AND GEOTECHNICAL CONSIDERATIONS

3. The geological and geotechnical environments of the two pits are very similar, the main difference being the shape of the deposits. The Area 2 deposit is longer, narrower and deeper and is covered with greater thicknesses of overburden. For this reason it was believed, prima facie, that Area 1 was more favourable, and this has been verified. However, Area 2 contains considerably greater reserves of coal, although much of this is so deep that the prospects for its exploitation, at least by surface mining, are doubtful. The prospects for underground mining are worse than in Area 1 because of the greater depth.

MINE DESIGN

4. Exactly the same principles have been used in developing the design of the two openpits, but owing to the elongated shape of the Area 2 deposit, Openpit No 2 will be elongated and less circular than Openpit No 1. This shape may lead to greater slope stability problems. The main means of access into the pits which has been selected is a long incline at the north end where the cover is least. These would suit power station sites north of the deposits but if another site were selected consideration would be given to changing the location of the inclines.

Depth

5. In each case a nominal depth of pit of 600 ft has been used although a conceptual 1,500-ft pit has also been considered. It is reasonable to assume (in the absence of adequate geotechnical data) that a 600-ft pit is feasible but the 1,500-ft pit cannot, at present, be considered prima facie feasible leaving aside all questions of economics. The 600-ft pit in Area 1 contains reserves of coal sufficient only for 30 year of 2,000-MW power plant operation, although it is confidently expected that further reserves may be proved to extend the pit for the full 35 years life specified. In the case of Openpit No 2, the 600-ft pit contains ample reserves for the 35 years. In both pits, but particularly in Openpit No 2, considerable coal resources exist

at greater depth although this may not be economically mineable by surface mines. Underground mining would be extremely difficult and also uneconomic at current price levels.

Bucket-Wheel Excavator Systems

6. It was considered from the outset that the possibility existed of using bucket-wheel excavator/conveyor systems because of the weak nature of many of the rocks present in the valley. However, the geometry of Openpit No 1 does not fit such systems very well (although conveyor/spreader systems were adopted for waste disposal). In the case of Openpit No 2, the geometry of the deposit is more favourable and the quantities of weak rocks (eg superficials) are much greater and therefore a scheme has been drawn up as an alternative to the large fleet of scrapers. This appears to offer the possibility of saving 43¢/ton of coal, although the front-end capital expenditure is higher. The success of such a scheme depends, of course, on the absence of large quantities of hard rock (eg boulders) in the waste which cannot be handled by the system - which is unknown at present.

ENVIRONMENTAL CONSIDERATIONS

7. Openpit No 2 is less favourable from an environmental point of view because of the larger quantities of waste material involved. However, after some 20 years it may be possible to backfill the north end of the pit. The size of the pit is greater and hence the total area of despoilation is greater. The Hat Creek diversion problem is easier because the pit is further upstream and it is possible to arrange a gravity-flow canal, at least around the 600-ft pit. Openpit No 2, being higher up the valley, will leave the northern end of the valley relatively free from interference as far as the mine is concerned.

COMPARISON OF OPENPIT NO 1 AND 2

8. Table LVII lists some of the significant features of the two pits and in all cases except the coal reserves within the pits, Openpit No 2 compares unfavourably with Openpit No 1. The economic comparisons are particularly important. The capital

investment to start-up is 118% greater (and would be even more using bucket-wheel excavators) and the uniform selling price (15% discount basis) 79% greater.

CONCLUSION

9. It is concluded that the results of these studies confirm the preliminary conclusion that Openpit No 1 was the most favourable for first exploitation despite the short-fall in coal reserves in the 600-ft pit.

APPENDIX "F"

LIST OF DOCUMENTS AND DRAWINGS RECEIVED BY PD-NCB FROM 24TH FEBRUARY TO 7TH JUNE, 1976

1. "A Preliminary Assessment of the Washability of Coal from the Hat Creek Property of BC Hydro with an Estimate of the Capital and Operating Costs of a Preparation Plant". Birtley Engineering (Canada) Ltd, August 1975.

2. "Analyses of Hat Creek Coals". K. Fletcher, 2nd April, 1976.

3. "Palynological Zonation and Correlation of Hat Creek Core Samples".

4. Covering letter with the above report from G.E. Rouse to Lisle Jory.

5. E-W and N-S sections of No 2 coal deposit, 1in to 200 ft.

6. Graphical logs of boreholes 76-111, 76-112, 76-112A, 76-113, 76-114, 76-115, 76-116, 76-117, 76-118 and 76-119.

7. Drill hole water level records from December, 1974 to March 1976.

APPENDIX "G"

ANALYSES OF SAMPLES FROM
BOREHOLE NO 75-74

LORING LABORATORIES LTD.

Phone 274-2777

629 Beaverdam Rd. N.E.
Calgary 67, Alberta
T2K 4W2

DOLMAGE CAMPBELL & ASSOCIATES LIMITED
VANCOUVER, BRITISH COLUMBIA
CANADA

CLIENT: B.C. Hydro & Power Authority

PROJECT: Hat Creek Coal

Date: October 1, 1975

D.D.H. 75-74
Sample No. 74-409 (#41-#45)
Footage 1678-1844
Width 166'

Analysis Report No. 10464

PROXIMATE ANALYSIS

	As Rec'd.	Dry Basis	Lab Basis
H_2O		–	7.74
Ash		28.14	25.96
V.M.		35.72	32.96
F.C.		36.14	33.34
BTU		8,723	8,048
S		.80	.74
% Alk. as Na_2O			

ULTIMATE ANALYSIS % Weight

	As Rec'd.	Dry Basis	Lab Basis
H_2O		–	7.74
C		51.00	47.05
H		4.81	4.44
N		1.05	.97
Cl		Trace	Trace
S		.80	.74
Ash		28.14	25.96
O (diff)		14.20	13.10

FUSION TEMP. OF ASH

	Reducing	Oxidizing
Initial Def.	2480	+2640
S (H=W)	+2640	+2640
S (H=½W)	+2640	+2640
Fluid	+2640	+2640

% EQUILIBRIUM MOISTURE =

HARDGROVE GRIND. INDEX = 51

SULPHUR FORMS

Pyritic	% S	.16
Sulphate	% S	.01
Organic	% S	.63
Total	% S	.80

MINERAL ANALYSIS % Wt. Ignited Basis

	Ignited Basis
P_2O_5	.15
SiO_2	47.38
Fe_2O_3	5.72
Al_2O_3	31.01
TiO_2	1.92
CaO	7.49
MgO	.65
SO_3	3.74
K_2O	.10
Na_2O	.93
Undetermined	.91

[signature]
Licensed Assayer of British Columbia

LORING LABORATORIES LTD.

629 Beaverdam Rd. N.E.
Calgary 67, Alberta

Phone 274-2777

DOLMAGE CAMPBELL & ASSOCIATES LTD.
VANCOUVER, BRITISH COLUMBIA
CANADA

CLIENT: B.C. Hydro & Power Authority
PROJECT: Hat Creek Coal

DATE: October 20, 1975
D.D.H. No.: 75-74
Sample No.: 74-709 (#41-#45)
Footage: 1678'-1844'
Width: 166'

Analysis Report No. 10635

FLOAT & SINK ANALYSIS

Sample Crushed to ½sq. x 0

SPECIFIC GRAVITY SINK FLOAT	% WT.	LAB BASIS % MOIST.	DRY BASIS % ASH	DRY BASIS % SULPHUR	DRY BASIS CALORIFIC VALUE, BTU/lb
1.30	10.91	3.09	3.46	.82	12,464
1.30 x 1.50	43.92	3.94	14.34	.84	10,932
1.50 x 1.60	13.81	2.44	27.31	.77	8,857
1.60 x 1.70	5.11	1.79	34.37	.70	7,390
1.70 x 1.90	3.87	1.80	45.94	.67	5,479
1.90	22.38	0.76	62.26	.60	1,937

Licensed Assayer of British Columbia

March, 1976

APPENDIX "H"

C.L.5

CORRA LABORATORIES LIMITED

(NCB)

Examination of Hat Creek Coal - British Columbia

W.M. Robertson
Chief Scientist

Golden Smithies Lane.

HAT CREEK COAL - BRITISH COLUMBIA

INTRODUCTION

A 20 lb sample of coal from a borehole core was left at the Corex Laboratories by Mr. S.C. Brealey, Director of P.D.-N.C.B. Consultants Limited, with a request for analysis.

The sample was taken from a split core approximately two inches in diameter obtained by diamond drilling. The details given were as follows:

Sample Number	74-41
B.H. Number	75-74
Depth	1678-1710 ft
Location	Area 2

The coal was described as "typical" of the vast quantity of coal in the Hat Creek valley although it could not be regarded as in any way "representative".

SAMPLE PREPARATION

The sample was roughly crushed to minus ¼in and about half of it (about 10 lbs) removed for float and sink analysis.

From the remainder two samples were prepared:

One <0.5mm for maceral and reflectance analysis and
one <0.2mm for proximate, ultimate and other analyses.

RESULTS

The results of the various analytical tests and the petrographic and palynological investigation are summarised in the following pages.

TABLE 1. FLOAT AND SINK ANALYSIS

(Sample crushed to -¼in square)

Specific Gravity	Weight %	% Moisture (Air dried basis)	DRY BASIS		
			% Ash	% Sulphur	Calorific Value Btu/lb
ts 1.30	25.0	7.8	7.05	0.75	11690
s 1.30 Floats 1.40	22.3	7.2	11.28	0.71	11660
s 1.40 Floats 1.50	15.8	7.0	17.94	0.80	11510
s 1.50 Floats 1.60	10.9	5.9	26.47	0.74	11400
s 1.60 Floats 1.70	7.5	5.5	34.41	0.71	11160
s 1.70 Floats 1.80	5.1	4.8	41.15	0.67	10710
s 1.80	13.4	2.7	67.30	0.90	8110
Total	100.0		23.70	0.76	11060

I am very sorry that some errors occurred in our report No. CL 5, please accept my apologies.

The correct Calorific Values, calculated to the dry-coal basis, which were telephoned to you on Friday are confirmed in the following table:-

Specific Gravity	Calorific Value Btu/lb, (dry basis)
Floats 1.30	11,790
Sinks 1.30 Floats 1.40	11,140
Sinks 1.40 Floats 1.50	10,160
Sinks 1.50 Floats 1.60	8,920
Sinks 1.60 Floats 1.70	7,750
Sinks 1.70 Floats 1.80	6,620
Sinks 1.80	2,720
Total	9,290

Please correct your copies of the report in Table 1 by substituting the above corrected values for calorific value in the last column.

TABLE 2. ANALYSIS OF <0.2mm COAL

	As Received Basis	Dry Basis	Dry Ash Free Basis
MATE ANALYSIS %			
oisture	11.3	-	-
sh	22.3	25.1	-
olatile Matter	32.7	36.9	49.2
ixed Carbon	33.7	38.0	
alorific Value Btu/lb	Gross 8110 Net 7660		12210
otal Sulphur	0.68	0.77	
yrite Sulphur	0.07	0.08	
ulphate Sulphur	0.02	0.02	
rganic Sulphur	0.59	0.67	
Phosphorus	0.021	0.024	
Specific Gravity	1.50		
H FUSION TEMPERATURE °C (n semi-reducing atmosphere)			
Initial Deformation	1400+		
Hemispherical Temperature	1400+		
Flow Temperature	1400+		
Arsenic parts/million	7	7.9	
IMATE ANALYSIS %			
Moisture	11.30	-	-
Carbon	47.20	53.21	71.08
Hydrogen	3.65	4.11	5.49
Nitrogen	0.90	1.01	1.35
Chlorine	0.02	0.03	0.04
Sulphur	0.68	0.77	1.03
Ash	22.30	25.14	-
Oxygen (by difference)	13.95	15.73	21.01

B.S.S. Number	0
Gray King Coke Type	A
Dilatometry (Ruhr Dilatometer)	10% contraction at 500°C
N.C.B. Coal Rank Code Number	902
International Classification Code/Number	900

TABLE 3. ANALYSIS OF COAL ASH

	%
Silica as SiO_2	54.6
Iron as Fe_2O_3	4.8
Magnesium as MgO	0.5
Calcium as CaO	2.1
Aluminium as Al_2O_3	33.1
Titanium as TiO_2	1.2
Sodium as NaO_2	1.2
Potassium as K_2O	0.3
Phosphorus as P_2O_5	0.27
Sulphur as SO_3	1.7
Undetermined	0.23
Silica Ratio	88.1

TABLE 4. ANALYSIS OF SHALE SAMPLES

	Hard Shale (Carbonate ?)	Fragile Shale (Mudstone ?)
% Loss in weight in nitrogen at 105°C	3.4	5.4
% Loss on ignition at 200°C	5.1	6.2
SiO_2	13.9	59.1
Fe_2O_3	14.0	3.2
MgO	2.2	1.0
CaO	19.4	0.9
Al_2O_3	15.9	21.0
TiO_2	0.6	1.0
Na_2O	0.2	1.0
K_2O	0.1	1.1
P_2O_5	2.0	0.1
CO_2	21.1	0.8
Water and Organic Matter	10.0	10.0

P9-69

PETROGRAPHIC AND PALYNOLOGICAL INVESTIGATION

Maceral and reflectance analyses were made on a suitably prepared composite sample of crushed coal (0.5mm - 0) from the core sample. A qualitative examination was also made of three lumps of coal selected for their contrasting appearance by eye.

RESULTS

Maceral group and Mineral composition	Maceral	Volume %
Huminite		83.0
Liptinite	Sporinite	0.4
	Resinite	2.4
Inertinite		0.6
Clay mineral		13.4
Pyrites		0.2

REFLECTANCE

R_o range	Frequency
0.20 - 0.29	27
0.30 - 0.39	57
0.40 - 0.49	16

Average maximum reflectance in oil 0.34% (at 546 nm and in oil RI 1.518).

DESCRIPTION OF COAL

The coal is mainly composed of the maceral sub groups Humotelinite, Humodetrinite and Humocollinite. The Humotelinite is mainly composed of the maceral ulminite. Sporinite and resinite are present in small amounts and are the main macerals of the Leptinite group. Macerals of the inertinite group are uncommon and are mainly represented by sclerotinite (fungal spores). Clay mineral is finely dispersed through much of the huminite but also occurs as thin bands and particles of shale.

Examination of the pieces of coal showed that the bedding is very distorted. One piece of dull coal was found to consist of shale and carbonaceous shale. Embedded in the shale were several resin bodies 2-5mm in length. A piece of coal selected for its bright appearance contained a large amount of submicroscopically dispersed resin which imparted an orange tint to certain bands.

PALYNOLOGY

Various maceration methods were tried and all gave similar results. Very few palynomorphs were recognised comprising very few species. The forms were mainly thin walled and unornamented, mostly within the size range 25-35 µm. From the condition of the palynomorphs it would appear that the original environment of deposition may not have been favourable for their preservation. The most abundant palynomorphs were the remains of fungal spores.

RANK OF COAL

Based on its volatile matter content (d.a.f.) and calorific value (a.f. raw coal) the coal would be classed according to the German (DIN) classification as a Mattbraunkohle and according to the American (ASTM) classification as a sub bituminous B rank coal. However, the measured maximum reflectance in oil is lower than the values generally recognised for coals of this rank (0.4 - 0.5%). Coals having a R_o max of 0.34 would be classed as Lignite in the American system. The microscopic evidence suggests that the coal has reached the late lignite (Mattbraunkohle) stage of coalification since ulminite is the predominant maceral of the humotelinite sub group. The formation of this maceral takes place at about this rank stage as a result of the process of geochemical gelification.

APPENDIX "I"

LABOUR REQUIREMENTS

MINE LABOUR FORCE

1. The mining labour requirements and associated wage costs related to the development and operation of the mine, but excluding building and construction work, are shown in Table XLVIII. It will be noted that an initial labour force of some 400 employees is required during the pre-production stage. The peak labour requirement is reached in Stage 4 at a total of 726 employees apportioned to the major sections of the mine operation as follows:-

Mobile mining equipment	544
Fixed installations	142
Infrastructure	40
Total	726

2. The three categories of operators shown in Table XLVIII, and the hourly rates of pay, are the same as those used in Report No 2.

MANAGERIAL, TECHNICAL AND ADMINISTRATIVE STAFF

3. The numbers of managerial, technical and administrative staff are identical to the requirements in Report No 2.

<u>Explanatory Note</u>

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

TABLE IR

BASIC PLANNING DATA

Density of in-situ coal	-	1.39 tons/bank yd^3
Swell	-	25%
Swell factor	-	0.8
Density of in-situ waste in coal	-	1.87 tons/bank yd^3
Swell	-	50%
Swell factor	-	0.67
Density of superficial deposits	-	1.56 tons/bank yd^3
Swell	-	15%
Swell factor	-	0.87
Density of claystone (assumed wet)	-	1.87 tons/bank yd^3
Swell	-	40%
Swell factor	-	0.715
Density of in-situ marble	-	2.3 tons/bank yd^3
Swell	-	50%
Swell factor	-	0.67
Density of in-situ volcanics	-	2.2 tons/bank yd^3
Swell	-	50%
Swell factor	-	0.67
Estimated in-situ waste content	-	22%
Estimated waste extraction by selective mining	-	15%
Waste remaining in rom coal	-	7%
Working days per year	-	350
Hours per shift	-	8
Teams of men	-	4
No of producing shifts per week	-	20
No of maintenance shifts per week	-	1

TABLE XXIX

OPEN PIT No. 2. VOLUMES TONNAGES AND RATIOS.

A. REFERRED TO IN SITU COAL

PIT STAGE	PIT FLOOR ELEVATION (Ft)	MARBLE STAGE 10^6 b yd³	MARBLE CUMULATIVE 10^6 b yd³	VOLCANICS STAGE 10^6 b yd³	VOLCANICS CUMULATIVE 10^6 b yd³	SUPERFICIALS STAGE 10^6 b yd³	SUPERFICIALS CUMULATIVE 10^6 b yd³	OVERLYING WASTE STAGE 10^6 b yd³	OVERLYING WASTE CUMULATIVE 10^6 b yd³	TOTAL OVERBURDEN STAGE 10^6 b yd³	TOTAL OVERBURDEN CUMULATIVE 10^6 b yd³	IN SITU COAL STAGE 10^6 s tons	IN SITU COAL CUMULATIVE 10^6 s tons	STRIPPING RATIO STAGE b yd³/s ton	STRIPPING RATIO CUMULATIVE b yd³/s ton	INSTANTANEOUS STRIPPING RATIO AT END OF STAGE b yd³ / s ton
1	3000	-	-	-	-	19	19	10	10	29	29	2	2	14.5	14.5	17.7
2	2900	-	-	-	-	41	60	12	22	53	82	20	22	2.7	3.7	4.9
3	2900	-	-	-	-	179	239	90	112	269	351	117	139	2.3	2.5	4.7
4	2900	-	-	95	95	127	366	89	201	311	662	172	311	1.8	2.1	3.0
5	2900	-	-	56	151	199	565	108	309	363	1025	207	518	1.8	2.0	3.2
6	2900	-	-	43	194	161	726	143	452	347	1372	108	626	3.2	2.2	5.1
7	2900	-	-	101	295	125	851	188	640	414	1786	96	722	4.3	2.5	7.8
8	2900	26	26	72	367	74	925	154	794	326	2112	59	781	5.5	2.7	10.2
9	2000	642	668	865	1232	1492	2417	5220	6014	8219	10 331	3215	3996	2.6	2.6	14.3

B. REFERRED TO R.O.M. COAL

PIT STAGE	PIT FLOOR ELEVATION (Ft)	MARBLE STAGE 10^6 b yd³	MARBLE CUMULATIVE 10^6 b yd³	VOLCANICS STAGE 10^6 b yd³	VOLCANICS CUMULATIVE 10^6 b yd³	SUPERFICIALS STAGE 10^6 b yd³	SUPERFICIALS CUMULATIVE 10^6 b yd³	OVERLYING AND SEGREGATED WASTE STAGE 10^6 b yd³	OVERLYING AND SEGREGATED WASTE CUMULATIVE 10^6 b yd³	TOTAL OVERBURDEN & SEGREGATED WASTE STAGE 10^6 b yd³	TOTAL OVERBURDEN & SEGREGATED WASTE CUMULATIVE 10^6 b yd³	R.O.M. COAL STAGE 10^6 s tons	R.O.M. COAL CUMULATIVE 10^6 s tons	STRIPPING RATIO STAGE b yd³/s ton	STRIPPING RATIO CUMULATIVE b yd³/s ton	INSTANTANEOUS STRIPPING RATIO AT END OF STAGE b yd³ / s ton
1	3000	-	-	-	-	19	19	10	10	29	29	2	2	14.5	14.5	19.1
2	2900	-	-	-	-	41	60	14	24	55	84	17	19	3.2	4.4	5.3
3	2900	-	-	-	-	179	239	100	124	279	363	99	118	2.8	3.1	5.1
4	2900	-	-	95	95	127	366	103	227	325	688	146	264	2.2	2.6	3.2
5	2900	-	-	56	151	199	565	125	352	380	1068	176	440	2.2	2.4	3.4
6	2900	-	-	43	194	161	726	152	504	356	1424	92	532	3.9	2.7	5.5
7	2900	-	-	101	295	125	851	195	699	421	1845	82	614	5.1	3.0	8.4
8	2900	26	26	72	367	74	925	159	858	331	2176	50	664	6.6	3.3	11.0
9	2000	642	668	865	1232	1492	2417	5478	6336	8477	10 653	2733	3397	3.1	3.1	15.4

NOTES
1. SPECIFIC GRAVITIES USED IN SHORT TONS / YD³ IN SITU COAL 1.39, ROM COAL 1.29, CLAYSTONE 1.87, MARBLE 2.3, SUPERFICIALS 1.56, VOLCANICS 2.2
2. STRIPPING RATIO DEFINED AS WASTE PRODUCTION (BANK YD³) : COAL PRODUCTION (SHORT TONS)
3. CUMULATIVE STRIPPING RATIO BASED ON TOTAL PIT VOLUMES TO END OF STAGE.
4. INSTANTANEOUS STRIPPING RATIO BASED ON VOLUMES MINED IN THE LAST INCREMENT, AT THE END OF THE STAGE.
5. SEGREGATED WASTE ASSUMED TO BE 15% OUT OF THE 22% WASTE IN THE IN SITU COAL.

No 2 DEPOSIT REPORT No. 3 2/6/76

PRODUCTION SCHEDULE / YEARLY AND CUMULATIVE TABLE XXX SHEET

PROJECT: HAT CREEK CONTRACT No. TS 605 SCHEME: No. 2 DEPOSIT CURRENCY UNIT: 1ST JUNE 1976 DATE

START UP DATE JULY 1982 ↓ No1 JULY 1984 ↓ No2 ↓ No3 APR '85

① ② ③ (↓ 18/19, ↓ 19/20)

LINE No.	ITEM	REF	79/80	80/81	81/82	82/83	83/84	84/85	85/86	86/87	87/88
1	MINING STAGE		1	1	1	1	2	2	3	3	3
2											
3											
4	POWER STATION REQUIREMENTS	10^6 S.TONS	–	–	–	1	4	7	12	12	12
5	(6000 BTU/lb COAL)										
6	IN SITU COAL PRODUCTION YEARLY	10^6	0.1	0.2	0.5	0.5	5.1	8.6	16	15	16
7	CUMULATIVE	S.TONS	0.1	0.3	0.8	1.3	6.4	15	31	46	62
8											
9	ROM COAL PRODUCTION YEARLY	10^6	0.1	0.2	0.4	0.4	4.3	7.6	13	13	13
10	CUMULATIVE	S.TONS	0.1	0.3	0.7	1.1	5.4	13	26	39	52
11											
12	SEGREGATED WASTE YEARLY	10^6	–	–	0.1	0.1	0.8	1	3	2	3
13	CUMULATIVE	S.TONS	–	–	0.1	0.2	1.0	2	5	7	10
14											
15	SUPERFICIALS YEARLY	10^6	19	19	19	18	18	18	18	18	18
16	CUMULATIVE	B Yd^3	19	38	57	75	93	111	129	147	165
17											
18	PIT WASTE YEARLY	10^6	8	8	8	9	9	9	9	9	9
19	CUMULATIVE	B Yd^3	8	16	24	33	42	51	60	69	78
20											
21	VOLCANICS YEARLY	10^6	–	–	–	–	–	–	–	–	–
22	CUMULATIVE	B Yd^3	–	–	–	–	–	–	–	–	–
23											
24	MARBLE YEARLY	10^6	–	–	–	–	–	–	–	–	–
25	CUMULATIVE	B Yd^3	–	–	–	–	–	–	–	–	–
26											
27	TOTAL WASTE YEARLY	10^6	27	27	27	27	27	27	27	27	27
28	(excluding segregated) CUMULATIVE	B Yd^3	27	54	81	108	135	162	189	216	243
29											
30	YEARLY STRIPPING RATIO	–	270	135	54	54	5.3	3.1	1.7	1.8	1.7
31	(In Situ Coal Basis)										
32											
33	INSTANTANEOUS STRIPPING RATIO	–	–	–	–	17	13	7.5	4.7	4.6	4.4
34	(In Situ Coal basis)										
35											
36	PIT WASTE PLUS VOLCANICS YEARLY	10^6	15	15	15	17	18	18	20	19	20
37	PLUS SEGREGATED WASTE CUMULATIVE	S.TONS									
38											
39	YEARLY STRIPPING RATIO	–	312	156	62	62	6.1	3.6	2.1	2.1	2.1
40	(Rom Coal basis)										
41											
42	INSTANTANEOUS STRIPPING RATIO	–	–	–	–	19.1	15	13	5.3	5	4.6
43	(Rom Coal basis)										
44											
45	Specific Gravities :-										
46	Superficials 1.56 ston/b yd^3										
47	Pit Waste 1.87 ston/b yd^3										
48	Volcanics 2.20 ston/b yd^3										
49	Marble 2.30 ston/b yd^3										
50											

LINE No.	ITEM	88/89	89/90	90/91	91/92	92/93	93/94	94/95	95/96	96/97	97/98	98/99	99/2000	2000/2001	01/02	02/03	03/04	04/05	05/06	06/07	07/08	08/09
1	MINING STAGE	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4	4	5	5	5	5	5
4	POWER STATION REQUIREMENTS	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12	12
6	IN SITU COAL PRODUCTION YEARLY	15	15	16	15	16	15	15	16	15	16	15	15	16	15	16	15	15	16	15	16	15
7	CUMULATIVE	77	92	108	123	139	154	169	185	200	216	231	246	262	277	293	308	323	339	354	370	385
9	ROM COAL PRODUCTION YEARLY	13	13	14	13	13	13	13	13	13	13	13	13	14	13	13	13	13	13	13	13	13
10	CUMULATIVE	65	78	92	105	118	131	144	157	170	183	196	209	223	236	249	262	275	288	301	314	327
12	SEGREGATED WASTE YEARLY	2	2	2	2	3	2	2	3	2	3	2	2	2	2	3	2	2	3	2	3	2
13	CUMULATIVE	12	14	16	18	21	23	25	28	30	33	35	37	39	41	44	96	48	51	53	56	58
15	SUPERFICIALS YEARLY	18	18	18	18	12	12	12	12	12	12	12	12	12	12	12	12	13	13	13	13	13
16	CUMULATIVE	33	201	219	237	249	261	273	285	297	309	321	333	345	357	369	381	394	407	420	433	446
18	PIT WASTE YEARLY	9	9	9	9	9	7	7	7	7	7	7	7	7	7	7	7	9	9	9	9	9
19	CUMULATIVE	87	96	105	114	123	130	137	144	151	158	165	172	179	186	193	200	209	218	227	236	245
21	VOLCANICS YEARLY	–	–	–	–	6	8	8	8	8	8	8	8	8	8	8	8	5	5	5	5	5
22	CUMULATIVE	–	–	–	–	6	14	22	30	38	46	54	62	70	78	86	94	99	104	109	114	119
24	MARBLE YEARLY	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
25	CUMULATIVE	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
27	TOTAL WASTE YEARLY	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27	27
28	(excluding segregated) CUMULATIVE	270	297	324	351	378	405	432	459	486	513	540	567	594	621	648	675	702	729	756	783	810
30	YEARLY STRIPPING RATIO (In Situ Coal Basis)	1.8	1.8	1.7	1.8	1.7	1.8	1.8	1.7	1.8	1.7	1.8	1.8	1.7	1.8	1.7	1.8	1.8	1.7	1.8	1.7	1.8
33	INSTANTANEOUS STRIPPING RATIO (In Situ Coal basis)	4.4	4.5	4.6	4.6	4.7	4.6	4.4	4.2	4.1	4.0	3.8	3.7	3.5	3.4	3.1	3.0	2.8	2.7	2.6	2.6	2.6
36	PIT WASTE PLUS VOLCANICS YEARLY	9	19	19	19	33	33	33	34	33	34	33	33	33	33	33	33	30	31	30	31	30
39	YEARLY STRIPPING RATIO (Rom Coal basis)	2.1	2.1	1.9	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1	1.9	2.1	2.1	2.1	2.1	2.1	2.1	2.1	2.1
42	INSTANTANEOUS STRIPPING RATIO (Rom Coal basis)	4.9	4.9	5.0	5.1	5.2	5	4.8	4.6	4.4	4.2	4.0	3.9	3.7	3.6	3.5	3.2	3.0	2.9	2.8	2.8	2.8

LINE No.	ITEM	09/10	10/11	11/12	12/13	13/14	14/15	15/16	16/17	17/18	18/19	19/20				
1	MINING STAGE	5	5	5	5	5	5	5	6	6	6	6	6	7	8	9
4	POWER STATION REQUIREMENTS	12	12	12	12	12	12	12	12	12	8	4*	–	–	–	–
6	IN SITU COAL PRODUCTION YEARLY	16	15	15	16	15	15	16	15	16	11	5	86	96	59	3215
7	CUMULATIVE	401	416	431	447	462	477	493	508	524	535	540	626	722	781	3996
9	ROM COAL PRODUCTION YEARLY	14	13	13	13	13	13	13	13	14	9	4	73	82	50	2733
10	CUMULATIVE	341	354	367	380	393	406	419	432	446	455	459	532	614	664	3397
12	SEGREGATED WASTE YEARLY	2	2	2	3	2	2	3	2	2	2	1	13	14	9	482
13	CUMULATIVE	60	62	64	67	69	71	74	76	78	80	81	94	108	117	599
15	SUPERFICIALS YEARLY	13	14	15	15	15	15	15	15	15	12	10	126	125	74	1492
16	CUMULATIVE	459	473	488	503	518	533	548	563	578	590	600	726	851	925	2417
18	PIT WASTE YEARLY	9	9	8	8	8	8	8	8	8	8	8	117	188	154	5220
19	CUMULATIVE	254	263	271	279	287	295	303	311	319	327	335	452	640	794	6014
21	VOLCANICS YEARLY	5	4	4	4	4	4	4	4	4	2	2	34	101	72	865
22	CUMULATIVE	124	128	132	136	140	144	148	152	156	158	160	194	295	367	1232
24	MARBLE YEARLY	–	–	–	–	–	–	–	–	–	–	–	–	–	26	642
25	CUMULATIVE	–	–	–	–	–	–	–	–	–	–	–	–	–	26	668
27	TOTAL WASTE YEARLY	27	27	27	27	27	27	27	27	27	22	20	277	414	326	8219
28	(excluding segregated) CUMULATIVE	837	864	891	918	945	972	999	1026	1053	1075	1095	1372	1786	2112	10331
30	YEARLY STRIPPING RATIO (In Situ Coal Basis)	1.7	1.8	1.8	1.7	1.8	1.8	1.7	1.8	1.7	2.0	4.0	3.2	4.3	5.5	2.6
33	INSTANTANEOUS STRIPPING RATIO (In Situ Coal basis)	2.6	2.6	2.6	2.7	2.7	2.8	2.9	3.1	3.3	3.4	3.5	5.1	7.8	10.2	14.3
36	PIT WASTE PLUS VOLCANICS YEARLY	30	28	26	27	26	26	27	26	26	21	20				
39	YEARLY STRIPPING RATIO (Rom Coal basis)	1.9	2.1	2.1	2.1	2.1	2.1	2.1	2.1	1.9	2.4	5.0	3.7	5	6.5	3
42	INSTANTANEOUS STRIPPING RATIO (Rom Coal basis)	2.8	2.9	2.9	3.0	3.1	3.2	3.3	3.4	3.4	3.6	3.7	5.5	8.4	11	15.4

* 1 million tons stock consumed

① No.1 Generator closes March 2018
② No.2 Generator closes July 2019
③ No.3 Generator closes March 2020

MA PREPARED

CHECKED

MOBILE MINING EQUIPMENT REQUIREMENTS

NO. OF UNITS

			79/80	80/81	81/82	82/83	83/84	84/85	85/86	86/87	87/88	88/89	89/90	90/91	91/92	92/93	93/94	94/95	95/96	96/97	97/98	98/99	99/2000	00/01	01/02	02/03	03/04	04/05	05/06	06/07	07/08	08/09	09/10	10/11	11/12	12/13	13/14	14/15	15/16	16/17	17/18	18/19	19/20
	STAGE				1			2					3									4												5								6	
	COAL	$ 000s																																									
	DRILLS AND COMPRESSORS	100	1	1	1	1	2	3	4	5	5	5	5	5	5	5	5	5	5	5	5	5	[illegible]	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	3	2
	SHOVELS	1390	1	1	1	1	1	2	3	3	3	3	[illegible]	3	3	3	3	3	3	2	3	3	[illegible]	3	3	2	3	2	3	3	3	3	2	3	2	3	3	3	3	3	3	2	1
	TRUCKS	520	2	2	2	2	2	3	7	7	7	7	7	7	7	7	9	9	9	9	9	9	10	10	10	10	10	11	11	11	11	11	11	12	12	12	12	12	12	12	12	9	4
	BULLDOZERS	230	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1
	WHEELDOZERS	160	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	WATER TANKERS	210	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	DIESEL TANKERS	30	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	MAINTENANCE VEHICLES	30	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	GRADERS	170	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	PICK UPS (1TON)	10	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	4	4	4	4	4	4	4	4	4	4	4	4	5	5	5	5
	SUMP PUMPS	30	2	3	3	3	4	4	7	7	7	7	7	7	7	7	10	10	10	10	10	10	10	10	10	10	10	13	13	13	13	13	13	13	13	13	13	13	13	16	16	16	16
	EXPLOSIVE TRUCKS	40	1	1	1	1	1	1	1	1	1	1	1	1		1	2	2	2	2	2	2	2	2	2	2	2	3	3	3	3	3	3	3	3	3	3	3	3	4	4	4	4
	SEGREGATED WASTE																																										
	SHOVELS	1390	← USE COAL SHOVEL →				1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	TRUCKS	520	1	1	1	1	1	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	3	3	2	1
	PIT WASTE																																										
	DRILLS AND COMPRESSORS	100	3	3	3	5	5	5	5	5	5	5	5	5	5	5	3	3	3	3	3	3	3	3	3	3	3	5	5	5	5	5	5	5	3	3	3	3	3	3	3	3	3
	SHOVELS	1390	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	TRUCKS	520	8	8	8	8	8	8	9	9	9	9	9	9	9	9	10	10	10	10	10	10	10	10	10	10	10	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14	14
	BULLDOZERS	230	1	1	1	1	1	1	1	1	1	1	1	1	1	2	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3
These include ancillary vehicles for volcanics below.	WHEELDOZERS	160	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
	WATER TANKERS	210	2	2	2	2	3	3	3	3	3	3	3	3	3	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
	DIESEL TANKERS	30	2	2	2	2	3	3	3	3	3	3	3	3	3	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
	MAINTENANCE VEHICLES	30	2	2	2	2	3	3	3	3	3	3	3	3	3	4	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5	5
	GRADERS	170	2	2	3	3	3	3	4	4	4	[illegible]	4	4	4	5	5	5	5	5	5	5	5	5	5	5	5	6	6	6	6	6	6	6	6	6	6	6	6	7	7	7	7
	PICK UPS (1TON)	10	2	2	3	3	3	3	4	4	4	4	4	4	4	5	5	5	5	5	5	5	5	5	5	5	5	6	6	6	6	6	6	6	6	6	6	6	6	7	7	7	7
	EXPLOSIVE TRUCKS	40	2	2	2	2	2	2	3	3	3		2	3	3	5	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	6	7	7	7	7
	VOLCANICS																																										
See Above	DRILLS 60R	510														4	5	5	5	5	5	5	5	5	5	5	5	3	3	3	3	3	3	3	3	3	3	3	3	3	3	2	2
	SHOVELS	1390														2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1	1	1	1	1	1
	TRUCKS	520														6	11	11	11	11	11	11	11	11	11	11	11	8	8	8	8	8	8	7	7	7	7	7	7	7	7	4	4
	SUPERFICIALS																																										
* South Conveyor from Stage 4	SCRAPERS	370	30	30	30	30	21	[illegible]	[illegible]	[illegible]	32	32	32	32	32	22	19*	19	19	19	19	19	19	19	19	19	19	12	12	12	12	12	12	13	14	14	14	14	14	8	8	7	5
	PUSHERS / BULLDOZERS	230	10	10	10	10	6	6	[illegible]	[illegible]	10	10	10	10	10	6	5	5	5	5	5	5	5	5	5	5	5	4	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3
	WATER TANKERS	210	2	2	2	2	2	[illegible]	4	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	DIESEL TANKERS	30	2	2	2	2	2	2	[illegible]	[illegible]	[illegible]	[illegible]	3	3	3	2	3	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
	MAINTENANCE VEHICLES	30	5	5	5	[illegible]	4	4	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	6	6	4	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	3	2	2	2	2	2	[illegible]
	GRADERS	170	2	2	2	2	4	4	4	4	[illegible]	[illegible]	[illegible]	4	4	4	4	4	4	4	4	4	4	4	4	4	4	3	3	3	3	3	3	3	3	3	[illegible]	3	3	3	3	3	2
	COMPACTORS	160	2	2	2	2	3	3	3	3	[illegible]	[illegible]	[illegible]	[illegible]	[illegible]	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	3	[illegible]

TABLE XXXII

PROJECT: HAT CREEK | CONTRACT No. TC 605 | SCHEME: SCHEDULE OF MOBILE MINING EQUIPMENT – INITIAL & REPLACEMENT COSTS | CURRENCY UNIT: $ 000's | DATE

LINE No.	ITEM	REF	1979/80 No	1979/80 Cost	1980/81 No	1980/81 Cost	1981/82 No	1981/82 Cost	1982/83 No	1982/83 Cost	STAGE 1 (1979/80 to 1981/82) No	STAGE 1 Cost	1983/84 No	1983/84 Cost	1984/85 No	1984/85 Cost	STAGE 2 (1983/84 to 1984/85) No	STAGE 2 Cost	STAGE 3 (1985/86 to 1992/93) No	STAGE 3 Cost	STAGE 4 (1993/94 to 2003/2004) No	STAGE 4 Cost	STAGE 5 (2004/2005 to 2015/16) No	STAGE 5 Cost	STAGE 6 (2016/17 to 2019/20) No	STAGE 6 Cost	No. of units	Total Cost
	COAL																											
1	DRILLS & COMPRESSORS	100	1	100				100			2	200	2	200	1	100	3	300	20	2000	28	2800	30	3000	7	700	990	9000
2	SHOVELS	1390	1	1390							1	1390	–	–	1	1390	1	1390	2	2780	3	4170	5	6950	—	—	122	16680
3	TRUCKS	520	2	1040			[illegible]	1040			4	2080	2	1040	1	520	3	1560	28	14560	56	29120	69	35880	16	8320	1796	91520
4	BULLDOZERS	230	1	230			1	230			2	460	1	230	–	–	1	230	4	920	6	1380	12	2760	4	920	223	6670
5	WHEELDOZERS	160	1	160			1	160			2	320	1	160	–	–	1	160	4	640	6	960	6	960	2	320	221	3360
6	WATER TANKERS	210	1	210							1	210	1	210	–	–	1	210	2	420	3	630	3	630	—	—	110	2100
7	DIESEL TANKERS	30	1	30							1	30	1	30	–	–	1	30	2	60	3	90	3	90	—	—	110	300
8	MAINTENANCE VEHICLES	30	1	30							1	30	1	30	–	–	1	30	2	60	3	90	3	90	—	—	110	300
9	GRADERS	170	1	170			1	170			2	340	1	170	–	–	1	170	4	680	6	1020	6	1020	2	340	221	3570
10	PICK UPS (1 TON)	10	2	20							2	20	2	20	–	–	2	20	4	40	9	90	12	120	3	30	332	320
11	SUMP PUMPS	30	2	60	1	30					3	90	1	30	–	–	1	30	6	180	14	420	23	690	3	90	550	1500
12	EXPLOSIVE TRUCKS	40	1	40							1	40	1	40	–	–	1	40	2	80	6	240	8	320	4	160	222	880
13	SUB – TOTAL			3480		30		1700	–	NIL		5210		2160		2010		4170		22420		41010		52510		10880		136200
14	SEGREGATED WASTE																											
15	SHOVELS	1390	–	–	–	–	–	–	–	–	–	—	1	1390	–	–	1	1390	—	—	2	2780	1	1390	—	—	44	5560
16	TRUCKS	520	1	520			1	520	–	–	2	1040	1	520	–	–	1	520	8	4160	12	6240	12	6240	5	2600	440	20800
17	SUB – TOTAL			520				520				1040		1910				1910		4160		9020		7630		2600		26360
18	PIT WASTE																											
19	DRILLS AND COMPRESSORS	100	3	300	–	–	3	300	2	200	8	800	3	300	2	200	5	500	20	2000	16	1600	25	2500	6	600	80	8000
20	SHOVELS	1390	2	2780			–	–			2	2780	–	–	–	–	–	–	2	2780	2	2780	2	2780	—	—	8	11120
21	TRUCKS	520	8	4160			8	4160		–	16	8320	8	4160	–	–	8	4160	36	18720	60	31200	84	43680	28	14560	2332	120640
22	BULLDOZERS	230	1	230			1	230	–		2	460	1	230	–	–	1	230	5	1150	17	3910	18	4140	6	1380	449	11270
23	WHEELDOZERS	160	1	160			1	160			2	320	1	160	–	–	1	160	4	640	6	960	6	960	2	320	221	3360
24	WATER TANKERS	210	2	420			–	–			2	420	3	630	–	–	3	630	7	1470	14	2940	15	3150	2	420	443	9030
25	DIESEL TANKERS	30	2	60			–	–			2	60	3	90	–	–	3	90	7	210	14	420	15	450	2	60	443	1290
26	MAINTENANCE VEHICLES	30	2	60			–	–			2	60	3	90	–	–	3	90	7	210	14	420	15	450	2	60	443	1290
27	GRADERS	170	2	340			3	510	–		5	850	3	510	–	–	3	510	17	2890	29	4930	36	6120	14	2380	1004	17680
28	PICK UP (1 TON)	10	2	20				10			3	30	2	20	–	–	2	20	9	90	14	140	18	180	5	50	551	510
29	EXPLOSIVE TRUCKS	40	2	80			–	–			2	80	2	80	–	–	2	80	8	320	16	640	18	720	7	280	553	2120
30	SUB TOTAL			8610	–	–		5370	–	200		14180		6270		200		6470		30480		49940		65130		20110		186310
31	VOLCANICS						—	—	—	—	—																	
32	DRILLS (EOR)	510	–	–	–	–	–	—	–	–	—	—	–	—	—	—	—	—	4	2040	5	2550	3	1530	—	—	122	6120
33	SHOVELS		–	–	–	–	–	—	–	—	—	—	–	—	—	—	—	—	2	2780	2	2780	1	1390	—	—	55	6950
34	TRUCKS		–	–	–	–	–	—	–	—	—	—	–	—	—	—	—	—	6	3120	60	31200	45	23400	9	4680	1220	62400
35	SUB – TOTAL																		-	7940	-	36530		26320		4680		75470
36	SUPERFICIALS																											
37	SCRAPERS	370	30	11100			30	11100			60	22200	21	7770	–	–	21	7770	128	47360	117	43290	66	24420	14	5180	4066	150220
38	PUSHERS / BULLDOZERS	230	10	2300			[illegible]	2300			20	4600	6	1380			6	1380	40	9200	30	6900	24	5520	3	690	1233	28290
39	WATER TANKERS	210	2	420			–	–			2	420	2	420			2	420	8	1680	9	1890	6	1260	-	-	227	5670
40	DIESEL TANKERS	30	2	60			–	–			2	60	2	60			2	60	6	180	9	270	6	180	-	–	25	750
41	MAINTENANCE VEHICLES	30	5	150			–	–			5	150	4	120			4	120	12	360	13	390	9	270	-	–	42	1260
42	GRADERS	170	2	340			2	340			4	680	4	680			4	680	16	2720	24	4080	18	3060	3	510	69	11730
43	COMPACTORS	160	2	320			2	320			4	640	3	480			3	480	12	1920	18	2880	18	2880	3	480	58	9280
44	SUB TOTAL			14690				14060				28750		10910				10910		63420		59670		37590		5560		207200
45	+ 10% INITIAL SPARES ON NEW EQUIPMENT			2730				20		20		2770		230		200		430		1490		560		370		90		5740
46	TOTAL			30300		30		21650		220		51950		21450		2410		23890		129910		196760		185550		45220		637280
47	TOTAL DEPRECIATION / STAGE	($1000's)		11640		11640		11730		11830		46840		10180		10630		20810		120350		182710		189770		53440		
48	AV. DEPRECIATION $/ton			–		–		–		–		–		2.36		1.40		1.75		1.15		1.27		1.21		1.33		
49	¢/10^6 BTU			–		–		–		–		–		21		13		16		10		12		11		12		
50																												

PREPARED BY:

CHECKED BY:

2740

TABLE XXXIII

MOBILE EQUIPMENT - MACHINE/ACTIVITY COSTS EXPRESSED AS PERCENTAGES

Activity	% Total Cost	Machine Type	% Total Cost
Superficials	33	Shovels	6
Pit Waste	29	Trucks	47
Coal Extraction	22	Scrapers	24
Segregated Waste	4	Drills and Compressors	4
*Volcanics	12	Bulldozers	7
		Graders	5
		Others	7
	100		100

* Note that ancillary equipment is included in pit waste total.

TABLE XXXIV

SCHEDULE OF TYPICAL EQUIPMENT

Category	Type	Manufacturer	Model	Capacity
Shovel	Electric	Bucyrus Eyrie	195	15 yd^3
Drills	(Compressed air	Gardner Denver	3100A	4-in holes
	(Electric	Bucyrus Eyrie	60R	9"-12$\frac{1}{4}$"holes
Compressors	Diesel	Gardner Denver	SP600	600 ft^3/min
Off-highway trucks	Diesel	Wabco	150 C	117 tons coal
Bulldozers	Diesel	Caterpillar	D9H	-
Wheeldozers	Diesel	Caterpillar	824	-
Graders	Diesel	Caterpillar	16G	-
Scrapers	Diesel	Caterpillar	666	41 bank yd^3
Compactors	Diesel	Caterpillar	825	-
Water tanker	Diesel	Caterpillar	631	10,000 US gals
Bucket wheel excavator	Electric	Knupp	500C	2,000 yd^3/hr

NOTE: The manufacturers' names and model numbers have been given to enable production details and costs to be specified concisely. They are not intended to indicate any preference.

TABLE XXXV

HAT CREEK II

Time Schedule for superficials removal by bucket wheel excavator

Block	Volume of superficials in block ($10^6 yd^3$ bank)	Cumulative Volume	Date for commencement	Time for removal (Yrs.)	Completion date	Mining Scheduled Date	Cumulative superficials to be moved
XA	15	15	1979/80	0.75	Dec.'81	1983	19
Aa'	49	64	Jan.82	2.5	Jul.'84	Jul.85	60
a'B	71	135	Sep.84	3.5	Mar.88		
B b'	91	226	May 88	4.5	Dec.92	Jul.93	239
b' C	86	312	Mar.92	4.3	Jul.97		
C c'	81	393	Sep.97	4.1	Oct. 2001	July 2004	366
C' D	83	476	Dec.2001	4.2	March 2006		
D d'	84	560	May 2006	4.2	August 2010	July 2016	565

Based on extraction rate of 20 x 10^6 yd^3 superficials per annum. Note that mining schedule dates are for pit completion and superficials are required to be removed ahead of that date.

All volumes in 10^6 b yd^3

KJW/JT

[illegible] May, [illegible]

TABLE XXXVI

HAT CREEK II

COST OF SUPERFICIALS REMOVAL AND DUMPING
(COSTS IN $ 000's)

Item	No. off	Cost	Add 10% initial spares	Years Depreciation	Annual Depreciation including initial Spares	Average Investment	Interest on Investment (10%)	Total Cost	Cost of In pit Portion
Bucket Wheel Excavator (say Knupp C 500)	2	5000	5500	20	275	2888	289	564	564
Pit Bench Conveyor (2250 ft)	2	2480	2728	10	273	1500	150	423	423
Cross Collecting Conveyor (3000 ft)	2	3300	3630	10	363	1997	200	563	563
Main in pit conveyor (2250 ft)	5	6200	6820	10	682	3751	375	1057	1057
Spoil conveyor (S. Side) (16000 ft)	1	8800	9680	10	968	5324	532	1500	
Slowing conveyor (4000 ft)	1	2200	2420	10	242	1331	133	375	
Boom Stacker	1	3100	3410	20	170	1790	179	349	
Cables and O.H. line to S.Side		600	660	20	33	347	35	68	68
Electrics in pit - Substations	12	1200	1320	20	66	693	69	135	101
Trailing cables 1000 ft		248	273	2	137	205	21	158	126
Total ownership cost					3209			5192	2902

Operating Cost

Labour

BWE operator	8	
Conveyor mechanics	12	
Conveyor operators	16	
Stacker operators	4	
Engineering staff	12	
Total	52 at 2080 hrs each and $8.50 average cost	915
Spares and maintenance @ 20% depreciation		642
Power cost (8830 kW for 5000 hrs @ .011 $/kWh)		486
Lubricants etc.		64
Total Annual operating cost		2107
Total Cost		7299
Cost per yd^3 (bank) superficials at $20 \times 10^6 yd^3$ per annum		0.365 $

As stacker and associated belt are common to both shovel and BWE systems

Cost of BWE system in pit to deliver to main spoil conveyor is :-

Ownership cost	2902
Labour (31)	645
Spares maintenance	358
Power 5000 kW	275
Lubricants etc.	36
	4216

Cost/yd^3 (bank) to remove & transport superficials out of pit 0.211 $

HAT CREEK No. 2 DEPOSIT — CASH FLOW SCRAPERS V. BUCKET WHEEL EXCAVATORS — TABLE XXXVII

STAGE							1			2	3	4	5	6	
YEAR	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1977/78 TO 1982/83	1983/84	1984/85	1983/84 TO 1984/85	1985/86 TO 1992/93	1993/94 TO 2003/04	2004/05 TO 2015/16	2016/17 TO 2019/20	
SCRAPERS															
CAPITAL INVESTMENT $000's			14351	1367	15718	2730	34170	9150	—	9150	74440	64974	47820	5810	
DIRECT OPERATING COST								9770	9770	19540	84675	70312	51308	10497	
INSURANCE								184	183	367	2081	4401	4380	1162	
SUB TOTAL – CASH FLOW EXPENSES	—	—	14351	1367	15718	2730	34170	19104	9953	29057	161175	139687	103508	17469	
DISCOUNTED CASH FLOW AT 15%	—	—	9436	782	7815	1180	19213	7181	3254	10435	29442	6892	970	55	67007
COAL PRODUCTION 10^6 S.TONS	—	—	.1	.2	.4	.4	1.1	.43	7.6	11.9	105	144	157	40	
DISCOUNTED COAL PRODUCTION (15%)	—	—	.07	.12	.2	.17	.56	1.62	2.48	4.1	19.21	7.29	1.62	0.12	32.90

DISCOUNTED COST $= \frac{67007}{32900} =$ \$2.04/S.TON OR 19¢/10^6 BTU

BUCKET WHEEL EXCAVATORS															
CAPITAL INVESTMENT	—	39029	3717	4009	3745	4065	54565	—	273	273	26370	37533	26916	—	
DIRECT OPERATING COST								2107	2107	4214	16856	23177	25284	2107	
INSURANCE								738	678	1416	5478	7522	7969	1384	
SUB TOTAL – CASH FLOW EXPENSES	—	39029	3717	4009	3745	4065	54565	2845	3058	5903	48704	68232	60169	3491	
DISCOUNTED CASH FLOW AT 15%	—	29510	2444	2292	1862	1757	37865	1069	1000	2069	8971	3272	635	12	52824

DISCOUNTED COAL PRODUCTION AS ABOVE

DISCOUNTED COST $= \frac{52824}{32900} =$ \$1.61/S.TON OR 15¢/10^6 BTU

COST DIFFERENCE = \$0.43/S.TON OR 4¢/10^6 BTU.

TABLE XXXVIII

COAL REQUIREMENTS FOR DIFFERENT DEGREES OF WASHING - BASED ON LORING TESTS ON SAMPLE FROM BOREHOLE NO 75-74

	Rom coal as assumed in this report (unwashed)	Laboratory Sample		
		Unwashed	Washing to 15% ash	Washing to 10% ash
At boilers:				
Calorific value, (Btu/lb)	5,500	6,700	8,000	8,800
Moisture, %	20	20	20	20
Ash, %	32	22.4	15	10
Boiler feed requirement, (10^6 tpa)	13.1	10.7	9.0	8.2
At washery:				
Estimated yield, %	100	100	80	56
Washery rejects, (10^6 tpa)	0	0	2.3	6.4
Washery feed (ie rom) requirement, (10^6 tpa)	13.1	10.7	11.3	14.6
In pit:				
Pit rejects, (10^6 tpa)*	2.3	1.9	2.0	2.6
In-situ coal requirement (10^6 tpa)*	15.4	12.6	13.3	17.2

* based on same selective mining as discussed in Chapter III

Note: Sample (dry basis):-

CV - 8,400 Btu/lb
Ash - 28%

TABLE XXXIX

COMPARISON OF WASTE PRODUCTION DUE TO WASHING

(10^6 tpa)

	Rom coal as assumed in this report (unwashed)	Laboratory Sample		
		Unwashed	Washing to 15% ash	Washing to 10% ash
Pit rejects	2.3	1.9	2.0	2.6
Washery rejects	-	-	2.3	6.4
Boiler dust and grit	4.2	2.4	1.4	0.8
Total rejects	6.5	4.3	5.7	9.8

HAT CREEK II — Schedule of Equipment — Fixed Installations — Table XL

Stage			Stage I					Total Stage I	Stage 2		Total Stage 2	Stage 3	Stage 4	Stage 5	Stage 6
												Totals by Stages			
Item			1978/79	1979/80	1980/81	1981/82	1982/83		1983/84	1984/85		1985/93	1994/2003	2004/2015	2016/20
YEAR	Quantity	Years for Depreciation	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's
COAL-HANDLING															
Conveyors out of pit		10	434	300	300	300	300	1634				1634	1634	1634	
Conveyors from coal stock		10				4656		4656				4656	4656	4656	
Reclaiming B.W.E.		20				3000		3000	3000		3000		6000		
Stacker (coal)		20	1000	1250	1000	1250	1000	5500	1250		1250		6750		
Crushers		5	1400	1200	1400	1200	1400	6600	1200	7800	9000	7800	15600	23400	
Interchange Station		40	2000					2000							
Dozer/Compactor		2	300		300		300	900		300	300	1200	1500	1800	600
Conveyor Extension (stage 2)	1500 ft x 3	10							2644		2644		5288	2644	
Pit Bottom Conveyors per stage	4000 ft x 3 per stage	10										5040	5425	5425	
Replacement cost conveyors														10465	
Sub Total			5134	2750	3000	10406	3000	24290	8094	8100	16194	20330	51893	50024	600
Waste and Ash Disposal															
Conveyors		10	7422					7422				22206	22206	22206	
Spreader		20	6200					6200				3100	6200	[illegible]	
Ash conveyors		10				110		110				110	110	110	
Dozer		2	140		140		140	420		140	140	560	700	940	280
Sub Total			13762		140	110	140	14152		140	140	25976	29216	26456	280
Miscellaneous															
Cables-Feeder		2	222		222		296	740	74	370	444	2143	[illegible]	4066	830
Cables-Power line		40	100			50		150				223	160	140	
Sub Transmission lines		40	126				25	151				525			
Transformers & Switchgear		40	15	70		20	135	240	35	310	345	455			
Lighting, Communications		40	125		125	10	125	385		135	135	[illegible]	125		
Sub Total			618	70	341	110	624	1801	[illegible]	815	984	[illegible]	4752	4206	830
Annual Total			19514	2820	3481	10686	3826	[illegible]	[illegible]	9115	17318	53033	85810	80486	1710
Initial Spares			1951	282	282	1040	317	3873	341	58	405	2483	561	551	—
Grand Total			21465	3102	3770	11726	4152	[illegible]	3050	3173	18223	52522	86431	81037	1710
Total Depreciation including initial spares			2020	2080	2110	2340	4270	12820	5110	5170	10280	45200	72300	96640	22380

TS 605 HAT CREEK No. 2 DEPOSIT 5/5/76

TABLE XLI

MINE WASTE AND POWER STATION ASH DISPOSAL

PIT STAGE	MARBLE BANK VOLUME 10^6 Yd^3 STAGE	MARBLE BANK VOLUME CUM.	MARBLE LOOSE VOLUME 10^6 Yd^3 STAGE	MARBLE LOOSE VOLUME CUM.	VOLCANICS BANK VOLUME 10^6 Yd^3 STAGE	VOLCANICS BANK VOLUME CUM.	VOLCANICS LOOSE VOLUME 10^6 Yd^3 STAGE	VOLCANICS LOOSE VOLUME CUM.	SUPERFICIALS BANK VOLUME 10^6 Yd^3 STAGE	SUPERFICIALS BANK VOLUME CUM.	SUPERFICIALS LOOSE VOLUME 10^6 Yd^3 STAGE	SUPERFICIALS LOOSE VOLUME CUM.
1	–	–	–	–	–	–	–	–	19	19	24	24
2	–	–	–	–	–	–	–	–	41	60	51	75
3	–	–	–	–	–	–	–	–	179	239	224	299
4	–	–	–	–	95	95	142	142	127	366	159	458
5	–	–	–	–	56	151	84	226	199	565	249	707
6	–	–	–	–	43	194	65	291	161	726	201	908
7	–	–	–	–	101	295	151	442	125	851	156	1064
8	26	26	39	39	72	367	108	550	74	925	92	1156
9	642	668	963	1002	865	1232	1298	1848	1492	2417	1865	3021

PIT STAGE	OVERLYING AND SEGREGATED WASTE BANK VOLUME 10^6 Yd^3 STAGE	OVERLYING AND SEGREGATED WASTE BANK VOLUME CUM.	OVERLYING AND SEGREGATED WASTE LOOSE VOLUME 10^6 Yd^3 STAGE	OVERLYING AND SEGREGATED WASTE LOOSE VOLUME CUM.	R.O.M. COAL PRODUCTION 10^6 s. tons STAGE	R.O.M. COAL PRODUCTION CUM.	DRY ASH PRODUCTION 10^6 s. tons STAGE	DRY ASH PRODUCTION CUM.	CONDITIONED ASH PRODUCTION WEIGHT 10^6 s. tons STAGE	CONDITIONED ASH PRODUCTION WEIGHT CUM.	CONDITIONED ASH PRODUCTION LOOSE VOLUME 10^6 Yd^3 STAGE	CONDITIONED ASH PRODUCTION LOOSE VOLUME CUM.	TOTAL WASTE 10^6 LOOSE Yd^3 STAGE	TOTAL WASTE CUM.
1	10	10	14	14	2	2	—	—	–	–	–	–	38	38
2	14	24	20	34	17	19	5	5	6	6	5	5	76	114
3	100	124	140	174	99	118	32	37	38	44	32	37	396	510
4	103	227	144	318	146	264	47	84	55	99	46	83	491	1001
5	125	352	175	493	176	440	56	140	66	165	55	138	563	1564
6	152	504	213	706	92	532	29	169	34	199	28	166	507	2071
7	195	699	273	979	82	614	26	195	31	230	26	192	606	2677
8	159	858	223	1202	50	664	16	211	19	249	16	208	478	3155
9	5478	6336	7669	8871	2733	3397	875	1086	1029	1278	858	1066	12653	15808

NOTES

1. BANK VOLUMES FROM TABLE
2. SWELL — MARBLE 50%. VOLCANICS 50%. SUPERFICIALS 25%. OVERLYING AND SEGREGATED WASTE 40%.
3. DRY ASH 32% OF ROM COAL (BY WEIGHT)
4. ASH CONDITIONED TO 15% MOISTURE
5. LOOSE DENSITY OF CONDITIONED ASH 1·2 s.tons/Yd^3.

TABLE XLII

TOTAL DUMPING SPACE AVAILABLE
Units - 10^6 yd^3

Elevation (ft	No. 3 Dump		No. 4 Dump		No. 5 Dump		No. 6 Dump		Total	
	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.
3,200-3,300	-	-	8	8	-	-	-	-	8	8
3,300-3,400	-	-	29	37	-	-	-	-	29	37
3,400-3,500	-	-	52	89	-	-	-	-	52	89
3,500-3,600	11	11	100	189	-	-	-	-	111	200
3,600-3,700	21	32	143	332	-	-	-	-	164	364
3,700-3,800	27	59	197	529	21	21	-	-	245	609
3,800-3,900	33	92	244	773	105	126	1	1	383	992
3,900-4,000	36	128	307	1,080	214	340	13	14	570	1,562
4,000-4,100	-	128	-	1,080	331	671	50	64	381	1,943
4,100-4,200	-	128	-	1,080	434	1,105	99	163	533	2,476
4,200-4,300	-	128	-	1,080	528	1,633	143	306	671	3,147
4,300-4,400	-	128	-	1,080	604	2,237	193	499	797	3,944
4,400-4,500	-	128	-	1,080	676	2,913	247	746	923	4,867

Refer to Plate 77

TABLE XLIII

DUMPING SPACE AVAILABLE IN DUMPS 4 AND 5

Units - 10^6 yd^3

Elevation (ft)	Dump No. 4		Dump No. 5		Total	
	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.
3,200-3,300	8	8	-	-	8	8
3,300-3,400	29	37	-	-	29	37
3,400-3,500	52	89	-	-	52	89
3,500-3,600	100	189	-	-	100	189
3,600-3,700	143	332	-	-	143	332
3,700-3,800	197	529	21	21	218	550
3,800-3,900	244	773	105	126	349	899
3,900-4,000	307	1,080	214	340	521	1,420
4,000-4,100	-	1,080	331	671	331	1,751
4,100-4,200	-	1,080	434	1,105	434	2,185
4,200-4,300	-	1,080	528	1,633	528	2,713
4,300-4,400	-	1,080	604	2,237	604	3,317
4,400-4,500	-	1,080	676	2,913	676	3,993

Refer to Plate 78

TABLE XLIV

HAT CREEK NO. 2 DEPOSIT

DUMPING SPACE AVAILABLE IN DUMPS 4, 5 AND 6

Units - 10^6 yd^3

Elevation (ft)	No. 4 Dump		No. 5 Dump		No. 6 Dump		Total	
	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.	Elev.	Cum.
3,200-3,300	8	8	-	-	-	-	8	8
3,300-3,400	29	37	-	-	-	-	29	37
3,400-3,500	52	89	-	-	-	-	52	89
3,500-3,600	100	189	-	-	-	-	100	189
3,600-3,700	143	332	-	-	-	-	143	332
3,700-3,800	197	529	21	21	-	-	218	550
3,800-3,900	244	773	105	126	1	1	350	900
3,900-4,000	307	1,080	214	340	13	14	534	1,434
4,000-4,100	-	1,080	331	671	50	64	381	1,815
4,100-4,200	-	1,080	434	1,105	99	163	533	2,348
4,200-4,300	-	1,080	528	1,633	143	306	671	3,019
4,300-4,400	-	1,080	604	2,237	193	499	797	3,816
4,400-4,500	-	1,080	676	2,913	247	746	923	4,739

Refer to Plate 79

TABLE XLV

HAT CREEK DIVERSION - ESTIMATED COST

	Amount $
Ponding dam and spillway	350,000
Dam outlet culvert	60,000
Western canal	1,775,000
Pit-wall conduit	76,000
Eastern and southern ditches	50,000
Sub-total	2,311,000
Engineering and contingencies ±25%	579,000
Total	2,890,000

Notes:

1. The amount included for the western canal provides for a one-lane maintenance road adjacent to the canal. Widening of this road for the relocation of the Hat Creek road would cost an additional $200,000, very approximately.

2. It is emphasised that these estimates are very approximate due to lack of detailed hydrological, topographic and geological information.

HAT CREEK II Schedule of Equipment — Infrastructure Table XLVI

Stage		Stage I						Stage 2		Stage 1	Stage 2	Stage 3	Stage 4	Stage 5	Stage 6
										Totals by Stages					
YEAR		1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1983/84	1984/85	1977/83	1983/85	1985/92	1992/2003	2004/2015	2016/20
Item	Unit Cost	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's	$ 000's
Hat Creek Diversion															
Dam and spillway	350	350								350					
Dam Outlet Culvert	60	60								60					
Western Canal	1775	1775								1775					
Pit wall conduit	70	70								70					
East and South ditches	50	50								50					
Contingencies	579	579								[illegible]					
Sub Total	2890	2890								[illegible]	-	-	—	—	—
Road Relocation	200	200								[illegible]					
Buildings & Roads															
Administration Block	286	286								[illegible]					
Change House	236	236								[illegible]					
Shops & Warehouse	3805	805	3000							3805					
Core Sheds	4	4	4	4	4	4	4	4	4	24	8	32	52	[illegible]	-
Magazines	20	20	[illegible]					30							
Roads	860	230	240	230	30	30	30			790		70	70	70	—
Sub Total		1561	3264	234	34	34	34	24	4	5161	8	122	122	[illegible]	—
Services															
Power & Water Supply	615	200	315	100						[illegible]					
Buses	15	30		30	60 (R)	30 (R) 30		30 (R) 60		150	60	330	540	450	60
Sewage Disposal	50	25	25							50					
Pick-Ups	6	30		30		30 (R) 30		30 (R)	30	120	60	290	510	540	90
Graders	168	168		168	168 (R)	168 (R) 168	168 (R)	338 (R) 168	168 (R)	1008	672	2688	3864	4032	840
Sub Total		453	340	328	—	228	—			1349					
				/											
Initial Spares		35	55	70	70	70	70		10	370	10	10	—	-	—
Total Capital Costs		5139	3659	632	104	332	104	262	44	9970	206	232	[illegible]	106	—
Total Expenditure including replacements		5139	3659	632	[illegible]	560	[illegible]	658	[illegible]	[illegible]	810	[illegible]	[illegible]	[illegible]	990
Depreciation															
Civils		122	131	132	132	133	133	134	134	786	268	[illegible]	[illegible]	[illegible]	[illegible]
Total annual depreciation including mobile equipment		222	232	323	325	436	438	539	546	1397	1084	[illegible]	[illegible]	[illegible]	[illegible]

TABLE XLVII

HAT CREEK No. 2 DEPOSIT — SUMMARY OF ELECTRICAL ~~[illegible]~~ COSTS ($1000's)

STAGE / YEAR	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 (1977/78 to 1982/83)	1983/84	1984/85	2 (1983/84 to 1984/85)	3 (1985/86 to 1992/93)	4 (1993/94 to 2003/04)	5 (2004/05 to 2015/16)	6 (2016/17 to 2019/20)
MOBILE MINE EQUIPMENT	–	–	40	50	100	100	290	140	160	300	1880	4290	4060	1440
FIXED INSTALLATIONS AND INFRASTRUCTURE	–	17	376	610	694	845	2542	971	1075	2066	9862	15595	17505	5912
TOTAL PER STAGE/YEAR	–	17	416	660	794	945	2832	1131	1235	2366	11742	19885	21965	7352
AVERAGE COST $/S.TON								0.26	0.16	20	11	14	14	18
AVERAGE COST ¢/10⁶ BTU								2	1	2	1	1	1	2

HAT CREEK No. 2 DEPOSIT — LABOUR SCHEDULE AND PAYROLL COSTS — TABLE XLVIII

CATEGORY	HOURLY RATE	RATE WITH FRINGE BENEFITS	ANNUAL RATE ($)	1978/79 No	1978/79 ($000) COST	1979/80 No	1979/80 COST	1980/81 No	1980/81 COST	1981/82 No	1981/82 COST	1982/83 No	1982/83 COST	1 (1979/80 TO 1982/83) No.	1 COST	1983/84 No	1983/84 COST	1984/85 No.	1984/85 COST	2 (1983/84 TO 1984/85) No.	2 COST	3 (1985/86 TO 1992/93) No	3 COST	4 (1993/94 TO 2003/04) No	4 COST	5 (2004/05 TO 2015/16) No.	5 COST	6 (2016/17 TO 2019/20) No	6 ($000) COST
MOBILE MINING EQUIPMENT																															
EQUIPMENT OPERATORS	6.90	8.30	17300	84	1458	176	3045	176	3045	176	3045	178	3079		13672	193	3339	200	3460		6791		32972		51141		52140		14659
MAINTENANCE PERSONNEL	7.20	8.70	18100	72	1310	141	2552	141	2552	142	2570	144	2606		11590	156	2824	160	2896		5720		27676		43026		43773		12331
LABOURERS	6.45	7.80	16200	20	330	33	535	33	535	33	535	33	535		2470	36	583	40	648		1231		5966		9253		9035		2653
OVERTIME					402		608		609		638		640		2897		733		760		1493		7244		11131		11431		3226
SUB TOTAL				176	3500	350	6740	350	6741	351	6788	355	6860	1582	30629	385	7479	400	7764	785	15243	480	73858	544	114551	510	116803	477	32964
FIXED INSTALLATIONS																															
EQUIPMENT OPERATORS	6.90	8.30	17300	11	190	11	190	14	253	16	277	16	277		1187	16	277	16	277		554		7291		12975		15778		5259
MAINTENANCE PERSONNEL	7.20	8.70	18100	7	127	7	127	7	127	8	145	8	145		671	8	145	8	145		290		4026		5798		5973		1931
LABOURERS	6.45	7.80	16200	12	194	12	194	11	178	11	178	11	178		922	11	178	11	178		356		4515		5670		6804		2268
OVERTIME					17		17		18		24		24		100		24		24		48		420		2764		2165		722
SUB TOTAL				30	528	30	528	32	576	35	624	35	624	162	2880	35	624	35	624	70	1245	118	19232	142	27201	144	30720	144	10240
INFRASTRUCTURE																															
EQUIPMENT OPERATORS	6.90	8.30	17300	4	69	4	69	4	69	4	69	4	69		345	4	69	4	69		138		1104		1730		2076		
MAINTENANCE PERSONNEL	7.20	8.70	18100	2	36	2	36	2	36	2	36	2	36		180	2	36	2	36		72		576		1066		1303		620
LABOURERS	6.45	7.80	16200	10	162	10	162	10	162	10	162	10	162		810	10	162	10	162		324		2592		3888		4277		589
OVERTIME					13		13		13		13		13		65		13		13		26		224		294		192		1277
																													53
SUB TOTAL				16	280	16	280	16	280	16	280	16	280	80	1400	16	280	16	280	32	560	32	4496	40	6998	40	7848	34	2344
TOTAL COST ($1000)					4308		7548		7597		7692		7764		34909		8383		8668		17051		96586		148750		155371		45438
TONS $\times 10^6$																	4.5		7.6		11.9		105		144		157		40
COST $/S TON																	1.095		1.14		1.43		0.92		1.03		0.99		1.14
¢/10^6 BTU																	18		10		13		8		9		9		10
ANNUAL EMPLOYEES				222		396		398		400		406				436		451				630		726		694		655	
EMPLOYEE HOUSING																															
TRAILER CAMPS					400		400								800														
PERMANENT STRUCTURES							1000		3000		3000		2000		9000		2000		1500		3500								
LAND DEVELOPMENT									500		500		500		1500		500				500								
TOTAL					400		1400		3500		3500		2500		11300		2500		1500		4000								

3/5/76

TABLE 12

HAT CREEK No. 2 DEPOSIT

MATERIALS AND FUEL COST (Excluding Electrical Power) SUMMARY (1975 PRICES)

STAGE							1			2	3	4	5	6
YEAR	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1977/78 TO 1982/83	1983/84	1984/85	1983/84 TO 1984/85	1985/86 TO 1992/93	1993/94 TO 2003/04	2004/05 TO 2015/16	2016/17 TO 2019/20
MOBILE MINING EQUIPMENT INCLUDING EXPLOSIVES AND EXPLORATORY DRILLING (BY CONTRACT)	450	450	14146	14156	14274	14520	57996	15361	16189	31550	156566	237965	240873	67509
FIXED INSTALLATIONS	—	150	301	317	317	317	1402	660	660	1320	21197	36068	55489	19488
INFRASTRUCTURE	147	147	147	147	147	147	882	147	263	410	2152	3097	3432	1076
ENGINEERING AND ADMINISTRATION	—	20	20	20	20	33	113	34	37	71	336	533	1146	498
TOTAL PER STAGE OR YEAR	597	767	14614	14640	14758	15017	60393	16202	17149	33351	180251	277663	300940	88571
COST ¢/S.TON								3.77	2.26	2.80	1.72	1.93	1.92	2.21
COST ¢/MILLION BTU								3	2	3	2	2	2	2

3/6/76 HAT CREEK No. 2 DEPOSIT

TABLE L

DIRECT OPERATING COSTS SUMMARY

	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1977-78 to 1982-83	1983/84	1984/85	2 1983-84 to 1984-85	3 1985-86 to 1992-93	4 1993-94 to 2003-04	5 2004-05 to 2015-16	6 2016-17 to 2019-2020
MOBILE MINING EQUIPMENT														
LABOUR	–	3500	6740	6741	6788	6860	30629	7479	7764	15243	73858	114584	116802	32905
MATERIALS, FUEL & MISC.	450	450	14146	14156	14274	14520	57996	15361	16189	31550	156[illegible]	237965	240873	67509
ELECTRIC POWER	–	–	40	50	100	100	290	140	160	300	1880	4290	4460	1440
SUB-TOTAL	450	3950	20926	20947	21162	21480	88915	22980	24113	47093	232303	356809	362135	101854
FIXED INSTALLATIONS														
LABOUR	–	528	528	576	624	624	2880	624	624	1248	18232	27201	30720	10240
MATERIALS + FUEL	–	150	301	317	317	317	1402	660	660	1320	21197	36068	55489	19488
ELECTRIC POWER	–	17	376	610	694	845	2542	991	1075	2066	9862	15595	17505	5912
SUB-TOTAL	–	695	1205	1503	1635	1786	6824	2275	2359	4634	49291	78864	103714	35640
INFRASTRUCTURE														
LABOUR	–	280	280	280	280	280	1400	280	280	560	4496	6998	7848	2344
MATERIALS + FUEL	147	147	147	147	147	147	882	147	263	410	2152	3097	3432	1076
ELECTRIC POWER (INCLUDED ABOVE)			–	–	–									
SUB-TOTAL	147	427	427	427	427	427	2282	427	543	970	6648	10095	11280	3420
ENGINEERING + ADMIN														
SALARIES	–	552	588	799	904	1221	4064	1476	1656	3132	13248	19872	24840	8280
MATERIALS	–	20	20	20	20	33	113	34	37	71	356	533	1146	498
ELECTRIC POWER — INCLUDED IN INFRASTRUCTURE														
SUB-TOTAL	–	572	608	819	924	1254	4177	1510	1693	3203	13584	20405	25986	8778
CONSULTANT FEES	–	500	500	500	500	500	2500	250	250	500	2000	2750	3000	1000
TOTAL	597	6144	23666	24196	24648	25447	104698	27442	28958	56400	303826	468923	506115	150692
TONNAGE S.TONS $\times 10^6$								4.3	7.6	11.9	105	144	157	40
COST/S.TON $								6.38	3.81	4.7[illegible]	2.89	3.26	3.22	3.77
COST/MILLION BTU ¢								58	35	43	26	30	29	34
TOTAL VOL. MATERIAL REMOVED 10^6 byd^3	–	–	–	–	–	–	109	–	–	64	305	419	449	138
AVERAGE COST/byd^3 REMOVED	–	–	–	–	–	–	0.96	–	–	0.88	1.00	1.12	1.13	1.09

HAT CREEK II 28/5/76

DEPRECIATION SUMMARY

TABLE L1

	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1977-78 to 1982-83	1983/84	1984/85	2 1983-84 to 1984-85	3 1985-86 to 1992-93	4 1993-94 to 2003-04	5 2004-05 to 2015-16	6 2016-17 to 2019-2020
MOBILE MINING EQUIPMENT	—	—	11640	11640	11730	11830	46840	10180	10630	20810	120350	182710	189770	53440
FIXED INSTALLATIONS	—	2020	2380	2770	3840	4270	15280	5110	5170	10280	115300	82900	96640	32980
INFRASTRUCTURE	223	232	333	335	436	438	1997	538	546	1084	11464	6259	6756	2236
OTHER CAPITAL	2485	2485	2485	2485	2485	2485	14910	2485	2485	4970	19880	27335	29820	9940
TOTAL	2708	4737	16838	17230	18491	19023	79027	18313	18831	37144	189994	299204	322986	98596
COST / SHORT TON $ 000's	–	–	–	–	–	–	–	4.26	2.48	3.12	1.81	2.08	2.06	2.46
COST / MILLION B.T.U ¢	–	–	–	–	–	–	–	39	23	28	16	19	19	22

HAT CREEK II 28/5/76

CAPITAL INVESTMENTS INTEREST DURING CONSTRUCTION

INTEREST AND INSURANCE (1976 PRICES)

TABLE L11

		1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 (1977-78 to 1982-83)	1983/84	1984/85	2 (1983-84 to 1984-85)	3 (1985-86 to 1992-93)	4 (1993-94 to 2003-04)	5 (2004-05 to 2015-16)	6 (2016-17 to [illegible])
	CAPITAL INVESTMENT														
1	MOBILE MINING EQUIPMENT	—	—	30030	30	21670	220	51950	21480	2410	23890	129710	196760	189550	45220
2	FIXED INSTALLATIONS	—	21460	3100	3770	11720	4150	44200	9050	9170	18220	52310	56440	81040	1720
3	INFRASTRUCTURE	5139	3659	632	332	560	272	10594	658	212	870	3580	5096	5128	990
4.	SUB-TOTAL EQUIPMENT COSTS	5139	25119	33762	4132	33950	4642	106744	31188	11792	42980	185800	258296	275718	47930
5	DIRECT OPERATING COSTS TO START UP	597	6144	23666	24196	24648	25447	104698							
5A	INSURANCE	103	502	675	83	679	93	2135							
6.	WORKING CAPITAL	1187	1187	1187	1187	1187	1187	7122	1187	1187	2374				
7.	TOTAL CAPITAL COST	7026	32952	59290	29598	60464	31369	220699	32375	12979	45354	185800	288296	275718	47930
8.	CORPORATE OVERHEAD	1539	1839	1839	1839	1539	1839	11034							
9.	TOTAL CAPITAL COST INCLUDING CORPORATE OVERHEAD	8565	34791	61129	31437	62303	33208	231733	32375	12979	45354	185800	288296	275718	47930
10.	CUMULATIVE CAPITAL INCL. CORP. OVHD	8865	43656	104785	136222	198525	231733	231733	264108	277087	277087	462887	751153	1026901	1074831
11.	INTEREST ON CUM. CAP. COST UP TO BEGINNING OF YEAR (10%)	—	887	4366	10479	13622	19853	49207							
12.	INTEREST ON CAPITAL COST IN YEAR (5%)	443	1740	3056	1572	3115	1660	11586							
13.	TOTAL INTEREST DURING CONSTRUCTION	443	2627	7422	12051	16737	21513	60793							
14	TOTAL INVESTMENTS	9308	37418	68551	43488	79040	54721	292526	32375	12979	45354	185800	288296	275718	47930
15	CUMULATIVE INVESTMENT	9308	46726	115277	158765	237805	272526	292526	324901	337880	337880	523680	811976	1087694	1135624
16	DEPRECIATION	2708	4737	16838	17230	18491	19023	79027	18313	18831	37144	189994	299204	322986	98596
17.	CUMULATIVE DEPRECIATION	2708	7445	24283	41513	60004	79027	79027	97340	116171	116171	306165	605369	928355	1026951
20.	OUTSTANDING INVESTMENT YEAR/STAGE END	6600	39281	79974	117252	177801	213499		227561	221709					
21.	AVERAGE OUTSTANDING INVESTMENT	3300	22941	65138	104123	147526	195650	538678	220530	224635	445164	1722387	2254655	2124616	533349
22.	INTEREST ON AVERAGE OUTSTANDING INVESTMENT (10%)	330	2294	6514	10412	14753	19565	53868	22033	22464	44517	172238	225466	212462	53335
22	INSURANCE (2%)	66	459	1303	2082	2951	3913	10774	4411	4493	8904	34446	45093	42491	10667

HAT CREEK II 27/5/76

COAL PRODUCTION COST (R.O.M.)

(× $1000) (1975 Prices)

TABLE LIII

		1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1977-78 TO 1982-83	1983/84	1984/85	2 1983-84 TO 1984-85	3 1985-86 TO 1992-93	4 1993-94 TO 2003-04	5 2004-05 TO 2015-16	6 2016-17 TO 2019-2020
1	COAL PRODUCTION ROM 10^6	—	—	—	—	—	—	—	4.3	7.6	11.9	105	144	157	40
2	DIRECT OPERATING COST	—	—	—	—	—	—	—	27442	28958	56400	303 826	468923	506 115	150 692
3	DEPRECIATION	2708	4737	16 838	17 230	18 491	19 023	79 027	18 313	18 831	37 144	189 994	299 204	322 986	98 596
4	INTEREST ON AVERAGE INVESTMENT	—	—	—	—	—	—	—	22 053	22 464	44517	172 238	225 466	212 462	53 335
5	INSURANCE	—	—	—	—	—	—	—	4 411	4 493	8 904	34 448	45 093	42 491	10 667
6	ROYALTY (67¢/ton)	—	—	—	—	—	—	—	2 880	5 090	7970	70 350	96 480	105 190	26 800
	PRODUCTION COST (2+3+4+5+6)	—	—	—	—	—	—	—	75 099	79 836	154 935	770 856	1135 166	1 189 244	340 090
	PRODUCTION COST/TON ($)	—	—	—	—	—	—	—	17.46	10.50	13.02	7.34	7.88	7.57	8.50
	PRODUCTION COST/10^6 BTU (¢)	—	—	—	—	—	—	—	159	95	118	67	72	69	77

7/6/76 Amended as shown
Please destroy previous copy.

CASH FLOW (EXPENSES) AND UNIFORM SELLING PRICE

($'000)

TABLE LIV

	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1977-78 to 1982-83	1983/84	1984/85	2 1983-84 to 1984-85	3 1985-86 to 1992-93	4 1993-94 to 2003-04	5 2004-05 to 2015-16	6 2016-17 to 2019-2020
CAPITAL INVESTMENT	9308	37418	68551	43488	79040	54721	272526	32375	12979	45354	185500	288296	275715	47930
DIRECT OPERATING COST	—	—	—	—	—	—	—	27442	25958	56400	302826	467923	506115	150692
INSURANCE	—	—	—	—	—	—	—	4411	4493	8904	34448	45093	42491	10667
ROYALTY	—	—	—	—	—	—	—	2880	5090	7970	70350	96480	105190	26800
CASH FLOW EXPENSES	9308	37418	68551	43488	79040	54721	272526	67108	51520	118628	594424	878792	929514	236089
DISCOUNTED CASH FLOW (@15%)	8094	28288	45072	24862	39291	23656	169263	25226	16842	42068	107564	45557	9277	732 (374461)
COAL PRODUCTION 10^6 TONS	—	—	.1	.2	.4	.4	1.1	4.3	7.6	11.9	105	144	157	40
DISCOUNTED COAL PRODUCTION @ 15% 10^6 TONS	—	—	.07	.12	.2	.17	.56	1.62	2.48	4.1	19.21	7.29	1.62	0.12 (32.90)
UNIFORM SELLING PRICE	$= \frac{374\,461\,000}{32.90 \times 10^6}$ = \$ 11.38 /S.TON = 104¢/10^6 BTU													
DISCOUNTED CASH FLOW (@ 10%)	8462	30907	51482	29702	47084	30863	200500	34426	24060	58486	183229	114317	39718	4579 (600849)
DISCOUNTED COAL PRODUCTION (@ 10%)	—	—	.08	.14	.25	.23	.70	2.21	3.55	5.76	32.62	18.51	6.80	0.80 (65.19)
UNIFORM SELLING PRICE	$= \frac{600\,849\,000}{65.19 \times 10^6}$ = \$ 9.22 /S.TON = 84¢/10^6 BTU													

HAT CREEK II 27/5/76

(ROM)

COAL PRODUCTION COST (INFLATED)

($ × 1000)

TABLE LV

	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1978 to 1982-83	1983/4	1984/5	2 [illegible] to 1994-95	3 1985 86 to 1992-93	4 1993-94 to 2003-04	5 2004-05 to 2015-16	6 2016-17 to 2019-2020
COAL PRODUCTION R.O.M. 10^6	–	–	–	–	–	–	–	4.3	7.6	11.9	105	144	157	40
DIRECT OPERATING COSTS	–	–	–	–	–	–	–	27442	28958	56400	303826	468923	506115	150692
DEPRECIATION	2708	4737	16838	17230	18491	19023	79027	18313	18831	37144	189994	299204	322986	98596
INTEREST ON AVERAGE INVESTMENT	–	–	–	–	–	–	–	22053	22464	44517	172238	225466	212462	53335
INSURANCE	–	–	–	–	–	–	–	4411	4493	8904	34448	45093	42491	10667
SUB-TOTAL	2708	4737	16838	17230	18491	19023	79027	72219	74746	–	–	–	–	–
INFLATION FACTOR	1.21	1.33	1.46	1.54	1.61	1.69	–	1.78	1.87	–	–	–	–	–
SUB-TOTAL INFLATED	3277	6300	24583	26534	29771	32149	122614	128550	139775	268325	1641846	3890423	7143202	2991005
ROYALTY (67¢/s.ton)	–	–	–	–	–	–	–	2880	5090	7970	70350	96480	105190	26800
PRODUCTION COST (INFLATED)	–	–	–	–	–	–	–	131430	144865	276295	1712196	3986903	7248392	3017806
COST /S.TON $	–	–	–	–	–	–	–	30.57	19.06	23.22	16.31	27.69	46.17	75.45
COST / 10^6 BTU ¢	–	–	–	–	–	–	–	278	173	211	148	252	420	686

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TABLE LVI

CASH FLOW (EXPENSES) AND UNIFORM SELLING PRICE (INFLATED)

	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1977-78 TO 1982-83	1983/84	1984/85	2 1983-84 TO 1984-85	3 1985-86 TO 1992-93	4 1993-94 TO 2003-04	5 2004-05 TO 2015-16	6 2016-17 TO 2019-2020
CAPITAL INVESTMENT	9308	37 418	38 551	43 488	79 040	54 721	292 526	32 375	12 979	45 354	185 800	288 296	275 718	47 930
DIRECT OPERATING COST	–	–	–	–	–	–	–	27 442	28 958	56 400	303 826	468 923	506 115	150 692
INSURANCE	–	–	–	–	–	–	–	4 411	4 493	8 904	34 448	45 093	42 491	10 667
SUB TOTAL (UNINFLATED)	9 308	37 418	38 551	43 488	79 040	54 721	292 526	64 228	46 430	110 658	524 074	802 312	824 324	209 289
INFLATION FACTOR	1.21	1.33	1.46	1.54	1.61	1.69	–	1.78	1.87	–	–	–	–	–
ROYALTY	–	–	–	–	–	–	–	2880	5 090	7970	70350	96 480	105 190	26 800
CASH FLOW (INFLATED)	11 263	49 766	56 284	66 972	127 254	92 478	404 017	117 206	91 914	209 120	1 294 610	3 408 694	5 563 877	2 024 331
DISCOUNTED CASH FLOW (15%)	9 794	37 623	37 007	38 288	63 258	39 978	225 948	44 058	30 047	74 105	226 186	146 131	51 361	6 210 (729 941)
COAL PRODUCTION 10^6 S. tons	–	–	·1	·2	·4	·4	1.1	4.3	7.6	11.9	105	144	157	40
DISCOUNTED COAL PRODUCTION AT 15% 10^6 S. tons	–	–	0.07	0.12	0.2	0.17	0.56	1.62	2.48	4.1	19.21	7.29	1.62	0.12 (32.90)

UNIFORM SELLING PRICE (WITH DISCOUNTING AT 15%) = $\frac{7299\,941\,000}{32.90 \times 10^6}$ = \$ 22.19/s. ton. OR 202 ¢/10^6 BTU.

	1977/78	1978/79	1979/80	1980/81	1981/82	1982/83	1 1977-78 TO 1982-83	1983/84	1984/85	2 1983-84 TO 1984-85	3 1985-86 TO 1992-93	4 1993-94 TO 2003-04	5 2004-05 TO 2015-16	6 2016-17 TO 2019-2020
DISCOUNTED CASH FLOW (AT 10%)	10 239	41 129	42 287	45 743	79 015	52 201	270 613	60 145	42 878	103 023	389 561	378 679	226 288	39 161 (1407325)
DISCOUNTED COAL PRODUCTION (AT 10%)	–	–	0.08	0.14	0.25	0.23	0.70	2.21	3.55	5.76	32.62	18.51	6.80	0.80 (65.19)

UNIFORM SELLING PRICE (WITH DISCOUNTING AT 10%) = $\frac{1407\,325\,000}{65.19 \times 10^6}$ = \$ 21.59/s. ton OR 196 ¢/10^6 BTU

TABLE LVII

COMPARISON OF OPENPIT NO 1 AND 2

	Openpit No 1	Openpit No 2	Unit
Minimum cover	0	25	ft
Maximum vertical height of pit slope:			
600-ft pit	1,150	2,100	ft
1,500-ft pit	2,500	3,000	ft
Elevation of pit floor:			
600-ft pit	2,400	2,900	ft
1,500-ft pit	1,500	2,000	ft
Area of excavation:			
600-ft pit	2,000	4,000	acres
1,500-ft pit	5,000	10,000	acres
Approximate maximum area of disturbance:			
600-ft pit	8,000	20,000	acres
Rom coal reserves within:			
600-ft pit	385	664	10^6 short ton
1,500-ft pit	775	3,397	10^6 short ton
Total waste rock within:			
600-ft pit	885	2,176	10^6 bank yd^3
1,500-ft pit	1,701	10,653	10^6 bank yd^3
Overall stripping ratio:			
600-ft pit	2.3	3.3	bank yd^3/short ton rom
1,500-ft pit	2.2	3.1	bank yd^3/short ton rom
Instantaneous stripping ratio at pit limits:			
600-ft pit	7.7	11.0	bank yd^3/short ton rom
1,500-ft pit	13.7	15.4	bank yd^3/short ton rom
Capital investment to start up (600-ft pit)	134	292	10^6 $
Uniform selling price (mean) (600-ft pit):			
10% discount	5.63	9.22	$/short ton rom
15% discount	6.35	11.38	$/short ton rom
On thermal basis:			
10% discount	51	84	¢/10^6 Btu
15% discount	58	104	¢/10^6 Btu