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CONFIDENTIAL

THE ELK RIVER COKING-COAL PROJECT

British Columbia, West Canada

Elaboration of a Mining Concept with Consideration Being Given to a Necessary Quality Control of the Raw Coal Production, Including Costing as well as Suggestions for Additionally Required Exploratory Work on the Basis of the existing Feasibility Study Dated 1971, Made Available by Emkay-Scurry.

Duesseldorf, October 1973

EXPLORATION UND BERGBAU GMBH

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1. INTRODUCTION

1.1 Geographical Location and Traffic Communications

The Elk River coal deposit is located in the southeast of British Columbia, West Canada, in the upper Elk River Valley, approximately 45 miles north of Sparwood, the site of the Balmer Mine belonging to Kaiser Resources Ltd., or 20 miles to the north of Elkford, the mining settlement established in connection with the exploitation of the Fording River coal deposit of the Canadian Pacific Oil and Gas Ltd. (Annexe 1).

From the Elk River deposit there are two roads linking up with Calgary, Alberta. The one runs via Elkford - Sparwood - Highway No. 3 as far as Fort McLeod and Highway No. 2 as far as Calgary (a total of 225 miles, 22 miles of which to Elkford consisting of an unmade forest track and from Elkford 205 miles of asphalted roadway). The second road connection (approx. 45 miles of metalled road) as far as Highway No. 1 and approx. 55 miles along Highway No. 1 in an eastern direction to Calgary. (Total distance 100 miles).

The nearest possibility of linking up with the Canadian Pacific Railroad to Vancouver consists of a siding of roughly 47 miles in length which runs from Sparwood to the deposit. Approximately 35 miles of this would have to be re-laid by the Canadian Pacific in easily accessible terrain. The total distance to the shipping port near Vancouver is 745 miles.

1.2 The Geology

(derived from the report of Montan Consulting GmbH)

In the Elk River Valley area several coal seams are found in the shales of the Lower Cretaceous Kootenay-Formation.

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These seams are underlain by a sandstone horizon. The seam series which exhibit very considerable changes in terms of development and thickness, have been grouped together into 20 seams on the basis of field oberservations. The entire stratigraphical sequence strikes NNW-SSE, and in the concession area forms an syncline about 6 km in width, the folding of which took place during the Tertiary. On the eastern limb of the syncline the strata dip at an average angle of 40°, while on the western limb they are steeply inclined. On the edges of the U-shaped glacier eroded valley, the lower-lying seams crop out on the hill slopes, whereas the higher-lying seams in the valley bottom are covered to a varying degree by moraine materials. Apart from the basic Tertiary folding no major faulting has been oberserved. In the SE part of the concession, south of Weary Creek, the bedded sequence appears to have undergone overthrust folding and is presumably also faulted.

1.3 Exploratory Work and Ownership Conditions

On account of the favorable stratigraphy and exposure conditions the exploratory work was directed exclusively to the eastern part of the syncline, Big Weary Ridge and Little Weary Ridge, since only this part of the deposit appeared to be capable of open-pit development.

In the years from 1883 till 1967, during which the property changed hands, various minor-scale exploratory programmes were carried out in the area of Big Weary Ridge. In 1967 the concession under exploration (currently 19,200 acres = 77.7 km^2 of Crown Forest Land) was acquired by Scurry Rainbow Oil Ltd. (Annexe 2).

The first systematic exploration took place in 1968. It

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was conducted in the Big Weary Ridge area by the North American Coal Co. under an option agreement with Scurry Rainbow Oil Ltd. A total of I3 boreholes were sunk here, which presumably encountered fairly highly disturbed stratigraphical conditions. The option ceased to be maintained by the North American Coal Co. but the exploratory work was continued by Scurry Rainbow Oil Ltd. in 1969 in the area of Little Weary Ridge, where undisturbed bedding conditions were encountered.

In 1970, 50 % of the Scurry Rainbow Concession was acquired by the Emkay Canada National Resources Ltd., a 100 % subsidiary of the Morrison-Knudson Company Inc. In 1970 and 1971 an extensive exploratory programme, headed by Emkay, was carried out in the Little Weary Ridge and in the Elk River Valley.

All together, from 1969 on, some 81 boreholes (53 core and 28 rotary drilling), comprising some 15.000 metres drilled, were put down. To these must be added approximately 7000 m of trenches as also some prospecting tunnels and auger drillings in the side of the valley of Weary Creek.

The results of this exploration programme together with those of other exploratory programmes have been embodied in the Feasibility Study of Emkay-Scurry of March 1971, at present before us, and supplemented at the end of 1971 by a coking study of the coals from seams Nos. 13 to 19.

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2. The Coal Reserves

2.1 Reserves, Capable of Open-pit Development, in the Exploration Area A + B, down to a Depth of 400' below the so-called Dragline Bench

(Annexe 3)

At the same time of the completion of the Feasibility Study of Emkay-Scurry in March 1971, adequate results in terms of both coal reserves and coal qualities were only available for the so-called "Initial Study Area A" (i.e. seams Nos. 2 to 13 as well as parts of seams Nos. 14 and 15 in the area between the baselines 180+00 and 360+00. Accordingly, the open-pit concept planned by Emkay-Scurry was adapted to this area, and in this study only the reserves of this area were stated for each of the individual seams. Concerning the remaining part of seams Nos. 14 and 15 as well as seams Nos. 16 to 19 (which correspond to the so-called "Area B"), which were demonstrated during and after the completion of the Study by means of core drillings Nos. 30-C to 48-C and the trenches Nos. T-16, T-17 and T-18, there is only a global figure available, but no statement of the figures for each individual seam. Detailed particulars on the seams in Area B are available only for the coal qualities (see Coking Study of the Department of Energy, Mines and Resources, Ottawa, from November 1971).

As well be seen from the Emkay-Scurry Study and the Report of Montan Consulting GmbH, seams Nos. 2 to 19 (not workable beyond seam No. 20) exhibit highly fluctuating quality characteristics. For a qualitative appraisal of the total reserves in terms of their

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amenability to preparation as well as for every production planning that has to take into account the differences in quality characteristics, it is absolutely essential to have knowledge on the reserves of each individual seam. Since these details are not contained in the Emkay-Scurry Study, a re-calculation was carried out of the reserves of the area A + B, designated as capable of being developed as an open-pit operation (Annexes 4 A to 4 L).

The inclination of the seams and the thickness of the overburden were taken for the purpose of reserves calculation from the section drawings in the baselines 180+00 to 360+00, to a scale of 1 inch = 400 ft, available at Scurry Rainbow (Annexes 5 A to 5 K). The stratigraphical seam and overburden thicknesses were calculated from the borehole results and prospecting trench data via the dip angles recorded in the deposit cross-section in question.

For the calculation of the amounts of oxidized coal recourse was had by Emkay-Scurry to the results of the oxidation tests, according to which oxidation zones are expected to occur up to a vertical depth of 35 ft in the outcrop area of the seams on the flanks of Little and Big Weary Ridge, whereas in the area of the Elk River Valley, as a result of the protection afforded by the overlying moraine materials, an oxidation zone of only 15 ft in depth on an average can be reckoned with.

When computing the mining losses and the anticipated percentage of dirt in the raw coal mined, the following criteria were taken as a basis:

a) generally speaking, efforts should be made to see that the coal is loaded in the cleanest possible state (without country rock), so as not to depress the washery recovery unnecessarily.

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b) In the case of seams or coal beds of more than 5 ft in thickness and dirt bands of more than 3 ft thick, this will be possible if the cutting limit in shovelling is sited inside the coal seam. However, in using this method of working it will be necessary to reckon with a mining loss of approx. 1 ft per each individual seam along the contact zones. At the same time, it will not be possible to prevent approx. 1/2 feet of dirt per seam or coal measure from being loaded together with the coal, the cause of this being the loosening and intimate mixing of the coal and dirt, especially along the contact with the hanging-wall, in consequence of blasting. Dirt bands of less than 3 ft in thickness cannot be removed by the large-scale mechanical units employed in open-pits, but pass entirely into the raw coal production.

c) In the case of seams or coal measures of between 2 and 5 ft in thickness it was assumed that the mining loss would have to be reduced to the minimum as far as possible, if the winning of these seams was to be at all worth while. In order to attain this, the cut-off line, especially at the contact with the hanging-wall, should during loading operations be laid within the country rock. In doing so, it will be necessary to reckon with a dirt fraction in the raw coal output of around 1 ft per individual seam. Again, at the same time, due to the loosening and intimate mixing of the coal and dirt at the contact with the hanging-wall resulting from blasting work, it will not be possible to avoid a mining loss of about 1/2 ft per each individual seam. This estimate will apply in turn, subject to the condition that the partings are thicker than 3 ft. Partings of less than 3 ft are loaded together with the coal.

d) Seams and coal measures of below 2 ft in thickness were - provided that they were separated by dirt partings of more than 3 ft in thickness from thicker seams - termed non-workable and shown as complete mining losses.

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In the calculation of reserves carried out by Emkay-Scurry a specific weight of the raw coal of only 1.3 tons/m^3 (corresponding to around 1.1 sht/cyd) was used as reckoning factor. As a result of using such a low specific weight, which appears impossible to us for raw coal, a non-official safety factor was incorporated by Emkay-Scurry in the reserves-calculation. In the present calculation, following consultation with the Geological Department of Scurry Rainbow and Montan Consulting GmbH, Essen, the figure used by our Calgary Office for the average weight of the raw coal is 1.45 tons/m^3 (1.222 sht/cyd). In our opinion this estimate is more realistic, though still on the safe side, since specific weights of the raw coal of up to 1.5 tons/m^3 are by all means to be reckoned with. By employing a higher specific weight this automatically raises the tonnages of the raw coal reserves and reduces the overburden-coal ratio (cyd overburden: sht coal), compared with the Emkay-Scurry calculation.

Seams Nos. 2 to 15 are adequately exposed over the entire range from baseline 180+00 to 360+00, thanks to boreholes, trenches and prospecting tunnels, so that the reserves in these seams may be claused as "measured". Seams 16 to 19 are exposed only in the area of baselines 180+00, 260+00 and 340+00 by means of drillholes or trenches. For calculating the reserves of these seams it was necessary to interpolate seam and overburden thicknesses over distances of up to 8,000 ft. For this reason, the reserves computed in this way for the seams Nos. 16 to 19 still require to be confirmed by means of further core drillings to be sunk in the intervening baselines. (See Section 4 - Supplementary Exploratory Work).

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For the areas A + B, designated as capable of open-pit development, the figures listed below have been established for the reserves of raw coal, amounts of oxidized coal, mining losses, overburden tonnages as well as amounts of dirt in the raw coal produced.(See Annexe 4).

	bank cyd	sht	%
a) raw coal in situ	138.340.000	169.051.000	100
b) oxidized coal	5.986.500	7.315.500	4
c) mining loss	16.216.200	19.816.200	12
d) utilizable raw coal (= a - b - c)	116.137.300	141.919.800	84
e) overburden	742.452.300	-	-
f) percentages of dirt in the raw coal mined	19.715.600	-	-

From the above one arrives at a ratio of 4.39 bank cyd per each sht of raw coal in situ, or 5.39 bank cyd for the total of non-utilizable material (overburden + oxidized coal + mining loss) per sht of utilizable raw coal.

The above figures for overburden tonnages will be valid in the event of draglines being used in the deep open-pit. If the choice falls on excavator and truck operations, the amount of overburden to be dealt with also in the deep open-pit will increase by comparison with the drag-line operation (a) as a result of the greater number of working-levels within the range of the dragline boxcut and of the shallower slope angle resulting from this, and (b) as a result of the removal of the entire over-burden down to the deepest level, the increase being around 7 % to a total of 794,424,000 bank cyd. From this we arrive at a ratio of 4.7 bank cyd overburden per sht of raw coal in situ, or of 5.75 bank cyd of total non-economic material (overburden + oxidized coal + mining loss) per sht of economic raw coal. The reserves of raw coal in the areas A + B are spread over the individual seams as follows (see also Annexe 4 A):

Seam No.	sht	%
2	14.781.700	10,4
3	6.148.500	4,3
4A+4	24.088.800	17,0
6	2.862.200	2,0
7	3.549.500	2,5
8A+8	12.609.400	8,9
9	9.633.600	6,8
10	14.797.900	10,4
11	3.743.800	2,6
12	9.341.600	6,6
13	6.817.700	4,8
14	7.673.100	5,4
15	7.700.600	5,4
16	4.769.800	3,4
17	5.764.600	4,1
18	5.216.500	3,7
19	2.420.500	1,7
Total 2-19	141.919.800	100,0

2.2 Potential open-pit reserves of the study areas C - E down to a depth of 400' below the so-called dragline bench

(Annexe 3 and Page 53 of Vol. 1 of the Emkay-Scurry Study)

	Raw Coal in situ sht	Overburden Ration bank cyd/sht
C	21.000.000	6,22 : 1
D	15.000.000	4,33 : 1
E	54.000.000	9,39 : 1
Total C - E	90.000.000	7,81 : 1

2.3 Potential open-pit reserves of the study areas F - H up to 5,200 ft above NN (which corresponds roughly to the valley bottom of the Elk River Valley)

	Raw Coal in situ sht	Overburden Ratio bank cyd/sht
F	123.000.000	7,41 : 1
G	39.000.000	7,62 : 1
H	25.000.000	7,62 : 1
Total F - H	187.000.000	7,48 : 1

The areas F - H represent the steeply dipping, in part overturned western limb of the deposit. The reserves in these areas are shown by Emkay and Scurry as being capable of open-pit development. In our opinion, a large part of these reserves would be better suited for winning by hydraulic methods underground.

2.4 Total Raw Coal Reserves in Situ

With regard to the study areas A - H a total of raw coal reserves in situ of 446.000.000 sht can be reckoned with within the above-stated depth ranges.

3. The properties of the Raw Material

The analysis results with respect to the properties of the Elk River coals, listed in the Emkay-Scurry Study and in the two coking studies of the Department of Energy, Mines and Resources, Ottawa, from 1970 and 1971, were subjected to a critical appraisal by Montan Consulting GmbH, Essen, within the framework of a Consultance Agreement, and the suitability of these coals for use as coking-coal and/or as blend coal for use with Ruhr coal was examined.

The report of Montan Consulting is appended as Annexe 7 hereto. For this reason we are reproducing at this point only extracts relating to the most important results.

3.1 Sulphur Content

The sulphur content of all the seam coals is extremely favorable. In the individual seams it lies between 0.45 and 0.90 % (i.wf).It is therefore possible to reckon with average sulphur contents of between 0,65 and 0.75 % (i.wf).

3.2 The Phosphorus Content

The phosphorus contents are found to range between approx. 0.05 and 0.1 % P_2O_5, thus being on the upper limit or partly above the values usual for the Ruhr District. In all probability the use of preparation techniques will not substantially lower the phosphorus content.

3.3 Ash Contents and Washery Recovery

On the average, the individual seams are found to have primary ash contents of between 6 and 12 % ash (i.wf). The wash curves available (only present for seams 1 - 12) display characteristics that are similar for all seams. Accordingly, recovery will drop very sharply at a specific weight of 1.5 kg/dm^3.

Owing to the differences in the individual wash curves and the very considerable differences in the primary ash contents, a raw coal blending bed should in every case be included ahead of the preparation process. If homogeneous blending is carried out, it will be possible to decidedly improve the recovery values, as is shown by a wash curve, constructed for demonstration purposes of a homogeneous blend of equal parts from the seams of the sequence 1 - 12. In the event of an individual preparation of these seam coals, a coking-coal recovery of 74.5 % with a target ash content of 9.5 % could be attained, while with homogenous blending of the seam coals the recovery could be increased to around 78 %.

The above-mentioned recovery values refer, however, only to the seams 1 - 12, and only to a raw coal sample excluding country rock. At the present status of the investigations these values serve merely for demonstrating how recovery values can be improved by the inclusion of a raw coal blending facility. The recovery actually to be expected can only be established once the results of the preparation tests have come to hand for the remaining seams 13 - 19 in terms of the proportions by weight from the individual seams and the percentages of country rock in the raw coal production. To this end it will also be necessary to test the country rock for hardness, solubility in water and petrographical composition.

To be on the safe side, in the cost calculation given in a subsequent chapter of the present report a figure of only 60 % has been used for the average overall recovery.

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3.4 Coking Properties

Based on the maceral composition, volatile constituents, the dilatometer results and G-value readings, the Elk River coals can be assigned to 3 groups of seams:

Group 1: The seams 3, 4, 6, 7, 8, 9 and 10, which are low volatile, and according to the quality analysis results available have slight to deficient coking properties;

Group 2: the seams 2, 12 (+11), 13, 14 and 15, which are low to medium volatile and show good coking properties;

Group 3: the seams 16, 17, 18 and 19, which are high volatile and have an excess of coking properties.

A blend of coals from the seams of Groups 2 and 3 will, depending on the proportions used from the individual seams, yield a coking-coal with adequate to good cokability. The proportions that can be taken from the seams of Group 1 and added to those of Groups 2 and 3 will depend on the following premises, which are still to be checked in detail:

a) If the macerals classed as inert in the present analyses also react inertly during coking, as could be deduced from the dilatometer test results usual on the Ruhr and from the G-value reading, it would be possible to include an amount of up to 30 % in the composite blend. If larger amounts were added, a decisive deterioration in the coking properties would have to be reckoned with.

b) If, however, the conclusion to be drawn from the carbonization rank and the coking-test results confirms that a part of the macerals of the Cretaceous coals classed as inert do perhaps not react fully inertly during coking, there might be a possibility - even in

the event of including parts of up to 50 % of Group 1 in the blend - this would still yield a readily cokable coking-coal.
Since Group 1 makes up about 50 % and Groups 2 and 3 also make up 50 % of the reserves of utilizable raw coal, operations might well be centred - if case (b) above is confirmed - on the production of only 1 product, namely a blend coal deriving from all seams proportionate to their parts by weight in the deposit.

A final decision on this point can only be made once supplementary sampling and analyses have been carried out on a laboratory and semi-industrial scale. It is quite conceivable that due consideration can also be given to the differences in quality demands of the participating works by producing - which is technically feasible - 2 products of different quality (and the sale of them at different prices).

4. Supplementary Exploratory Programme

For the purpose of confirming the reserves tonnages in seams 16 -19 as well as for checking on the raw coal characteristics of the Elk River coals and their suitability for use as coking-coal and/or as blend coal for the West-European market, an exploratory programme has been drawn up by our Geological Office at Calgary (see Annexe 8), which make provision in detail for the following work to be carried out:

4.1 Core Drillings to Confirm the Reserves of Seams 16 - 19

For confirming the above-mentioned reserves, a further 17 core drillings (EB 1 - EB 17, see Annexe 8 + 9) with a total footage of 7,650 ft will be necessary. The total expenditure involved in carrying out these drillings including gamma-ray neutron and density logging, as well as laboratory analyses of the drill cores, will amount to $ 271,800.

4.2 Winning of Bulk Samples for Quality, Preparation and Coking-Tests in Germany and Italy

From every seam it will be necessary to obtain samples of 12 tons each for the above tests, 6 tons of each sample being destined for Germany and 6 tons for Italy.

The samples from the seams 2, 3, 6, 7, 8 and 9 can be taken from existing prospecting tunnels in the ravine of the Weary Creek.The tunnels require clearing up only to a minor extent and driven some metres beyond the coal face that in all probability is oxidized with a view to obtaining fresh coal samples.

Since the existing prospecting tunnel at Weary Creek Valley only the upper measure 4 and not the lower measure 4 A is to be encountered, however, even though seam 4 as a whole constitutes the major part (17 %) of the reserves, it was planned with a view to obtaining a representative sample to drive an exploratory cross-cut to cut through both measures at a suitable point (in baseline 260+00 at an elevation of 6,030 ft above NN).

As regards seam 10, a sample was taken by Emkay-Scurry from tunnel No. 10 in the seam outcrop area on the southwestern flank of Little Weary Ridge. The sampling-point at the bottom end of the adit reveals an overlying cover of only 38', so that there is the possibility that this sample was partly oxidized and the relatively poor coking result of seam 10 attributable to this. Since seam 10 (currently included in Group 1 with deficient coking properties) likewise makes up a substantial part (10.4 %) of the reserves,

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an attempt should by all means be made to obtain a sample from the seam that comes with certainty from outside the oxidation zone.

If the analysis of this fresh sample reveals that the coking properties of seam 10 are better than they are considered to be at the present time, the proportion of coals with good to excessive coking properties to those with deficient coking properties will shift from 50 : 50 to 60 : 40. As a result of this, the prospects of being able to blend all the seam coals, in accordance with their percentages by weight in the total reserves to form one single product with good coking properties, would increase considerably (see above).

With a view to obtaining a non-oxidized sample from seam 10 it was planned to drive a new exploratory tunnel about 50 ft under the existing tunnel, or alternatively, to sink an inclined shaft until a depth of cover strata of approx. 90 ft has been reached at the sampling point.

To obtain samples from seams 12 -19 new trenches should be excavated north of the trenches T-16 and T-17, parallel to the old backfilled trenches of Emkay-Scurry T-16 and T-17.

In the case of seam 11, the taking of a sample was dispensed with, since seam 11(a)is found only in the northern part of the deposit in Elk River Valley and joins approx. near baseline 260+00 with seam 12, and because (b) the coal-bearing measures in this area are covered by a

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layer of moraine material between 30 and 60 ft in thickness, which would make the exploratory work very expensive.

The total expenditure involved in the work listed under Point 2.4, incl. of sampling and packing of samples, would amount to

$ 151,400

The total time requirement for the fieldwork listed under 4.1 and 4.2 would amount to around 2 1/2 months.

4.3 Large-Scale Laboratory Tests

The 17 bulk samples of 6 tons each that are destined for Germany will be dispatched in an unwashed state. First and foremost it is planned to carry out preparation tests with these samples on a semi-industrial scale at Bergbau-Forschung, Essen, to be followed by 10 coking-tests (170 tests in all) with the cleaned coal (about 3 tons per sample). The aim will be to ascertain the cokability of the various seams, possible seam blends and seam blends combined with Ruhr coal.

The samples destined for the coking-oven tests at Italsider will be washed prior to dispatch by the Cyclon Engineering and Minerals Testing, Calgary.

The total outlays that will be incurred by the large-scale tests at Bergbau-Forschung (including the preparation carried out in Canada with the Italsider samples) will amount to

$ 209,100

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4.4 Cost of Transporting the Bulk Samples

The estimated cost of transporting the samples within Canada itself, the maritime freightage to Rotterdam or Genoa, as the case may be, the inland transportation in Europe for the samples destined for Germany, including the transshipment of the samples in Canada and Europe, will amount to a total of

$ 45,600

4.5 Geological Work and Mine Survey Mapping

For supervising the core drillings, the taking of bulk samples from exploratory tunnels and trenches, as well as for the preparation of various geological field maps, 2 geologists, 1 field assistant (or draughtsman) and 1 unskilled workman will be required. Following completion of the fieldwork, the time required by a geologist and a draughtsman for the evaluation of the exploration results must be estimated at a further 2 months or thereabouts.

For carrying out the survey mapping of the drill-holes, exploratory tunnels, prospecting trenches as well as access roads etc. a surveyor team of 3 men will be required for about 1 month.

The total cost incurred by the above tasks will amount to

$ 54,400

4.6 Improvement to Existing Access Roads or Construction of New Ones

For improvements to the in part already existing access routes to the drilling sites, exploratory tunnels, and prospecting trenches as well as for the construction of additional access roads (total length of access roads = around 6,100 m) a figure of $ 10,200 has been estimated.

4.7 Board and Lodgings for the Exploration Crews

For the duration of the fieldwork it will be necessary to board and lodge 33 men in the area under exploration, in a mobile camp. The costs involved in this will amount to a total of

$ 66,700

4.8 Reclamation Costs (estimated) $ 30,000

4.9 Other Costs

For transport facilities in the field, tools, operating media, office materials, travel expenses etc. the estimated figure is

$ 38,500

4.10 Imponderables (14 %) $ 122,300

The total costs of the proposed exploration programme will be approx. $ 1,000,000. The total time requirement including the coke oven tests will be about 1 year.

5. Planning of Winning Operations

5.1 General

For the purpose of selecting suitable mining methods it is possible to subdivide in principle the deposit areas scheduled for open-pit development A + B into 2 separate parts:

a) the reserves occurring above the so-called drag-line bench on Little Weary Ridge, and

b) the reserves that can be recovered below the drag-line bench in the deep open-pit.

Owing to the necessity of having to interblend the various seam coals with their different quality characteristics, mining must be conducted simultaneously on Little Weary Ridge and in the deep open-pit. In doing so, for the obvious reasons set out below, this must start in the north of the concession area (approx. near baseline 370+00), in other words, at the point that is the furthest away from the preparation plant.

Starting the deep open-pit operations in the south (near baseline 170+00) as a result of which short haulage distances and consequently higher outputs would be attainable during the first years of operation, is not possible for the following reasons:

a) The bulk of the reserves of the deep open-pit in this area (from seam 2 to about seams 14, 15) would be still overlain at the beginning of operations by the reserves of Little Weary Ridge occurring above the valley level, so that deep open-pit mining could initially only be carried on in a relatively narrow strip and would have to be kept open over a period of several years (up to the complete extraction of the reserves of Little Weary Ridge).

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b) There would be no possibility of siting an overburden tip outside the deep open-pit area, since in doing so the seams south of the areas A + B would have to be covered again prior to extraction and one would thereby deprive himself automatically of the possibility of continuing the mining in a southern direction after the areas A + B have been mined out.

c) Only approximately 25 % of the overburden from Little Weary Ridge could be dumped in the Weary Creek Valley, while the remaining 75 % would have to be dumped in the worked-out parts of the deep open-pit. Since the deep open-pit in this case would have to be kept open, however, over the entire length, until the reserves of Little Weary Ridge had been completely extracted, and it would not be possible to site an overburden dump in the southern part of the deposit area in Elk River Valley, for the reasons set under Point b, above, there would not be sufficient space available for dumping the overburden from Little Weary Ridge.

If the deep open-pit is started in the north, however, then (a) mining can comprise the entire width of the coal measures, and (b) the worked-out deep open-pit can be filled in from the north, as mining advances. In consequence of this there will be sufficient dumping space available for the overburden from the deep open-pit itself as well as for the overburden from Little Weary Ridge. Whereas mining advances in the deep open-pit from north to south, the reserves of Little Weary Ridge will be worked level-by-level in such a way that the reserves below valley level will be exposed over their entire width, ahead of the deep open-pit operations.

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In the area of Little Weary Ridge the usual minig method based on excavators and heavy trucks offers a suitable solution. For the deep open-pit there are two alternative solutions available for removing the overburden, and hence two stripping methods:

a) also with excavators and heavy trucks, and

b) with draglines.

Basically speaking, it may be said that in both these alternatives there will be a certain percentage of overburden (i.e. thin-bedded dirt layers) that will have to be loaded by front-end loaders. In the Little Weary Ridge area this portion will constitute around 4 % and in the deep open-pit around 16 % of the total overburden from the deposit.

Both these methods provide for the coal to be loaded into heavy trucks by front-end loaders.

To be on the safe side it was assumed during planning and costing that both the coal and the overburden will have to be drilled and blasted.

5.2 Description of the Alternative Deep Open-Pit Mining Methods

5.21 Mining Method Using Excavators and Heavy Trucks (Annexe 10)

If excavators and heavy trucks are the choice for overburden stripping in the deep open-pit, mining will proceed also in this area on individual working-levels (15 m height of level) from the top downwards (in all 9 deep open-pit levels). In the process, every working-level will be split up into working blocks of about 120 m (400 ft) in width, which will be worked by cutting at right angles to the strike. In doing so, the entire seam sequence in every block, starting with seam 19, will be extracted one after the other. The sequence of working the individual blocks is selected in such a way that the individual levels will be driven step-by-step from north to south. The cross-cut system (half and half) of winning the individual blocks on the various levels will be phased in such a way that the working block of which ever is the upper level will precede the block of next lower-lying level.

In this method of working there are always various seams standing ready for winning in the individual blocks on the various levels at the same time. The time required for extracting one working block will amount on the average to 6 months. Based on a washed coal production of say 3,000,000 sht/year, there will be in the deep open-pit at least 6 working blocks and three levels on Little Weary Ridge in production at the same time, thus providing adequate flexibility for ensuring every blend of seam that is desired. The lower the planned output is, the lower will be the number of winning equipment units required and hence the lower number of working points, as a result of which the flexibility in seam blending will in turn be reduced.

Until such times as the lowest level of the deep open-pit has been reached, the entire overburden from the deep open-pit will have to be raised to the surface and dumped to the north of baseline 370+00. After having extracted the first block on the lowest level, the overburden can be dumped, as mining proceeds, in the worked deep open-pit itself. Due to the loosening of the material it will not be possible, however, to avoid having to haul up the overburden through an average of two levels.

5.22 Working Method Using Draglines

(Annexe 11)

If draglines are used for overburden stripping in the deep open-pit, mining will also take place in individual blocks that have been constructed by cross-cuts from seam 19 to seam 2. Winning does not proceed over the whole length of the block being cut at right angles to the strike as when excavators and heavy trucks are used, here every seam is exposed right down to the lowest level and extracted before the next seam is tackled. Here the chosen width of the working block should not be less than 2,000 ft (around 610 m) so as to create, on the one hand, a working space that accords with the size of the mechanical units, and on the other hand will enable the haulage roadway gradient to be kept below 10 % (planned average 6 %), even when hauling up the coal from the deepest level.

The criteria for the choice of the suitable size for the draglines are the following requirements:

a) the maximum possible range

b) the digging depth should be at least 3 levels (135')

c) the dumping height should be around 30 - 40 % greater than the digging depth, allowance to be made for the loosening of the material and the increase in volume that is associated with this.

d) Maximum possible dragline bucket capacity.

In the present planning model the calculations were based on the smallest unit in the Bucyrus Erie production range (see Annexe 13 B) that would meet these requirements (BE 1350-W with 302' boom, a clearance angle of 38° and a 43 cyd bucket capacity).

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When using this type of dragline the entire overburden from the deep open-pit has to be handled 2.7 times on an average (Annexes 11 and 12). If a washed coal production of 3,000,000 tons is planned, the number of units required will be 7 (Annexe 14 B). Consequently, working can proceed simultaneously in 2 blocks, and at times in 3.

The total time requirement for extracting 1 block will be 6.86 years = 82 months on an average, which means that, depending on their thickness and the thickness of the overlying overburden, the individual seams will be worked for between 2 and 6 months. Accordingly, despite the simultaneous working of 2 or 3 blocks, there is as good as no flexibility in terms of quality control.

The selection of the above-mentioned dragline-type and the number of units do not yet represent an optimum solution. If we calculate the final version of the deep open-cast mine - possibly with bigger units and with equipment of different sizes - there could, however, be expected that the sequence of dumping rehandling and the number of units could be reduced resulting last but not least in a further reduction of costs.

6. Cost Calculation

6.1 General

For both the alternative mining methods, (a) with excavators and heavy trucks, and (b) with draglines, the production costs are based on an annual washed coal output of 3,000,000 sht and a preparation recovery of 60 %. With this size of operation and the above-mentioned recovery figure the life of the mine will be 28.4 years (the calculation factor employed was approx. 30 years, since there is a possibility of extending the mining operations towards the south).

The production size of 3,000,000 sht of washed coal per annum was selected because this represents an economic size, according to a rough calculation. Alternative calculations for different operation sizes with the aim of determining the optimal production size were not carried out in the present context owing to lack of time, but should in any case be included in a final Feasibility Study.

With regard to the deep open-pit, the costs to be incurred by machinery, wages and explosives have been calculated in detail, whereas the other costs listed are only estimated costs.

6.2 Bases Employed for the Open-Pit Cost Calculation

Based on the data and particulars of Bucyrus Erie and Wabco as well as on empirical values from local coal-mines and BMC empirical values, the anticipated performance rates and unit costs were initially ascertained for selecting the suitable machines and combinations of equipment for various types and sizes of equipment (Annexe 13 A - 13 C). All listed prices and costs are 1973 values. The wage costs were taken from the Trade Union Agreements of McIntyre Porcupine, valid for 1973, the highest rates being taken in each case as basis.

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The figure employed for wage incidentals was 50 %. This figure represents a certain safety factor in the cost calculation, since the actual wage incidentals of the West-Canadian coalmines at present lie between 20 and 30 %.

The entire cost calculation is based on 3-shift system, 7 days/week and 355 productive days per year. 10 days per year have been deducted as official public holidays.

6.3 Cost calculation for a production of 3,000,000 sht of washed coal per year with a recovery rate of 60 % and employing a purely heavy truck/excavator system

(Annexe 14 A)

If it is decided to use the usual excavator and heavy truck combination for the entire overburden stripping operation, the anticipated overall costs will be

	10.3 $/sht	or	11.6 $/lgt washed coal FOR
and	15.7 $/sht	or	17.6 $/lgt washed coal FOB

The FOB costs already include a royalty of 1 $/sht.

The total investment outlay will amount to $ 75,380,000 or 28.1 $/lgt of the annual output capacity.

6.4 Cost calculation for a production of 3 million sht of washed coal per annum, based on a 60 % recovery and the use of draglines in the deep open-pit

(Annexe 14 B)

If draglines are chosen for overburden stripping in the deep open-pit, the anticipated total costs will be:

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	10.2 $/sht	or	11.4 $/lgt washed coal FOR
and	15.5 $/sht	or	17.4 $/lgt washed coal FOB

The FOB costs already include a royalty of 1 $/sht.

The total investment costs will amount to $ 98,558,000 or 36.8 $/lgt of annual production.

7. Summary

In the Areas A + B of the Elk River coal deposit in British Columbia, which are directly capable of open-pit development, there are minable economic raw coal reserves of around 142,000,000 sht spread over 19 seams with varying quality properties.

With a view to confirming the reserves of seams 16 - 19 and for checking on the quality properties of the Elk River coals for the West-European market a supplementary exploration programme has been planned involving a total expenditure of 1,000,000 dollars.

If by means of this exploration programme the theoretical considerations of Montan Consulting GmbH are confirmed, then not only a part of the seams but the entire seam sequence can be used for producing a blend coal with good coking properties, based on the ratio of their percentages by weight.

Based on a production of 3,000,000 sht of washed coal and a preparation recovery of 60 %, the following costs are to be expected in the case of both mining methods - (a) with excavators and heavy trucks, or (b) with draglines.

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	a) Excavators		b) Draglines	
	$/ sht	$/ lgt	$/ sht	$/ lgt
Open-pit costs	6,12		5,95	
Mixing bed	0,36		0,36	
Preparation and loading	1,50		1,50	
Overheads	1,37		1,37	
Royalty	1,00		1,00	
FOR costs	10,3	11,6	10,2	11,4
Rail transport and porthandling		6,00		6,00
FOB costs - Roberts Bank	15,7	17,6	15,5	17,4

The difference in costs between the usual excavator and heavy truck and a mining method using draglines is found to be around 0.2 $/lgt in favor of employing draglines.

If one relativizes this small difference in costs, which might well lie within the error margin of the cost calculation, with the advantages of a mining method employing excavators and heavy trucks in terms of flexibility in the quality control as well as flexibility in the open-pit mining operations as a whole, priority should be given to the mining method based on excavators and heavy trucks.