K - Sage Creek 75(8)A

SAGE CREEK PROJECT

FEASIBILITY REPORT

APRIL 1975

VOLUME I

XEROX COPY INCLUDES

PART	II	SUMMARY
PART	III	GEOLOGY + RESERVES
PART	IV	MINING
PART	V	COAL PREPARATION

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PART II SUMMARY

The Sage Creek Coal deposit is located in the Flathead Valley of southeastern British Columbia approximately 52 miles south of the town of Fernie and 10 miles west of the Alberta border.

2.1 Exploration, Geology and Coal Reserves

The identified coal reserves occur in two areas now known as the North and South Hills.

Investigation of the property has included mapping, trenching, drilling and underground sampling. Drilling has been carried out on a grid pattern on 800 foot centres and consists of 78 holes totalling approximately 51,000 feet. Underground lateral work consists of 4,000 feet of drifts and cross-cuts from 12 adits. Bulk samples from all seams have been taken for analysis and testing of coal quality and to check for possible oxidation along fault contact zones.

The coal seams underlying the Sage Creek property were deposited during Mesozoic time, and occur in the Kootenay Formation. Locally the Kootenay Formation occupies an east dipping monocline structure with the enclosed strata striking north to northeast, and dipping easterly at an average of 25° to 30°. On South Hill numerous, steeply dipping, north to northwest trending normal faults cut the strata causing apparent horizontal lengthening.

Three economically significant seams are identified on the property: Seam 5 - the lowest in the stratigraphic section - has an average thickness of 35 feet; seams 4 upper and 4 lower have average thickness of 27 and 20 feet respectively, and seam 2 - the highest in the stratigraphic section - has an average thickness of 10 to 12 feet. The approximate rock-to-coal ratio of the Kootenay Formation in the property area is 8:1.

In situ geological coal reserves have been calculated on a basis consistent with the proposed mining methods. A tabulation is shown in the following table:

(millions of long tons)

	North Hill	South Hill	Total
Proven	68.6	36.5	105.1
Probable, possible	23.2	19.2	42.4
	91.8	55.7	147.5

Additional exploratory work in the proven area is not expected to change the calculated reserves by more than 20%. ± ?

Based on geological sections and the preliminary design of the pits and assumptions made for mining recovery, it is expected that 110 million long tons of coal will be available for delivery to the wash plant.

The coal has been established as a medium volatile bituminous coal and possesses characteristics favourable for coking. Approximate analysis of a clean coal blend of all three seams is shown below:

Ash	-	9.5%
Raw Moisture	-	1.6%
Volatile Matter	-	22.5%
Free Carbon	-	66%
Sulphur	-	0.4%
Free Swelling Index	-	6.5
BTU's/lb	-	14,000

2.2 Mining

The open pit mining method proposed is based on proven techniques using 20 yd. shovels, 170-ton trucks and 15 yd. front-end loaders to remove raw coal from the mine at a rate of 5.3 million long tons per year. The overall stripping ratios are 9.7 cubic yards of waste per long ton of raw coal available for delivery to the washing plant for the North Hill and 8.6 for the South Hill.

It has been assumed that mining will commence on the North Hill. The coal seams on the North Hill are relatively unfaulted, and have an overall uniformity of dip and strike. The reserves on the North Hill are greater than those of the South Hill, and the North Hill is not influenced to the same extent by thick clay deposits overlaying part of the coal seams. With good conditions in the first few years of mining, efficiencies and operating costs will be more predictable.

2.3 Coal Preparation

The Coal Preparation Plant will consist of dense medium cyclones, water only cyclones and Froth flotation. This type of wash plant is now being successfully used by present Western coal producers.

Coal samples obtained from adits, core drilling and reverse circulation rotary drilling, from both the North and South Hills have been tested in the Laboratory and Pilot Plant at Birtley Engineering in Calgary. The data from the sampling program has been augmented by the use of Density logs and Gamma Ray, Neutron logs in the drill holes. The tests have shown that the raw ash from the seams varies from 20% in Seam #2 to 38% in Seam #5.

The blended feed to the Coal Preparation Plant will have a raw ash content of 28%-30%.

The percentage yield from the Coal Preparation Plant will be 60% when processing coal from the North Hill and 53% when processing coal from the South Hill; the clean coal ash content will be 9.5±0.5%. On this basis the annual production of saleable metallurgical coal will be 3 million long tons.

A middling product will be produced to generate power on site at an ash content of 20% to 25% and a BTU/lb. value of 10,500.

2.4 Project Infrastructure

Administrative, service and maintenance facilities have been designed to provide full and efficient support of operations and have been located having regard to both North and South Hill mining.

It has been assumed that electrical power and heat for clean coal drying will be generated on-site by burning middling (waste) coal in a steam power plant. The use of an on-site power plant rather than B.C. Hydro will increase capital costs by an estimated $7.3 million but will result in annual power cost benefits of $2.25 million.

It is expected that a townsite will be built three miles southeast of the minesite. A 48 mile logging road from the proposed townsite to Fernie is now being upgraded for standard highway loadings. It has been assumed that for the 628 man work-force it will be necessary to provide 300 family housing units and permanent accommodation and facilities for 250 single employees. The balance of the workforce is expected to find their own accommodation.

upgraded road ? rail spur ? Fernie?

2.5 Environmental Protection

Development of Sage Creek must be in accordance with British Columbia Government environmental protection requirements. The major areas of concern are tailings disposal, reclamation and control of solid particle emissions. Consultants will be retained beginning in 1975 to provide "base line surveys" and reports for future reference in environmental control measurements.

2.6 Transportation of Coal

It has been assumed that all coal will be shipped to, and sold, f.o.b. Roberts Bank. Although various forms of transportation have been studied it appears at present that it will be necessary to construct a spur line through the Flathead Valley from the minesite via the McEvoy Pass to the existing C.P. Rail at McGillivray.

C.P. Rail have indicated that their rail rate will be a function of the capital cost of the spur line and of their operating costs based on a firm commitment to ship three million tons per year. Based on an order-of-magnitude capital cost estimate ($45 million in 1975 $) the expected transportation cost per ton is:

	(Approx. 1975 $)
Rail charge	$10.02
Terminal charges & insurance	1.11
	$11.13

2.7 Project Organization & Scheduling

During the summer of 1975 further drilling, bulk sampling and pilot plant work will be carried out, and consultants will be retained to assist with:

(a) Development of background data for subsequent environmental impact evaluations.

(b) Final assessment of geology and coal reserves.

(c) Assessment of overall site situations in the mining area from a geotechnical point of view.

(d) Development of data for spur line definitive design and cost estimating.

(e) Overall technical development of the project. This will be undertaken by the consulting firm responsible for the final feasibility study.

The final feasibility study is scheduled for completion by June, 1976 with marketing and financing arrangements completed by the end of 1976. Assuming that the project is released for construction January, 1977 start-up is expected July, 1979. Production is built-up over a 30 month period so that the first full calendar year at the design capacity of 3.0 million long tons is 1982.

For purposes of cost estimating it has been assumed that a general contractor will be appointed with responsibility for major civil construction and co-ordination of all site construction. Subcontracts will be let for specialized packages. During the entire programme, administrative policy, financial, and technical control will be provided by Rio Algom.

2.8 Capital Costs

The total on-going capital requirements in escalated Dollars for the project are estimated at $205.4 million, summarized as follows:

Plant site services & facilities	$5.8	
Miscellaneous buildings	12.3	
Power plant & generator	21.6	
Crushing, blending & reclaiming	9.0	
Wash plant	23.6	
Tailings handling	3.2	
Townsite & off-site services	17.1	
Mine equipment	38.8	
Minesite services & utilities	1.4	132.8
Open pit development	11.9	
Project overhead & administration	25.9	
Start-up	0.9	38.7
Exploration & development		2.6
Working capital		16.3
Interest & financing charges		15.0
TOTAL CAPITAL REQUIREMENTS (Including escalation)		$205.4

Interest and financing charges have been based on the assumption that a $145 million loan is utilized.

The above estimate does not include costs to date nor the cost of the spur line and/or railway rolling stock. The townsite and off-site services are net of employee mortgages which are assumed to provide an additional $9.0 million for townsite construction.

2.9 Operating Schedule and Costs

Production rates are based of the following criteria:-

The mine will operate for 344 Days per year and the coal preparation plant for 320 days per year. The extra days in the mine will be scheduled for waste removal and to replenish the raw coal stockpile. The mine will be shut down for ten statutory holidays and twelve days have been allowed for severe winter conditions.

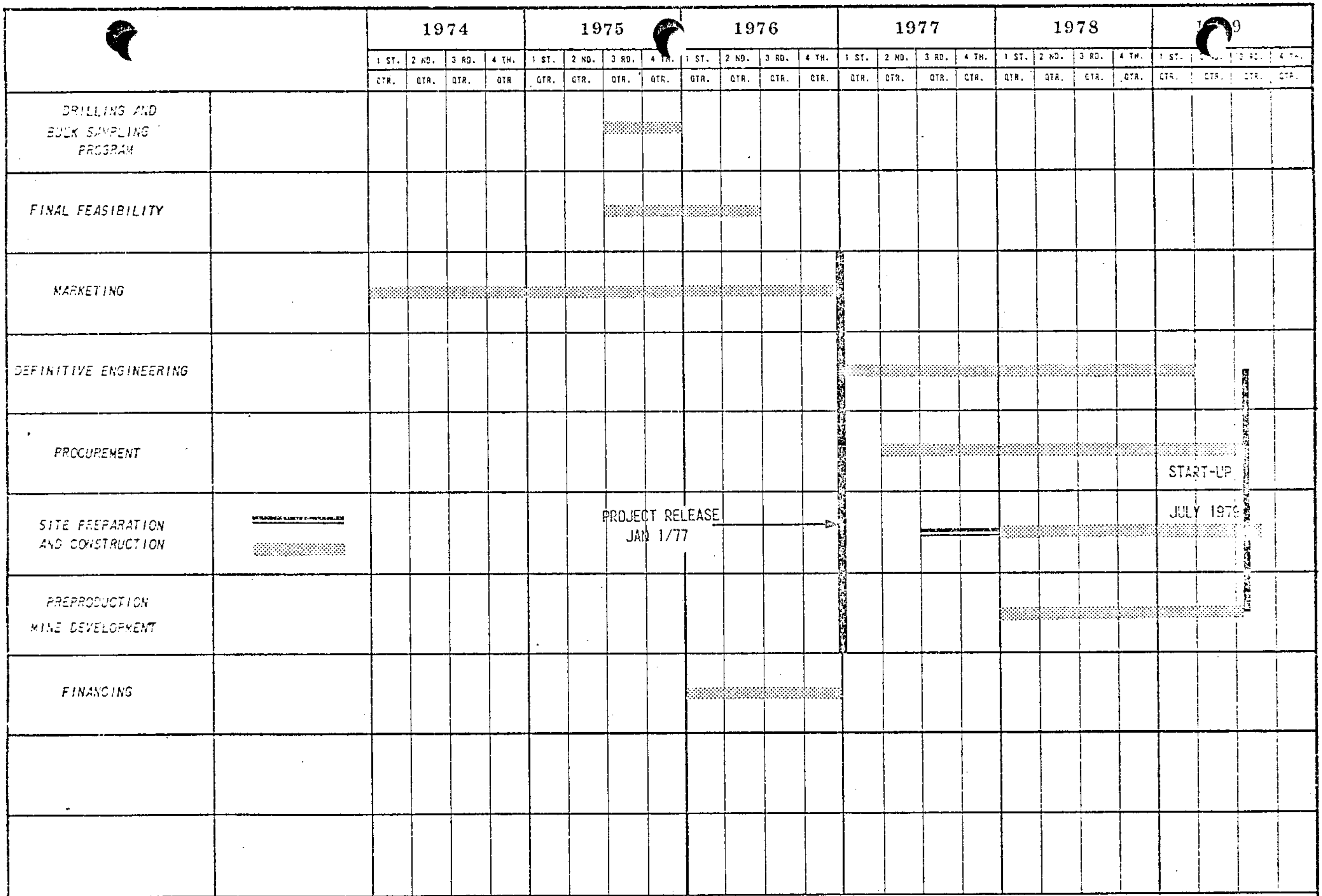
The coal preparation plant is scheduled to operate on a three shift basis as follows:

Operating days	-	320
Statutory Holidays	-	10
Planned Maintenance (two shifts/week)	-	30
Unplanned Maintenance	-	5
Total		365

Operating costs (in 1975 $) are summarized in the following table:

	$ Millions	$ per long ton
Mining	10.4	3.46
Coal preparation	1.2	0.41
Plant general	8.5	2.83
Management & administration*	7.6	2.53
Property & capital tax	2.1	0.71
	$29.8	$9.94

*Includes total fringe benefits and management fee.



Rio Algom

SAGE CREEK COAL

PROPOSED DEVELOPMENT SCHEDULE

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TABLE IV

MANPOWER SUMMARY

			Total
I	MANAGEMENT	6	6
	Senior Operating Supervision	4	
	Accounting	4	
	Warehouse	6	
	Personnel and Safety	8	
	Computer Department	4	
	Secretaries	4	
	Security	4	34
	Hourly - Warehouse and Townsite	15	15
II	MINING		
	Staff	43	
	Hourly	223	266
III	COAL PREPARATION PLANT		
	Staff	11	
	Hourly	40	51
IV	PLANT SERVICES AND MAINTENANCE		
	Staff	34	
	Hourly	222	256
	TOTAL WORK FORCE		628

TABLE V

OPERATING COST SUMMARY

	ANNUAL COSTS										
	Operating Super.	Labour Hourly	Maint. Super.	Labour Hourly	Operating Supplies	Maint. Supplies	Fringe Benefits	Total	Mat. Cost /Ton	Lab. Cost /Ton	Total Cost /Ton
Management & Administration	2,190,000	190,320			2,837,650		2,385,243	7,603,213	.945	1.588	2.533
Mining	606,000	3,495,720			6,289,947			10,391,667	2.096	1.367	3.463
Coal Preparation	156,000	568,276			500,000			1,224,276	.166	.241	.408
Plant General			553,000	3,286,659		4,635,000		8,474,659	1.545	1.279	2.825
TOTALS	2,952,000	4,254,316	553,000	3,286,659	9,627,597	4,635,000	2,385,243	27,693,815	4.755	4.475	9.23

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GEOLOGY AND RESERVES.

Coal occurrences in the Flathead Valley were first reported in 1910 and reference is made to these occurrences in the Geological Survey of Canada Memoir 87, published in 1916.

Serious attempts to develop significant reserves however, were not made until the late sixities and early seventies when Pan Ocean Oil Limited acquired coal licences covering the Sage Creek property and entered into an exploration and development agreement with Rio Tinto Canadian Exploration Limited.

3.0 Geology (map Reference DWG G4449) App. IV

3.01 General

The coal measures in the area occur in the Kootenay Formation which was laid down during late Jurassic and/or early Cretaceous time. The Kootenay Formation in this area lies in the upper plate of the Lewis Thrust and was preserved by subsequent down faulting between the Clark Range to the east and the MacDonald Range to the west. Locally, the formation occurs on the east flank of a northwest trending anticline. The strata strike north to northeast and dip easterly at approximately 30°.

The Formation is truncated against the Harvey Fault to the northeast and the Flathead Fault to the southeast. These are steeply dipping normal faults with inferred displacements of approximately 2,500 feet and 20,000 feet respectively. To the southwest the formation passes underneath the MacDonald Thrust sheet.

To the west and northwest the Kootenay Formation rests conformably on the marine shales and siltstones of the Fernie Group.

3.0.2 Geology of the Deposit Area

Coal Seams

Three coal seams of economic significance occur on the property; seam #5, #4 (upper and lower) and seam #2.

Seam #5

This is the lowest coal seam in the stratigraphic section and has an average thickness of 35 feet.

Shale partings and carbonaceous shale bands are more abundant in this seam than in seams #2 and #4.

The seam is split into two benches by a carbonaceous shale unit that varies in thickness from 3 to 8 feet. This shale parting increases in thickness to the southwest on South Hill.

Seam #4

This seam occurs as two distinct benches and, on North Hill, these benches form separate seams: Seam 4 Upper and Seam 4 Lower . The average thickness of Seam 4 Upper is 27 feet, and of Seam 4 Lower, 20 feet. The parting between the benches varies in thickness from a minimum of 3 feet at the south end of South Hill to a known maximum of 40 feet at the north end of North Hill. A further split in the lower bench develops towards the south and southwest of South Hill. North of grid line 17,856, 660N at the north end of North Hill, Seam 4 Lower abruptly thins and shales out.

Seam #2

Seam #2 is the highest coal horizon of economic significances in the stratigraphic section and has an average thickness of 10 to 12 feet, varying from 5 feet at the southwest of South Hill to 15 feet at the north end of North Hill.

3.0.3 Stratigraphy

The Kootenay Formation consists of non-marine strata of fine grained to conglomeratic sandstones, siltstones, shales and coal seams, deposited under varying and recurring conditions (bog to turbulent) in a fluvial and/or deltaic environment. The formation varies from 650 to 800 feet thick.

The Kootenay Formation is defined by a characteristic sandstone unit at its base, usually from 40 to 80 feet thick, and by the Cadomin, the basal conglomerate unit of the Blairmore Group, lying disconformably above the Kootenay Formation. The character of the intervening beds, being lenticular in shape and interfingering laterally, does not allow for the establishment of marker horizons but gross patterns of deposition are recognizable. The coal seams are fairly characteristic throughout the property and offer the best means of correlation. Holes 74-49 on North Hill and 74-07 on South Hill have been selected as intersecting complete and typical sections of Kootenay stratigraphy and have been used as type sections in assisting correlations.

Coal seam #5 rests on the basal sanstone unit. Approximately 180 to 220 feet of shales and fine clastics separate seam #5 from seam #4. Medium grained sandstone lenses may be developed locally in this interval: Approximately 120 to 150 feet above seam #5, a zone of carbonaceous shale, shaly coal and thin coal bands is developed. This zone is nearly always present and in restricted local areas may attain some economic significance.

The greatest stratigraphic variation is seen between seam #4 and Seam #2. This interval attains a maximum of 240 feet on the north slope of South Hill to a minimum of 40 feet on the northeast slope of North Hill. On the South Hill, the interval is characterized by the development of two distinctive, massive, medium to coarse grained sandstone units separated by shale with siltstone bands and a few carbonaceous to coaly shale bands. These sandstone units are recognized in drill holes on the southeast slope of North Hill but rapidly shale out to the northwest. The lower sandstone unit has a greater lateral extent but thins and shales out north of line 17,855,060N. The disappearance of the upper sandstone and the thinning and shaling out of the lower sandstone in a northward direction, together with a substantial thinning of the interval between seam #4 and seam #2, suggests a fairly widespread erosion surface prior to deposition of seam #2.

Seam #3 encountered in drill hole SCC 2 could be a remnant of an eroded seam or a local development on the erosion surface. No economic significance is attached to this occurrence.

The strata between Seam #2 and the basal Blairmore conglomerate is indicative of cyclical deposition. Two massively bedded medium to coarse grained sandstone units of similar thickness and character are bound above and below by carbonaceous to coaly zones. The lower of these zones contains coal seam #2 while the zone separating the two sandstone units contains coal seam #1 which is usually less than 2 feet thick, and of little economic significance.

The only facies of the Blairmore Group recognized on the property is the basal conglomerate unit or Cadomin Formation which is restricted to the east slope areas of North and South Hills. Later Blairmore deposition appears to have been eroded.

Underlying a flat-lying area on the west bank of Howell Creek are recent deposits of till and gravel, which rest unconformably on the Kootenay and Blairmore Formations. These deposits may attain a thickness of several hundred feet. 250 feet of gravel was intersected in drill hole 74-32.

To the south and southeast on South Hill the Kootenay and Blairmore Formations are truncated against Tertiary deposits of clays, marls and loosely consolidated gravels of the Kishenehn Formation. The erosional unconformity plunges to the southeast at 30° to 40° from a surface trace trending NE-SW approximating a line passing through holes 74-17 and 74-11. In drill hole 74-15, 800 feet of Kishenehn deposits were intersected before entering lower Kootenay strata and hole 74-20 was abandoned at 820 feet in Kishenehn clays and gravels.

3.0.4 Structure

Normal faulting subparallel to the Harvey and Flathead Faults is in evidence across the property where the normal succession of strata is interrupted, causing apparent down dip repetition.

The magnitude of the throw on the faults is generally less than 200 feet but may reach 800 feet along at least one fault passing just to the west of Stelco drill hole No. 4.

The beds are generally down dropped to the west but at least three faults, two on South Hill and one on North Hill, have been observed or inferred where downdrop is to the east.

Minor thrusting and adjustment faulting has been observed in outcrop with displacements ranging from a few inches to a few feet. These disturbances are probably local and cannot be traced for any distance along strike. They may be associated with glacial and slump structures.

North Hill is considered structurally simple when compared to South Hill, with few recognized faults.

One fault on North Hill, recognized through drill hole intersection, trends approximately N-S and passes through drill holes 74-50 and 74-42. The apparent throw against the fault is approximately 150 feet at drill hole 74-50 and diminishes northward to an inferred origin in the vicinity of hole 74-33.

A minor fault with a displacement of 10 feet to 15 feet cuts seam #2 in a road cut between holes 74-33 and 74-30 and could be an extension or an off-shoot of the above fault.

An east-dipping normal fault, trending approximately NW-SE, has been interpreted from the drill hole results to cut holes 74-24, SCC 6 and 74-32. The apparent displacement against this fault is approximately 250 feet in hole 74-32 and diminishes northward to 150 feet in hole 74-24. Subparallel to this fault and to the east are two intersecting, west-dipping, normal faults passing through and just west of hole 74-25. These faults appear to diverge to the northwest. The total apparent stratigraphic separation against these faults is approximately 250 feet at hole 74-25.

Other faulting in this area may become apparent as the Harvey Fault to the northeast is approached. The surface expression of the Harvey Fault approximates Howell Creek and the fault has a displacement of approximately 2,500 feet.

The South Hill, by comparison, is structurally complicated. Ten faults, trending north to northwest, are recognized or inferred across the east slope between grid lines 581,000E and 585,000E. The displacement on these faults varies from 50 feet to 250 feet, with the most westerly fault in the area having a throw of approximately 800 feet.

Five of the faults have been identified at the surface or have been intersected in drill holes. Attitudes and throw along the faults is inferred from structure contours and photogrammetry. The remaining faults are inferred to explain anomalous zones between drill holes.

It is reasonable to assume that there is more faulting of the formation overlying the South Hill than is shown by current mapping. Generally, movement along the faults is upward on the east side relative to the west. Thus the coal seams are brought progressively closer to the present surface as we traverse from west to east across the area of the proposed pit.

The opinion of consultants employed to design the pit is that operating costs should not be affected materially in the event than more faulting of the same order is encountered.

3.1 Exploration

Exploration of the property by Rio Tinto Canadian Exploration Limited commenced in the Fall of 1970 and has continued to the present time. Investigations have included mapping, trenching, drilling and bulk sampling.

3.1.1 Drilling Programme

Diamond drilling of coal seams has not proven satisfactory and the bulk of drilling on the property has been accomplished with rotary drilling equipment.

In order to develop the quality and quantity of coal reserves it was decided that holes should be drilled on an 800-foot by 800-foot grid pattern, on both the North Hill and the South Hill. In selected areas, special equipment was used to core the coal seams.

Representative rock chip samples were collected at 5-foot intervals, logged and retained in vials for future reference.

Cuttings from all coal seams intersected were collected in two-foot increments and sent to the laboratory for analysis. Following this test work, the samples were forwarded to Birtley Engineering in Calgary where further testing was carried out.

Down hole geophysical probing was carried out to assist in geological correlation and to establish ash content of the coal seams.

All holes were probed, with gamma ray/neutron and where hole condition allowed, with sidewall density and resistivity.

The gamma ray/neutron results were used in conjunction with the drill cuttings to plot the stratigraphy.

The principal use of the sidewall density results was to determine ash content of the coal seam intercepts. This method of ash determination was monitored by probing holes close to bulk sample locations and holes in which the coal seams had been cored and analysed. The coal seams in three of the rotary holes were cored for this purpose.

3.1.2 Bulk Sampling

Bulk samples of coal were obtained by driving a number of adits into the North Hill and South Hill. Sample locations have been so sited as to assure that all samples were taken far enough below surface in order to eliminate the effects of oxidation. The purpose of this bulk sampling program was to obtain representative samples from each seam in a sufficient number of locations in order to give the maximum information as to quality of the coal across the deposit.

To this end, 4,000 feet of drifting and cross cutting was carried out during the period 1972 to 1974 and approximately 220 tons of coal collected and sent for analysis and testing.

A second purpose of the adit program was to investigate the degree of oxidation that might occur in the coal parallel to fault planes. Two adits were driven to check this possible phenomenon and results to date show no abnormal oxidation associated with the faulting.

3.2.1 Determination of Raw Ash

The raw ash content of the coal seams was determined from down-hole sidewall density probe results and from tests carried out on drill core and bulk samples.

The bulk samples and core samples, which were obtained from all seams, included all shale partings and were representative of the coal in place at the sample points. The raw ash content and washability of the samples was determined by tests carried out by Birtley Engineering Limited.

The results of the down-hole sidewall density probing were used to determine the raw ash content in areas where bulk samples and core samples were not available. This was done by plotting the ash content, as obtained by Birtley Engineering, against the density or in-situ specific gravity recorded by the density probe in holes that had produced the core samples and in holes close to the bulk sample points.

Using the resulting graph, raw ash in areas remote from the sample points could be read directly from the specific gravity as determined by the density probe.

Raw ash was thus determined for all seams in all holes except where excessive caving of the walls prevented the use of the density tool.

The results showed that the raw ash content varies from 20% for seam #2 to 38% for seam #5.

3.1.3 Determination of Yield

The bulk samples and core samples extracted from the South and North Hills were tested to determine yield and produce washability curves. All samples were sink-float tested between specific gravities of 1.3 to 1.9.

The bulk samples were washed in a pilot plant at Birtley Engineering using dense medium cyclones, water only cyclones and Froth Flotation. These tests have shown that an overall yeild of 60% in the North Hill and 53% in the South Hill can be expected. (See Appendix I Birtley Engineering summarized results).

3.2 Coal Reserves

In situ geological raw coal reserves for North Hill and South Hill have been calculated and tabulated separately. For this exercise the apparent seam intervals from the bore hole logs were converted to true thickness assuming that the strata have a uniform dip of 30°. Only coal horizons having a true thickness of 5 feet or more have been included in the reserves. Shale parting of less than 5 feet true thickness have also been included in the reserves; shale exceeding 5 feet has been omitted from the calculations. The following long-ton (2,240 lbs.) factors were used in the calculations:

Seam #2:	25.4 Cubic Feet/Ton
Seam #4:	24.0 Cubic Feet/Ton
Seam #5:	22.0 Cubic Feet/Ton
Shale:	18.2 Cubic Feet/Ton

These factors were derived considering the specific gravities of the materials involved. Tests by Birtley Engineering Ltd., have indicated that the average specific gravity of Seam #2 is 1.41, Seam #4 (Upper and Lower) 1.50, and Seam #5 1.63; the specific gravity of the shale partings was considered to be 1.97.

For North Hill the in-situ reserves were calculated for an area between lines 17,850,860N and 17,858,060N. Two down-dip cut-offs have been used: 3,900 feet above sea level and 3,400 feet above sea level.

On South Hill the reserves were calculated for the area between line 17,844,460 and 17,848,860 and down-dip cut-offs of 3,900 feet above sea level and 3,600 feet above sea level.

Three categories have been used to describe the reserves: proven, probable and possible.

Proven reserves are considered to lie between the outcrop trace of seam #5 and a point 200 feet east of the most easterly drill hole in each section. On both hills the northern and southern cut-off boundary is 200 feet north or south from the last respective section. Additional work in the proven area is not expected to alter the calculated reserves by more than 20%.

Probable reserves are those which are not supported by direct bore hole evidence, but are interpreted from geological evidence. On North Hill, probable reserves are considered to lie in a 400 foot wide strip between lines 17,857,660N and 17,858,060N and in an 800 foot wide zone centered on line 17,854,260N between holes 74-37 and 74-39 (Seam #4 and #5 only); the reason here is that hole 74-39 was terminated short of the seams #4 and #5, but seam #2 did not indicate any abnormal geological behaviour of the coal horizons in this area. On South Hill, probable reserves lie in two 200 foot wide zones between lines 17,845,260N and 17,845,060N and 17,848,460N and 17,848,660N.

Possible reserves are considered to lie between the point 200 feet east of the most easterly drill hole on a given section and the down-dip extension of the seam to the particular cut-off elevation.

Total in-situ reserves for all categories based on the above parameters are calculated at:

North Hill

To 3,900 feet ASL

Proven	57,897,861	
Probable	3,796,088	
Possible	2,763,679	
Total	64,457,628	long tons

To 3,400 feet ASL

Proven	68,633,264	
Probable	6,173,609	
Possible	17,030,181	
Total	91,837,054	long tons

South Hill

To 3,900 feet ASL

Proven	33,293,427	
Probable	7,159,052	
Possible	8,391,855	
Total	48,844,334	long tons

To 3,600 feet ASL

Proven	36,412,181	
Probable	7,293,188	
Possible	12,050,428	
Total	55,755,797	long tons

A summary of the reserve calculations by sections is outlined in the following tables.

S O U T H   H I L L

GEOLOGICAL RESERVES FROM 3,900' TO 3,600' ABOVE SEA LEVEL

	(1) SECTION	(1) SEAM 2	(2) No.2 SHALE *	(3) SEAM 2 + SHALE (1)+(2)	(4) SEAM 4U	(5) SEAM 4L	(6) SEAMS 4U+4L (4)+(5)	(7) No. 4 SHALE	(8) 4U+4L+SHALE (6)+(7)	(9) SEAM 5U	(10) SEAM 5L	(11) SEAMS 5U+5L (9)+(10)	(12) No.5 SHALE	(13) 5U+5L+SHALE (11)+(12)	(14) TOTAL ALL SEAMS (1)+(6)+(11)	(15) TOTAL SHALE (2)+(7)+(12)	(16) TOTAL COAL-SHALE (14)+(15)
PROVEN	17,846,250	-	-	-	357,533	159,135	516,668	38,857	555,525	321,237	404,909	726,146	75,604	801,750	1,242,814	114,461	1,357,275
	17,847,060	58,961	-	-	312,000	287,267	599,267	115,296	714,563	68,073	107,782	175,855	-	175,855	834,083	115,296	949,379
	17,847,860	-	-	-	-	-	-	-	-	168,300	643,800	812,100	-	812,100	812,100	-	812,100
	TOTAL	58,961	-	-	669,533	446,402	1,115,935	154,153	1,270,088	557,610	1,156,491	1,714,101	75,604	1,789,705	2,888,997	229,757	3,118,754
PROBABLE	17,848,260	-	-	-	-	-	-	-	-	44,318	89,818	134,136	-	134,136	134,136	-	134,136
	TOTAL	-	-	-	-	-	-	-	-	44,318	89,818	134,136	-	134,136	134,136	-	134,136
POSSIBLE	17,846,250	94,992	-	-	346,000	127,500	473,500	24,000	497,500	-	-	-	-	-	568,492	24,000	592,492
	17,847,060	140,850	-	-	-	-	-	-	-	-	-	-	-	-	140,850	-	140,850
	17,847,860	225,551	-	-	552,000	234,000	786,000	-	786,000	33,000	55,500	88,500	-	88,500	985,051	-	985,051
	17,848,260	124,892	-	-	383,500	329,417	712,917	-	712,917	132,954	189,091	322,045	-	322,045	1,159,844	-	1,159,844
	17,848,660	72,708	-	-	206,250	152,500	358,750	47,760	406,510	-	-	301,118	-	301,118	732,576	47,760	780,336
	TOTAL	543,983	-	-	1,487,750	843,417	2,331,167	71,760	2,402,927	165,954	244,591	711,663	-	711,663	3,586,813	71,760	3,658,573
	PROVEN	58,961	-	-	669,533	446,402	1,115,935	154,153	1,270,088	557,610	1,156,491	1,714,101	75,604	1,789,705	2,888,997	229,757	3,118,754
	PROBABLE	-	-	-	-	-	-	-	-	44,318	89,818	134,136	-	134,136	134,136	-	134,136
	POSSIBLE	543,983	-	-	1,487,750	843,417	2,331,167	71,760	2,402,927	165,954	244,591	711,663	-	711,663	3,586,813	71,760	3,658,573
	TOTAL 3900'-3600'	602,944	-	-	2,157,283	1,289,819	3,447,102	225,913	3,673,015	767,882	1,490,900	2,559,900	75,604	2,635,504	6,609,946	301,517	6,911,463

TOTAL GEOLOGICAL RESERVES

A. NORTH HILL

B. SOUTH HILL

[illegible]	(1) SEAM 2	(2) NO. 2 SHALE	(3) SEAM 2 + SHALE (1) + (2)	(4) SEAM 4U	(5) SEAM 4L	(6) SEAMS 4U + 4L (4) + (5)	(7) NO. 4 SHALE	(8) 4U + 4L + SHALE (6) + (7)	(9) SEAM 5U	(10) SEAM 5L	(11) SEAMS 5U + 5L (9) + (10)	(12) NO. 5 SHALE	(13) 5U + 5L + SHALE (11) + (12)	(14) TOTAL ALL SEAMS (1) + (6) + (11)	(15) TOTAL SHALE (2) + (7) + (12)	(16) TOTAL COAL + SHALE (14) + (15)
A. NORTH HILL																
PROVEN TO 3,400'	9,209,879	684,903	9,894,782	17,054,760	10,349,186	27,403,946	210,285	27,614,231	15,150,737	14,568,968	29,719,705	1,404,547	32,124,252	66,333,529	2,295,735	68,633,264
PROBABLE TO 3,400'	258,945	-	258,945	2,477,383	1,126,400	3,603,783	-	3,603,783	628,182	1,496,545	2,124,727	186,154	2,310,881	5,987,455	186,154	6,173,609
POSSIBLE TO 3,400'	3,594,126	-	3,594,126	7,482,976	2,523,984	10,006,960	68,571	10,075,531	1,371,518	1,787,413	3,158,931	201,593	3,360,524	16,760,017	270,164	17,030,181
TOTAL TO 3,400'	13,062,950	684,903	13,747,853	27,015,119	13,999,570	41,014,689	278,856	41,293,545	17,150,437	17,852,926	35,003,363	1,792,294	37,795,657	89,081,001	2,756,053	91,837,054
B. SOUTH HILL																
PROVEN TO 3,600'	2,822,361	-	-	12,169,808	6,479,286	18,649,094	1,426,581	20,075,675	5,797,090	7,468,308	13,265,398	248,747	15,514,145	34,736,853	1,675,328	36,412,181
PROBABLE TO 3,600'	117,559	-	-	2,044,709	1,630,900	3,675,609	61,769	3,737,378	1,308,900	2,091,549	3,400,449	37,802	3,438,251	7,193,617	99,571	7,293,188
POSSIBLE TO 3,600'	1,020,503	-	-	3,490,067	1,869,992	5,360,079	254,754	5,594,833	870,607	1,424,046	5,261,993	153,099	5,415,092	11,642,575	407,853	12,050,428
TOTAL TO 3,600'	3,960,423	-	-	17,704,604	9,980,178	27,684,782	1,743,104	29,427,886	7,976,597	10,983,903	21,927,840	439,648	22,367,488	53,573,045	2,182,752	55,755,797

S O U T H   H I L L

GEOLOGICAL RESERVES FROM OUTCROP TO 3,900' ABOVE SEA LEVEL

SECTION	(1) SEAM 2	(2) No. 2 SHALE	(3) SEAM 2+SHALE (1)+(2)	(4) SEAM 4U	(5) SEAM 4L	(6) SEAMS 4U+4L (4)+(5)	(7) No.4 SHALE	(8) 4U+4L+SHALE (6)+(7)	(9) SEAM 5U	(10) SEAM 5L	(11) SEAMS 5U+5L (9)+(10)	(12) No.5 SHALE	(13) 5U+5L+SHALE (11)+(12)	(14) TOTAL ALL SEAMS (1)+(6)+(11)	(15) TOTAL SHALE (2)+(7)+(12)	(15) TOTAL COAL+SHALE (14)+(15)
17,845,460	192,992	-	-	1,610,125	632,541	2,242,666	185,308	2,427,974	1,134,654	925,662	2,060,316	113,407	2,173,723	4,495,974	298,715	4,794,689
17,846,260	881,071	-	-	2,846,633	2,164,101	5,010,734	421,890	5,432,624	1,568,073	2,047,564	3,615,637	59,736	3,675,373	9,507,442	481,626	9,989,068
17,847,060	943,307	-	-	4,274,467	1,638,867	5,913,334	582,153	6,495,487	1,412,872	1,638,819	3,051,691	-	3,051,691	9,908,332	582,153	10,490,485
17,847,860	746,030	-	-	2,769,050	1,597,375	4,366,425	83,077	4,449,502	1,123,881	1,699,772	2,823,653	-	2,823,653	7,936,108	83,077	8,019,185
TOTAL	2,763,400	-	-	11,500,275	6,032,884	17,533,159	1,272,428	18,805,587	5,239,480	6,311,817	11,551,297	173,143	11,724,440	31,847,856	1,445,571	33,293,427
17,845,460	64,331	-	-	536,709	207,150	743,059	61,769	805,628	378,218	309,367	687,585	37,802	725,387	1,495,775	99,571	1,595,346
17,848,260	53,228	-	-	1,508,000	1,423,750	2,931,750	-	2,931,750	886,364	1,692,364	2,578,728	-	2,578,728	5,563,706	-	5,563,706
TOTAL	117,559	-	-	2,044,709	1,630,900	3,675,609	61,769	3,737,378	1,264,582	2,001,731	3,266,313	37,802	3,304,115	7,059,481	99,571	7,159,052
17,844,660	-	-	-	348,400	79,300	427,700	-	427,700	-	-	1,488,900	-	1,488,900	1,916,600	-	1,916,600
17,845,460	-	-	-	415,500	81,467	496,967	70,769	567,736	110,909	127,964	238,873	-	238,873	735,840	70,769	806,609
17,846,260	26,205	-	-	156,000	95,333	251,333	29,231	280,564	151,272	369,600	520,872	153,099	673,971	798,410	182,330	980,740
17,847,060	-	-	-	-	-	-	-	-	272,290	386,909	659,199	-	659,199	659,199	-	659,199
17,847,860	110,551	-	-	364,000	234,975	598,975	-	598,975	170,182	294,982	465,164	-	465,164	1,174,690	-	1,174,690
17,848,260	221,339	-	-	357,500	268,000	625,500	-	625,500	-	-	-	-	-	856,839	-	856,839
[illegible],660	108,425	-	-	360,937	267,500	628,437	82,994	711,431	-	-	1,177,322	-	1,177,322	1,914,184	82,994	1,997,178
TOTAL	476,520	-	-	2,002,337	1,026,575	3,028,912	182,994	3,211,906	704,653	1,179,455	4,550,330	153,099	4,703,429	8,055,762	336,093	8,391,855
PROVEN	2,763,400	-	-	11,500,275	6,032,884	17,533,159	1,272,428	18,805,587	5,239,480	6,311,817	11,551,297	173,143	11,724,440	31,847,856	1,445,571	33,293,427
PROBABLE	117,559	-	-	2,044,709	1,630,900	3,675,609	61,769	3,737,378	1,264,582	2,001,731	3,266,313	37,802	3,304,115	7,059,481	99,571	7,159,052
POSSIBLE	476,520	-	-	2,002,337	1,026,575	3,028,912	182,994	3,211,906	704,653	1,179,455	4,550,330	153,099	4,703,429	8,055,762	336,093	8,391,855
TOTAL 3,900'	3,357,479	-	-	15,547,321	8,690,359	24,237,680	1,517,191	25,754,871	7,208,715	9,493,003	19,367,940	364,044	19,731,984	46,963,099	1,881,235	48,844,334

N O R T H   H I L L

GEOLOGICAL RESERVES FROM 3,900' TO 3,400' ABOVE SEA LEVEL

SECTION	(1) SEAM 2	(2) NO. 2 SHALE	(3) SEAM 2 + SHALE (1) + (2)	(4) SEAM 4U	(5) SEAM 4L	(6) SEAMS 4U + 4L (4) + (5)	(7) NO. 4 SHALE	(8) 4U + 4L + SHALE (6) + (7)	(9) SEAM 5U	(10) SEAM 5L	(11) SEAM 5U + 5L (9) + (10)	(12) NO. 5 SHALE	(13) 5U + 5L + SHALE (11) + (12)	(14) TOTAL ALL SEAMS (1)+(6)+(11)	(15) TOTAL SHALE (2) + (7) + (12)	(16) TOTAL COAL + SHALE (14) + (15)
[illegible]N	-	-	-	-	-	-	-	-	68,236	99,000	167,236	23,077	190,313	167,236	23,077	190,313
851,860N	-	-	-	125,066	121,100	246,166	-	246,166	364,145	317,050	701,195	115,296	816,491	947,361	115,296	1,062,657
852,660N	65,512	-	65,512	309,833	286,000	595,833	62,857	658,690	390,364	614,727	1,005,091	-	1,005,091	1,666,436	62,857	1,729,293
853,460N	44,221	-	44,221	375,200	264,467	639,667	-	639,667	334,691	654,727	989,418	-	989,418	1,673,306	-	1,673,306
854,260N	162,142	-	162,142	-	-	-	-	-	-	-	-	-	-	162,142	-	162,142
855,060N	147,402	-	147,402	648,267	322,667	970,934	-	970,934	573,345	132,655	706,000	-	706,000	1,824,336	-	1,824,336
855,860N	155,591	-	155,591	420,567	229,133	649,700	-	649,700	639,782	474,545	1,114,327	-	1,114,327	1,919,618	-	1,919,618
856,660N	-	-	-	169,867	157,300	327,167	-	327,167	546,036	1,187,128	1,733,164	113,407	1,846,571	2,060,331	113,407	2,173,738
857,460N	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
TOTAL	574,868	-	574,868	2,048,800	1,380,667	3,429,467	62,857	3,492,324	2,936,599	3,479,832	6,416,431	251,780	6,668,211	10,420,766	314,637	10,735,403
854,260N	-	-	-	800,567	550,400	1,350,967	-	1,350,967	154,000	686,400	840,400	186,154	1,026,554	2,191,367	186,154	2,377,521
851,060N	165,709	-	165,709	858,700	568,750	1,427,450	-	1,424,450	390,463	445,550	836,013	116,538	952,551	2,429,172	116,538	2,542,710
851,860N	209,417	-	209,417	955,867	617,035	1,572,902	-	1,572,900	257,782	170,181	427,963	85,055	513,018	2,210,282	85,055	2,295,335
852,660N	409,449	-	409,449	338,000	212,333	550,333	68,571	618,904	-	-	-	-	-	959,782	68,571	1,028,353
853,460N	424,189	-	424,189	473,467	104,000	577,467	-	577,467	-	-	-	-	-	1,001,656	-	1,001,656
854,260N	407,811	-	407,811	482,533	192,000	674,533	-	674,533	-	-	-	-	-	1,082,344	-	1,082,344
855,060N	245,669	-	245,669	476,667	137,133	613,800	-	613,800	-	-	-	-	-	859,469	-	859,469
855,860N	237,480	-	237,480	461,267	104,000	565,267	-	565,267	-	-	-	-	-	802,747	-	802,747
[illegible]660N	542,866	-	532,866	1,358,933	391,233	1,750,166	-	1,750,166	-	-	-	-	-	2,293,032	-	2,293,032
[illegible]460N	332,244	-	332,244	804,292	-	804,292	-	804,292	458,545	762,773	1,221,318	-	1,221,318	2,357,854	-	2,357,654
TOTAL	2,974,834	-	2,974,834	6,209,726	2,326,484	8,536,210	68,571	8,604,781	1,106,790	1,378,504	2,485,294	201,593	2,686,887	13,996,338	270,164	14,266,502
PROVEN	574,868	-	574,868	2,048,800	1,380,667	3,429,467	62,857	3,492,324	2,936,599	3,479,832	6,416,431	251,780	6,668,211	10,420,766	314,637	10,735,403
PROBABLE	-	-	-	800,567	550,400	1,350,967	-	1,350,967	154,000	686,400	840,400	186,154	1,026,554	2,191,367	186,154	2,377,521
POSSIBLE	2,974,834	-	2,974,834	6,209,726	2,326,484	8,536,210	68,571	8,604,781	1,106,790	1,378,504	2,485,294	201,593	2,686,887	13,996,338	270,164	14,266,502
TOTAL 3,900'-3,400'	3,549,702	-	3,549,702	9,059,093	4,257,551	13,316,644	131,428	13,448,072	4,197,389	5,544,736	9,742,125	639,527	10,381,652	26,608,471	770,955	27,379,426
TOTAL OUTCROP to 3,900'	9,513,248	684,903	10,198,151	17,956,026	9,742,019	27,698,045	147,428	27,845,473	12,953,048	12,308,190	25,261,238	1,152,767	27,414,005	62,472,530	1,985,098	64,457,628
TOTAL 3,900' to 3,400'	3,549,702	-	3,549,702	9,059,093	4,257,551	13,316,644	131,428	13,448,072	4,197,389	5,544,736	9,742,125	639,527	10,381,652	26,608,471	770,955	27,379,426
TOTAL OUTCROP to 3,400'	13,062,950	684,903	13,747,853	27,015,119	13,999,570	41,014,689	278,856	41,293,545	17,150,437	17,852,926	35,003,363	1,792,294	37,795,657	89,081,002	2,756,053	91,837,054

N O R T H   H I L L

GEOLOGICAL RESERVES FROM OUTCROP TO 3,900' ABOVE SEA LEVEL

[illegible]	(1) SEAM 2	(2) NO.2 SHALE	(3) SEAM 2 +SHALE (1) + (2)	(4) SEAM 4U	(5) SEAM 4L	(6) SEAMS 4U + 4L (4) + (5)	(7) NO.4 SHALE	(8) 4U+4L+SHALE (6) + (7)	(9) SEAM 5U	(10) SEAM 5L	(11) SEAMS 5U+5L (9) + (10)	(12) NO.5 SHALE	(13) 5U+5L+SHALE (11) + (12)	(14) TOTAL ALL SEAMS (1) + (6) + (11)	(15) TOTAL SHALE (2)+(7)+(12)	(16) TOTAL COAL + SHALE (14) + (15)
[illegible]60N	57,638	-	57,638	519,375	364,000	883,375	-	883,375	356,345	423,000	779,345	108,462	887,807	1,720,358	108,462	1,828,820
[illegible]0N	5[illegible]1,354	-	581,354	1,632,566	1,112,590	2,745,156	-	2,745,156	574,254	669,381	1,243,635	-	1,243,635	4,570,145	-	4,570,145
652,[illegible]N	[illegible]38,425	-	938,425	1,831,467	1,543,767	3,375,234	68,571	3,443,805	1,051,600	1,099,327	2,150,927	247,604	2,398,531	6,460,585	316,175	6,780,760
653,460N	1,[illegible]7,8[illegible]9	136,923	2,024,732	2,322,031	1,167,232	3,489,263	-	3,489,263	1,727,853	2,182,434	3,910,287	62,769	3,973,056	9,287,359	199,692	9,487,051
654,260N	1,[illegible]1,069	173,714	1,834,783	2,203,100	1,033,900	3,237,000	-	3,237,000	2,600,436	1,495,090	4,095,526	123,846	4,219,372	8,993,595	297,560	9,291,155
655,060N	1,[illegible]3,857	279,409	1,503,266	2,233,415	1,489,298	3,722,713	-	3,722,713	1,376,507	1,197,453	2,573,960	179,230	2,753,190	7,520,530	458,639	7,979,169
655,860N	[illegible]3,841	-	803,841	1,791,299	1,298,000	3,089,299	-	3,089,299	1,654,545	2,054,598	3,709,143	330,856	4,039,999	7,602,283	330,856	7,933,139
656,660N	1,[illegible]2,601	94,857	1,187,458	1,651,832	959,732	2,611,564	78,857	2,690,421	2,161,325	752,636	2,913,961	100,000	3,013,961	6,618,126	273,714	6,891,840
657,460N	[illegible]3,417	-	380,417	820,875	-	820,875	-	820,875	711,273	1,215,217	1,926,490	-	1,926,490	3,135,782	-	3,135,782
TOTAL	8,6[illegible]5,011	684,903	9,319,914	15,005,960	8,968,519	23,974,479	147,428	24,121,907	12,214,138	11,089,136	23,303,274	1,152,767	25,456,041	55,912,763	1,985,098	57,897,861
654,260N	-	-	-	1,129,566	576,000	1,705,566	-	1,705,566	-	-	-	-	-	1,705,566	-	1,705,566
657,460N	[illegible]8,945	-	258,945	547,250	-	547,250	-	547,250	474,182	810,145	1,284,327	-	1,284,327	2,090,522	-	2,090,522
TOTAL	[illegible]8,945	-	258,945	1,676,616	576,000	2,252,816	-	2,252,816	474,182	810,145	1,284,327	-	1,284,327	3,796,088	-	3,796,088
651,060N	115,276	-	115,276	283,925	145,600	429,525	-	429,525	-	-	-	-	-	544,801	-	544,801
651,860N	92,220	-	92,220	151,867	51,900	203,767	-	203,767	-	-	-	-	-	295,987	-	295,987
655,460N	35,024	-	35,024	-	-	-	-	-	-	-	-	-	-	35,024	-	35,024
657,460N	376,772	-	376,772	837,458	-	837,458	-	837,458	264,728	403,909	673,637	-	673,637	1,887,867	-	1,887,867
TOTAL	6[illegible]9,292	-	619,292	1,273,250	197,500	1,470,750	-	1,470,750	264,728	403,909	673,637	-	673,637	2,763,679	-	2,763,679
PROVEN	9,6[illegible]5,011	684,903	9,319,914	15,005,960	8,968,519	23,974,479	147,428	24,121,907	12,214,138	11,089,136	23,303,274	1,152,767	25,456,041	55,912,763	1,985,098	57,897,861
PROBABLE	[illegible]8,945	-	258,945	1,676,816	576,000	2,252,816	-	2,252,816	474,182	810,145	1,284,327	-	1,284,327	3,796,088	-	3,796,088
POSSIBLE	6[illegible]9,292	-	619,292	1,273,250	197,500	1,470,750	-	1,470,750	264,728	408,909	673,637	-	673,637	2,763,679	-	2,763,679
[illegible] TO 3,900'	9,5[illegible]3,246	684,903	10,198,151	17,956,026	9,742,019	27,698,045	147,428	27,845,473	12,953,048	12,300,190	25,261,238	1,152,767	27,414,005	62,47[illegible],530	1,985,098	64,457,628

PART IV MINING

(Report by Dames & Moore)

4.0 Introduction and Scope of Work

It is anticipated that the Sage Creek project will undergo a major feasibility study, associated with arrangements for senior financing in the near future. Mine design work is required in order to investigate such items as recoverable coal reserves, stripping requirements, equipment selection and mine infrastructure. Further, the whole complex must be developed within a framework of current environmental legislation in British Columbia.

Dames & Moore's involvement in such design work was outlined in a proposal to Rio Algom Mines Limited of January 9,1975. The work was to consist of the actual design of two open pits suitable for delivery of some 3 million long tons of clean coal annually from the coal preparation plant. The pit designs were to include ultimate haul roads, mine services, dump access and preliminary design of run-of-mine waste dumps. Recommendations for additional field work to reinforce any estimates made were also to be identified and discussed in the final report.

A number of constraints were applied to the studies at the outset on account of limited time available. Thus, Dames & Moore were to use geological plans and sections provided by Rio Algom, without further confirmatory drilling for reserves, or geotechnical purposes, nor were field visits deemed useful at this preliminary stage. Such assumptions as were required to complete the work were to be developed jointly with Rio Algom staff, and these assumptions noted in the report.

4.1 Method of Study

In order to expedite the preparation of the two ultimate designs for the North and South Hill open-pit mine operations at Sage Creek, the designs were based upon:

(a) currently available existing geological information

(b) data from Rio Algom's April and August 1974 reports on the Sage Creek project

(c) reasonable assumptions based upon other operators' experience.

Riocanex provided East-West sections at 400 foot intervals for both deposits as well as topographical and other ancillary data.

The design for the two open pits has been treated as an integrated package with power lines, dump locations, haulage roads and the washing plant being regarded as interrelated services rather than separate entities. The study also examined the design from an environmental standpoint, in view of the prominence and importance of British Columbia's existing legislation.

In the course of evolving two feasible ultimate pit designs, the operational projections of equipment, maintenance and manpower requirements and costs were critically compared with data available from other Canadian open-pit mines of equivalent operation and complexity.

Throughout the study frequent discussions have been held with Rio Algom personnel, and any assumptions made have been thoroughly reviewed at each stage.

4.2 Ultimate Pit Designs.

4.2.1 Stripping Ratios

The concept of a stripping ratio is well-known in open-pit or strip mining. Simply stated, it is the ratio of the quantity of waste material which must be removed from the mine in order to obtain a given unit of the desired mineral. The units adopted may be volumetric or gravimetric, depending on the operator's preference, or even mixed, which in many cases is the better choice. In other words, the Stripping Ratio may be expressed as:

(a) tons per ton

(b) cubic yards per cubic yard

(c) cubic yards per ton

However, on closer inspection, it becomes apparent that there are three different and significant stripping ratios to consider:

(i) the Overall stripping ratio (also known as the general stripping ratio)

(ii) the Cut-off stripping ratio (also known as the break-even stripping ratio)

(iii) the Instantaneous stripping ratio (also known as the current stripping ratio)

In order to evaluate the relevance of each stripping ratio to a mining operation, the following concepts must be considered.

4.2.2 The Overall Stripping Ratio

This ratio indicates the total amount of waste material which must be removed in order to obtain the total tonnage of desired mineral. In the case of Sage Creek, the desired mineral is raw coal. The overall stripping ratio is not an operating parameter, per se, but may be used to compare different designs of the same pit or different designs of different pits, assuming similar recoverable mineral reserves.

The overall stripping ratio is used to determine waste dump space requirements and capital equipment requirements. It is obtained only after the Ultimate Pit Design is completed. Significantly, the overall stripping ratio is always less than the cut-off stripping ratio. The overall stripping ratio is dependent on such factors as the shape, dimensions and attitudes of the coal seams, the geometry of the excavation (including the sensitive slope angles), and the topography of the area.

4.2.3 The Cut-off Stripping Ratio

This ratio indicates the amount of waste stripping which can be paid for by a given unit of desired mineral. In the case of Sage Creek it demonstrates how many tons of waste can be economically removed from the pit in order to mine one ton of coal.

Stated slightly differently, mining at the cut-off stripping ratio results in a break-even operation, showing neither profit nor loss.

It can readily be seen that the cut-off stripping ratio (C.O.S.R.) is the primary tool used to determine the economic pit limits, leading to the design of the Ultimate Pit. The C.O.S.R. is susceptible to changes in operating cost, product price, processing and transportation costs, and waste stripping costs. It is not traditionally sensitive to capital costs since in most cases they are not firmly established at the time the pit is designed and are handled in the cash flow analysis.

At Sage Creek the following calculation indicates how the ultimate C.O.S.R. was derived.

The general expression may be stated as follows:

$$\text{C.O.S.R.} \doteq \frac{\text{Value of in-situ coal/ton - average production cost/ton}}{\text{average stripping cost/ton waste}}$$

Using Rio Algom's estimates of operating costs, (Sage Creek Coal Project, Interim Review Aug. '74.)which were reviewed by Dames & Moore and generally accepted, the overhead costs were redistributed in terms of dollar values in order to allocate overhead costs to the mining of waste. The selling price of coal, F.O.B. Vancouver, was assumed to be $30 per ton.

Then, considering the in-situ long ton of raw coal as the basic unit:

	$/ton	Yield Factor	Extended $/ton
Recoverable value	30.00	0.57	17.10
Transport (Rail and Spur Amortization)	8.75	0.57	6.61
Terminal Charge	1.10		
Royalty	1.50		
Property Taxes	0.25		
Mining cost/raw ton	0.358	1.00	0.358
Process cost/raw ton	0.705	1.00	0.705
Stripping cost/ton waste	0.358	1.00	0.358

so,

$$C.O.S.R. = \frac{(17.10 - 6.61) - 1.06}{0.358} = \underline{\underline{26.34}} \text{ tons/ton} \quad **$$

The actual unit mining cost is seen to be fairly insensitive when considered in the light of some of the assumptions made, e.g. the potential hydrological situation at the North Hill; the optimum slope angles in the overburden, clays and hanging-wall conglomerates; the selling price of clean coal; transportation costs; etc. However, it will also be seen that the application of any revised C.O.S.R. will change somewhat the size and shape of the ultimate pits and may change the overall stripping ratio. The process is an iterative one and should be optimized during the definitive feasibility study.

Note: **

Since these calculations were completed a new coal price has been established, and operating costs recalculated. The following table and calculations reflect the increase in maximum stripping ratios that can be attained.

	$/ton	Yield Factor	Extended $/ton
Recoverable value	47.00	0.57	26.79
Transport (Rail and Spur Amortization)	11.25		
Terminal Charge	1.10	0.57	8.04
Royalty	1.50		
Property Taxes	0.25		
Mining Cost/raw ton	0.431	1.00	0.431
Process Cost/raw ton	0.85	1.00	0.85
Stripping Cost/ton waste	0.431	1.00	0.431

so,

$$\text{C.O.S.R.} = \frac{(26.79 - 8.04) - 1.281}{0.431} = 40.5 \text{ tons/ton}$$

This demonstrates that the C.O.S.R. is increased by 14.1. This increase allows the ultimate pit level to go to approximately 3000 feet elevation. However, constraints such as the Howell creek and geotechnical problems must be investigated carefully before an ultimate pit bottom can be established. This will be done as a part of the final feasibility study.

4.2.4 The Instantaneous Stripping Ratio

This ratio applies to a specific point in time, or to a given period, such as one month. Consequently, the instantaneous stripping ratio is a production scheduling tool and is of prime importance in determining the profitability of a mine. The correct stripping ratio at any time is that which will maximize the present value of total future profits. This makes this particular ratio sensitive to real changes such as an increase in mining costs without an equivalent dollar increase in selling price. In large copper mines where the price of metal fluctuates almost on a daily basis, the ratio may also be adjusted to compensate for expected future market fluctuations. This is risky and not considered desirable for Sage Creek.

Consequently, the instantaneous stripping ratio may range anywhere from infinity (as at the very first opening of a new mine) to zero (as in mining the last coal in the life of the pit). However, it must be clearly understood that the integrated sum of instantaneous stripping ratios, over the life of the mine, must equal the overall stripping ratio. There are many documented cases of operators neglecting this basic fact. They reduce their instantaneous ratio to an unreasonably low level for a protracted period. Eventually they "run out of ore".

Figure 1, (Page IV -10) is an idealized graph showing how the instantaneous stripping ratio (converted to cubic yards per raw long ton) might be expected to change with time. It also demonstrates how the second pit (in this case the South Hill) could be brought into production at the proper time in order to smooth out the project's equipment requirements, maintain production, and equalize stripping.

This type of scheduling is complicated by the need to plan the construction of waste dumps with almost as much care as the pit designs. Obviously, with well over one billion cubic yards of waste to dispose of, and in accordance with environmental requirements, any major fault in scheduling waste removal could seriously jeopardize the project's viability.

4.3 Method of Mining

4.3.1 General Description

The fundamental policy that underlies the Mining Method considerations described herein is one that stresses an orthodox approach in which no experimental or unproven technique is recommended. However, some novel items are presented that reflect the scope of the intent of new Mining Environmental Legislation. The production and waste handling requirements are such that emphasis has been placed upon large sized operating equipment, where such units have been proved by significant field performance. This policy has the added benefit in reducing overall manpower requirements, for an area where difficulties in attracting and maintaining

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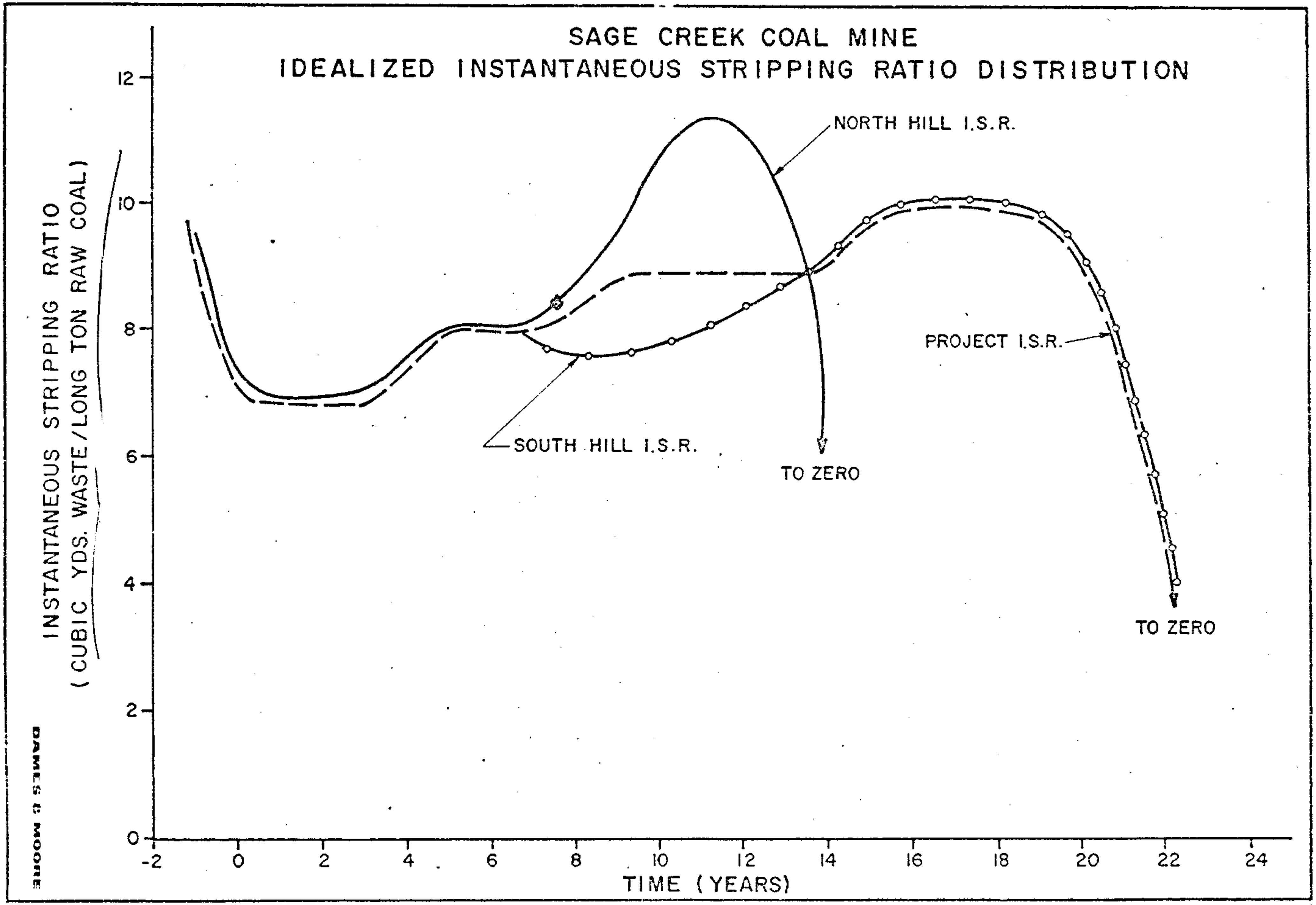

a skilled work force may be anticipated. In calculating capital operating equipment requirements allowance has been made for lost days due to inclement weather, and a conservative approach adopted towards exposing coal reserves in the field, in which this work is undertaken only during daylight hours.

4.3.2 Mining Sequence

The evidence available on the geological sections developed to date indicates that mining should commence on the North Hill, where lower unit operating cost may be anticipated. Here the coal is relatively unfaulted, and has an overall uniformity of dip and strike. The reserves on the North Hill are greater than those of the South Hill, and the North Hill is not influenced to the same extent by deep clay deposits overlying the coal seams. It should prove possible to bring all operating crews and supervisory staff to efficient performance levels earlier than if the South Hill was mined first.

Operations should commence at the upper levels of the mine (circa 5,200 elevation) and advance down dip, over the full strike length. Such a sequence developed over a limited number of benches, will allow deferment of the maximum amount of stripping, provide for easier movement of shovels and drills and reduce overall road maintenance, when compared to mining a greater number of benches over a limited strike length. There are approximately 12 years of reserves in the North Hill, and analysis of stripping requirements and coal blending considerations indicate that the first mining at upper levels of the South Hill should commence about year 8 after start up of coal production.

One factor of great importance in this design is the dump layout and the timing in construction of particular dump areas. Thus, while traditional practice might suggest that the early stripping is dumped off the ends of the developing pit, the net result would be to cause lower level stripping to be hauled greater horizontal and vertical distances. Clearly, fleet requirements are closely related to such operating plans, and these concepts should be fully developed in more definitive work.

Operations could commence with the removal of all trees from the area to be stripped and the stockpiling of any soil encountered, for use in subsequent reclamation activities. The minimum amount of such work compatible with satisfactory access should be undertaken at a given time, as this will minimize environmental impact. For the upper benches a horizontal slice may be taken from the footwall of number 5 seam to daylight, at existing topography, and as mining progresses in depth, increasing amounts of waste may be left behind, overlaying seam 2.

Conventional drilling and blasting of waste rock is envisaged, for the bulk of the stripping. The wedge of material that overlays a given coal seam, may be removed by a combination of blasting, and ripping and pushing of broken material. Depending upon the coal thickness at a given location, the coal may be mined by a number of methods, and it is assumed throughout that the coal will not require blasting. In thick seams (say +40 feet) about one quarter of the coal may be free dug by shovel. The remaining coal will be ripped and pushed towards the pit floor by tractor, where it may be gathered by a front end loader. The wedge of waste material underlying a given coal seam may be removed by three possible techniques, depending upon its local constituents and attitude. The softer, more gently dipping strata may be ripped and pushed as with the coal, while air-track equipment may be utilized to drill harder formations for subsequent blasting. A minimum amount of air-track work is foreseen. More steeply dipping, harder underlying waste may be rotary drilled, off a pad of material dumped back over these strata.

The overall efficiency of the operation to a large extent will be influenced by the refinement possible in cleaning waste from above the coal, and in leaving minimum coal above underlying waste. It is assumed throughout these studies that coal losses of 1 foot are possible at the top and bottom of each seam, and that it is considered more advantageous to accept such coal losses rather than to inundate the preparation plant with waste rock. Where waste rock of less than five feet in thickness separates two adjacent coal seams it has been assumed that such waste will have to be extracted with the coal.

Grade control requirements suggest that each coal seam should be exposed at a number of locations on a given bench. The strike length of the coal is such that this may readily be achieved by commencing operations from each end of the deposits, and a third group of faces may be developed about the centre of the strike in the case of the North Hill.

As with any large scale open pit operation, road maintenance and local drainage are of great importance.

Some special problems may have to be overcome, should it be required that the foot-wall slope be revegetated, as is possible after mining, as is indicated by recent legislation. On the one hand is the problem of introducing and retaining a satisfactory soil onto a slope of some 30°, and on the other is the general problem of encouraging growth at altitudes in excess of 5000 foot elevation.

4.3.3 Equipment Selection

Annual production of 3,000,000 long tons of clean coal is equivalent to a raw coal extraction of 5,263,000 long tons assuming a plant recovery of 57%. While the instantaneous stripping ratio may vary during the life of the operation, the bulk of the waste material will be mined at a ratio of approximately 9 yds. of waste per ton of raw coal. Accordingly, initial equipment sizes and numbers may be estimated using these factors.

Figure 1 shows, in an idealized manner, one strategy for distribution of the rate of stripping over the life of both the North and South Hill deposits. Despite the cash flow advantages to the project of deferring early stripping, a number of physical difficulties may be expected towards the end of the operating life of the complex. Thus, inspection of the equipment requirement at say, year 18, shows that six shovels, and twenty-six trucks in the 170 ton range are scheduled to operate from the lower levels of the

South Hill. In practice, there may not be sufficient operating space available to allow this configuration to be effective, and an alternate strategy would need to be developed by about year 15.

There is increasing evidence that trucks with a capacity of 170 tons are proving themselves in Canada, and it is anticipated that all development problems should be cleared by the time the Sage Creek project would come on stream. Teamed to a 20 cubic yard shovel, this size unit may be recommended for stripping. The development of large front end loaders appears to be lagging truck size increases, and the most reliable loaders have bucket capacities of 15 cubic yards rather than 22 cubic yards. Accordingly, trucks in the 170 ton range teamed to 15 cubic yard loaders are proposed for coal loading operations, that will not be performed by the shovels. Increased selectivity should also result in the adoption of the 15 cubic yard loaders.

Capital costs are given in current dollars, and include an amount for spares inventory. Attention is drawn to the fact that Dames & Moore estimate current inflation of capital items at the rate of 1% per month, and this factor, together with pre-delivery and progress payments on major equipment orders may significantly influence cash flow estimates. Lead times required for delivery have not yet been examined in detail, but a current estimate is 12-18 months for trucks, loaders, dozers and drills, with 18-30 months for shovel units.

4.3.4 Mine Design

The design of the open pits is constrained by the physical characteristics of the coal deposits and the operational requirements associated with the extraction of the coal.

In determining the economic pit bottom for both pits, 50 foot high benches were assumed, and taken to coincide with regular surface elevations, i.e. at 3000 feet and then 50 foot intervals.

A pit bottom width was examined and rather than have a pit bottom which exposed all coal seams down to a certain level, the alternative of a minimum width of 200 feet was proven to give the optimum coal recovery for any given set of pit limits. This minimum is determined by the size of the equipment to be used in mining. For the size of trucks and shovels under consideration at Sage Creek, the accepted minimum in the industry is 200 feet.

In determining the overall shape of each pit, the bottom of seam 5, that is the footwall sandstone, was assumed to provide a stable slope on the western wall. The natural slope of the footwall is $30° \pm 3°$. Where faulting occurs, particularly on the South Hill, the resultant steps reduce the overall slope angle. The existing geotechnical data on the overlying overburden, gravels (on the North Hill) and clays (on South Hill) are minimal. Accordingly, a conservative estimate has been made by Dames & Moore's geotechnical engineers in order to allow design work to proceed.

These angles are 45° in rock and 30° in clay and gravel. This assumes that the gravels will be dewatered. Field testing and analysis will be required before any of these slopes may be considered definitive.

In order to improve safety, a 50 foot berm at the rock/gravel or rock/clay interface was incorporated into the design. The berm was provided to catch any material which might degrade, and allow access for dozers to clean up this material. In practice, the berm would be sloped away from the pit edge to allow for natural drainage to run away from the pit. Subsequent geotechnical work may increase the slope angles in any future design, but these current assumptions are conservative ones and allow for greater safety in this preliminary design.

The edge of the pit was constrained in part by the proximity of the Howell and Cabin Creeks. It was thought prudent and reasonable to limit the pit edge to no less than 400 feet from the river banks as identified by the large scale topographical maps available.

Smooth edges to the pit outline were built into the design to provide for greater geotechnical stability and often resulted in a pit bottom of over 200 feet.

The siting of the main access and haulage roads into the pits was selected to be the lowest topographical level around the pit crest. In the North pit this occurs in the south east corner, and in the South pit in the north east corner. These entrances are adjacent to one proposed site for the washing plant. The roads grade at -8% from the entrance, remaining in rock where possible and on the eastern edge of both pits down to the pit bottom. The siting of the ultimate road on the eastern wall of the pit is necessary since only on this wall could it be maintained, on grade, in competent rock. The natural angle of the footwall slope on the western edge would require the road to be either cut into the slope, thereby undercutting the footwall slope, or built up on fill material dumped on the footwall slope. Both of these options were considered to be technically unsound for geotechnical and operational reasons. Furthermore, this siting would have exposed the road to potential avalanche hazards from the long shallow slope, which is facing the winter sun. The road is designed to be 100 feet wide, which includes a 20 foot drainage ditch on the inside edge. Final equipment selection may cause a slight change in this configuration, but should not cause any significant change in the pit economics.

The roads, as designed, as ultimate haul-roads, would not be present until the latter stages of the mine's life. Allowance for interim roads coming out of the pit at higher elevations along the contours would be necessary in actual operations.

Some preliminary thought has been given to the availability of power for the electric shovels in the pits. Operational experience suggests that the main power line should loop the pit in order to provide a measure of safety in ensuring supply of power. Should one part of the circuit fail, then the power supply would still be available.

This loop is of particular importance in the North pit, where early indications are that some continuous dewatering of the gravel to the north and east of the ultimate pit will be necessary.

Consequently, the pit designs incorporate a looped powerline, supplied from the main powerhouse. This electricity supply would be stepped down through transformers before feeding the electric shovels at 4160 V.

4.3.5 North Hill

In applying the C.O.S.R. to the North Hill sections, the economic bottom was shown to be at 3400. This represented the maximum depth of the pit, attainable from existing geological and geotechnical data. The other limiting factors previously mentioned, particularly the proximity of the rivers, produced a pit of 6000 feet in an east-west direction and 8000 feet in the north-south direction, of which 3000 feet reached the economic bottom of 3400 feet.

The depth of the pit requires the haulage road to decline, from 4320 feet elevation in the south east corner, to the pit bottom of 3400 at 8%. This requires approximately 7260 feet of roadway, all of which has been incorporated into the eastern face of the pit. The road declines to the north up to section 856,660 N approximately, then it doubles back and declines to the south.

4.3.6 South Hill

The calculation of an economic pit-bottom for the South Hill indicated that a depth of greater than 3300 feet could be achieved. However, the geological information available on the coal seams and their thicknesses below 4000 feet is weaker than for the North Hill. This is a consequence of the changed in-situ nature of the coal in the South Hill, and the presence of many faults in the upper reaches whilst these faults appear to be absent in the deeper, western reaches.

The decision was made therefore to limit the South Hill to the same overall depth as the North Hill, i.e. 3400 feet. However, when it came to designing the pit, with a 100 foot road, declining from an elevation of 4275 at the north east corner, further constraints were introduced. The South pit has a shorter north-south axis (length) and a longer east-west axis (width). The road needed to be kept in competent rock formations, rather than the overlying clay, on the long western-sloping footwall. This consideration moved the pit-bottom up to 3600 feet, where it was possible to design in an 8% haulage road which did not consume excessive stripping.

4.3.7 Dump Design

Over the life of the mines there will be a requirement to dispose of some 1.3 billion cubic yards of run-of-mine waste rock and approximately 40 million cubic yards of plant reject. Clearly the effective handling of such volumes of material will have an important bearing on overall project profitability. A number of fundamentals of dump design may be identified, perhaps the most obvious being that of the selected site proximity to the mines. Constraints on these considerations would include land ownership and topographical features. Potential dump areas have also to be cleared of the possibility of sterilization of future mineral resources. A number of geotechnical aspects are also important; prime amongst these being the constituents of the base material, and its attitude with respect to bedrock. Limits may also be imposed upon planned dump heights by bulk cohesion and friction angles anticipated in the waste products. Hydrology is important in that dumps have been undercut and eroded in the past through the repeated action of flowing water, and also to be considered is the effect of surface water contamination from fines run-off. Environmental factors influence the base preparation prior to dumping and also the method of dump construction with respect to the timing of reclamation activities.

At the operational level the sequence of dump construction has influence upon truck fleet requirements. A trade-off between early dumping in close proximity to the ultimate pit, and later, more distant, dumping from material mined from the pit depths has to be analysed in detail.

Dumps constructed in lifts of between 50'-100' are inherently more stable than dumps created by tipping waste over increasingly high banks, and each lift may be revegetated, as soon as it is filled to ultimate.

It is reasonable to assume that the final plant location may be in the area to the east of both ultimate pits at a point close to the centre of mass of the coal reserves. There is evidence of additional coal to the south and west of the South Hill, which would discourage selection of this area also for run-of-mine waste dump purposes. Topographic plans to a suitable scale are unavailable over a number of locations that may logically be examined for site selection, but three sections have been prepared showing a typical configuration for dump areas to the immediate west of the mines. The dumps here have not been projected at heights over 200 feet to 300 feet above the elevation of mining, and it may be noted that some clearance to the Howell and Cabin creek has been allowed (Figure 2) App.lV. A further potential dump site lies to the north west of the North Hill, between the Howell Creek and Flathead River, and for the South Hill an area to the south east may prove viable. Assuming that the reserves in the North Hill will be mined first, then significant volumes of waste may be dumped back in this pit, from stripping on the South Hill.

While examinations of the options available for tailings disposal does not yet appear complete, the advantages seem to lie with trucking a mixture of filtered and pressed -28 mesh reject combined with 11/2 inches to 28 mesh refuse, mixed if necessary with mine waste. The site selected for this dump should be in close proximity to the plant area, on a base that may be readily prepared to structural and environmental requirements.

Allowance for truck requirements for these tailing operations has not been made under Mine Equipment summary. The above discussion, based on broad principles, further suggests that a detailed search for the optimum plant site be closely integrated to overall dump requirements, since it may prove more economical to move coal further to an alternate plant/tailings pile location, and utilize the land so freed for dump construction.

4.3.8 Coal Reserves

The calculation of Coal Reserves of a mineable nature has been based upon the east-west sections provided by Riocanex. Each section has been given a projection of 400 feet (200 feet north and south of the section line). In the sections at the extreme north and south of each pit, the projection has been adjusted to compensate for the end effects of the two open pits.

All coal down to a depth of 80 feet below the natural surface was assumed to have been oxidized. Losses attributable to mining have been conservatively accounted for by deducting one foot of coal (measured perpendicular to the dip) from the top and bottom of each seam.

When the waste between adjacent coal seams is thin, some operational problems are anticipated in mining it separately.

For this reason, thin waste bands have been regarded as inseparable from the coal when they are less than 5 feet thick, and have been included in the calculated coal reserves. Similarly, any coal seams less than 5 feet in thickness have been regarded as unmineable and therefore not included in coal reserves. The oxidized coal and the losses in mining have been included in the volumes of waste material to be moved.

These deductions from the quantities of coal available for delivery to the washing plant amount to 16.9% of gross in-situ coal for the North Hill and 15.2% for the South Hill. Therefore, the mining recovery is 83.1% for the North Hill and 84.8% for the South Hill, (overall average 83.8%).

The volume of coal available has been converted into long tons of raw coal using the following data:

Coal Seam	5	4	2	Waste
Cubic feet per long ton	22.0	24.0	25.4	18.2

The resultant coal reserves, that is raw coal available for delivery to the wash plant, is summarized in Table 1 Page IV -25

TABLE 1 - SAGE CREEK COAL RESERVES

Seam	North Hill '000 l.t.	South Hill '000 l.t.	Total '000 l.t.
5	26,477	18,990	45,467
4 lower	10,115	9,037	19,152
4 upper	17,334	14,893	32,227
2	8,800	2,871	11,671
Thin Waste Seams	443	799	1,242
Total Coal	63,169	46,590	109,759
Associated Waste in '000 cubic yards	614,644	400,866	1,015,510
Waste:Coal Ratio	9.7	8.6	9.2

The volumes of waste are expressed as bank cubic yards. When blasted and loaded, a swell factor of 1.25 has been applied, in determining the size of required dump areas and fleet requirements. Complete coal reserves are tabulated in Tables I and II, Pages IV - 27 & 28.

4.3.9 Plant Location

The proposed coal preparation plant for Sage Creek is planned to handle over five million long tons of raw coal per year for a clean product amounting to three million long tons per year.

It is anticipated that conventional slurry lagoons may not be used at Sage Creek; however, filter tests have indicated the viability of fines disposal as solid waste.

The necessary economic operating and environmental factors to consider in siting the plant may be tabulated as follows:

(a) regional topography

(b) distance of plant from the centre of mass of coal in each pit

(c) provision for raw coal storage and blending

(d) provision for product storage and disposal (i.e. rail access potential)

(e) waste disposal facilities

(f) potential secondary useage of same site (i.e. mine-dry, warehouse, maintenance shops, etc.)

(g) foundation conditions, including groundwater hydrology

(h) environmental considerations (i.e. legislative constraints, climatic conditions, aesthetics, surface water and groundwater quality, etc.)

Most of the above items of concern are fairly well defined for the purpose of this study. The areas which have not been investigated fully at this time are the last two - the foundation conditions and a detailed environmental impact study. Each of these factors, when fully investigated, may over-rule decisions made at this time.

However, based on a consideration of the other major factors the plant location is recommended near the confluence of Cabin and Howell Creeks. This is shown on Figure 5, App. IV.

TABLE I

SAGE CREEK - NORTH HILL COAL RESERVES

SECTION	COAL ('000 long tons)						TOTAL WASTE in '000 cu.yds.	RATIO Waste:Coal
	#5 Seam	#4 Seam Lower	#4 Seam Upper	#2 Seam	Waste-5ft	Total Mineable Coal		
850660N	-	-	100	124	-	224	7,341	32.8
851060N	198	297	539	98	22	1,154	15,678	13.6
851460N	1,106	659	975	210	-	2,950	33,260	11.3
851860N	1,430	699	1,151	233	-	3,513	39,645	11.3
852260N	1,260	775	1,098	435	-	3,568	44,974	12.6
852660N	1,453	865	1,188	500	-	4,006	46,859	11.7
853060N	1,536	806	1,250	800	-	4,392	50,840	11.6
853060N	2,059	721	1,460	1,114	-	5,354	50,006	9.3
853860N	1,645	1,080	862	735	-	4,322	50,568	11.7
854260N	2,253	839	1,850	862	107	5,911	49,332	8.4
854660N	1,815	852	1,094	827	50	4,638	38,272	8.3
855060N	1,836	758	1,225	675	138	4,632	40,220	8.7
855-60N	1,933	320	825	544	-	3,622	35,032	9.7
855860N	1,901	499	889	377	26	3,692	28,935	7.8
856260N	1,458	227	587	458	-	2,730	27,043	9.9
856660N	1,491	310	991	417	100	3,309	22,244	6.7
857060N	1,360	297	784	371	-	2,812	18,870	6.7
857-60N	1,492	111	466	20	-	2,089	11,922	5.7
857860N	251	-	-	-	-	251	3,603	9.8
TOTAL NORTH PIT	26,477	10,115	17,334	8,800	443	63,169	614,644	9.7

SAGE CREEK - SOUTH HILL COAL RESERVES

SECTION	COAL ('000 long tons)						TOTAL WASTE in '000 cu.yds.	RATIO Waste:Coal
	#5 Seam	#4 Seam Lower	#4 Seam Upper	#2 Seam	Waste-5ft	Total Mineable Coal		
543860N	-	-	-	-	-	-	7,000	-
544260N	-	-	-	-	-	-		-
544660N	793	134	172	-	-	1,099	23,189	21.1
545060N	922	408	721	34	-	2,085	33,208	15.9
545460N	1,388	759	1,410	168	175	3,900	42,638	10.9
545860N	2,038	1,160	1,804	210	248	5,460	48,476	8.9
546260N	2,547	1,200	1,752	613	-	6,112	45,464	7.4
546660N	1,846	1,294	1,863	418	118	5,539	45,355	8.2
547060N	2,959	1,034	1,322	405	-	5,720	44,461	7.8
547460N	2,027	1,032	1,918	502	102	5,581	42,675	7.6
547860N	2,877	1,233	2,619	368	156	7,253	34,774	4.8
548260N	1,388	662	1,093	111	-	3,254	26,291	8.1
548660N	205	121	219	42	-	587	7,335	12.5
TOTAL SOUTH PIT	18,990	9,037	14,893	2,871	799	46,590	400,866	8.6
TOTAL FOR NORTH AND SOUTH PITS	45,467	19,152	32,227	11,671	1,242	109,759	1,015,510	9.2

4.4 General Comments and Recommendations

In the course of developing the ultimate pit designs presented herein, a number of assumptions have been made, which have been generally discussed with Rio Algom staff members. These assumptions concern the pits, the waste dumps, the tailing pile and plant location, and may be grouped under consideration of local geology, geotechnics and hydrology. Environmental aspects pervade all areas.

4.4.1 Geology

It is understood that a significant fill-in exploration drilling program is proposed for reserve evaluation purposes during the summer season 1975. Analysis of the sections prepared indicates that reserve information is weakest for coal in both hills below the 4000 foot elevation.

To date, no drilling appears to have been undertaken to clear potential dump sites, although it is realized that much of this input may be available from more general published regional mapping. There exists some uncertainty concerning the effect on coal quality by, and losses to be anticipated adjacent to, local faults, and a program of adit work has been suggested as one way of obtaining an understanding of this point.

The work noted above may be undertaken solely for reserve or structural purposes, but many economies may be effected by integrating geotechnical and hydrological investigations.

4.4.2 Geotechnics

Slope stability factors have a direct bearing on the ultimate pit designs. The footwall slopes have been assumed capable of standing over their full exposure, whereas in fact a number of major failures have recently been recognized to be associated with bedding and fault attitudes (e.g. Frank slide, Alberta). For this reason, either core samples or underground exposure to the footwall strata would provide valuable input. Surface trenching and mapping in the footwall would also be appropriate. The slope angle of the east pit walls has been selected at 45°, which is a traditional, conservative value. In fact, the attitude of the bedding planes is such that perhaps a steeper slope angle could safely be applied, which would favourably affect the overall economics. The angle of slope in the gravels and clay overlying the bedrock is 30°, but this angle may reasonably lie within a range of, say, 18° to 35°, depending upon the slope constituents and groundwater regime.

A number of natural slump features may be noted in the site area, and as a generalization, it would not be good practice to design mine dumps over such material. Dump stability is related to base preparation and in turn to sub-surface bedrock topography and the soils/clay ground cover. Strategies for soils stockpiling for subsequent reclamation activity cannot be developed without an inventory of resources related to requirements.

The plant location suggested is also influenced by the geotechnics of foundation design.

4.4.3 Hydrology

The ultimate pit design of the North Hill has been constrained by the proximity of Howell Creek, but no estimate has been made of pumping requirements associated with mining below the groundwater table. The possible influence of Cabin Creek upon either pit has not been postulated, since the hydrological properties of the rocks in the area are essentially unknown. Indeed the groundwater regime over the whole site is as yet undetermined.

Background levels of sediment loading and seasonal values of total flow of the major surface water systems are of importance from an environmental standpoint. Dump design is also constrained by surface water activity.

Annual projections of snow cover and run-off are of interest from both the point of view of avalanche potential and pumping and ditching design philosophy.

It should prove possible to develop a pattern of test borings to address most of the concerns identified above in a manner designed to optimize the value of each probe, and to select an integrated mix of field studies, to investigate specific factors.

Environmental considerations, discussed earlier in the report, are closely related to geotechnical and hydrological studies, in addition to having their own special influences. Generally, preparation for handling potential problem areas is best established early in the life of a major project, through preliminary baseline surveys and impact assessment studies.

It is recommended that Rio Algom Mines extend in detail the concepts noted, and prepare to investigate these areas over the course of the 1975 field season.

5

PART V COAL PREPARATION

Introduction

This section of the report consists of a description of the plant production installations from the point of raw coal delivery from the mine to the loading of clean coal in rail cars. It consists of four main sub-sections.

5.1 - Raw Coal Crushing, Stockpiling and Blending
5.2 - Coal Preparation Plant
5.3 - Clean Coal Storage and Loadout
5.4 - Quality and Blendability of the Coal

5.1 Raw Coal Crushing, Stockpiling and Blending

The raw coal is hauled from the pit by trucks to the breaker where it is discharged into a hopper through a grizzly with 12" square openings, and oversized rock slabs or frozen chunks will be broken by a pneumatic pick located at the base of the grizzly bars. The coal is then discharged from the hopper via vibratory feeder and conveyer into a rotary breaker. The +1½" waste material from the breaker is discarded, while the -1 1/2" material is conveyed to the blending plant which will include both stacking and reclaiming units. The latter will move 18,000 tons per day.

The coal will be sampled prior to being stacked in one of the two stockpiles. These samples will be analyzed for ash, volatiles and F.S.I. The stacker unit will spread the raw coal in successive layers over a period of 24 hours, when the stockpile will be completed with overall consistency in ash, volatiles,and F.S.I. specifications.

The reclaim unit will remove the coal at right angles to the stacker spreader for feeding to the preparation plant. A sample system will be set up between the reclaim unit and the preparation plant to recheck the analysis of the raw coal plant feed.

The complete and extensive blending arrangements as described above have not been used by current Western Canadian coal producers, however, their experience to date indicates that an effective blending system would have been an asset, and in the case of Sage Creek, such a facility is thought to be essential.

Because of its exceptionally good coking quality, No. 2 seam coal will be stockpiled and washed separately. The raw feed from seams 4 and 5 will be blended and washed as a mixture.

The capacity of the reclaim system will operate at an hourly rate of 680 tons, which corresponds with the hourly capacity of the preparation plant. The stacking and reclaiming systems have the capability to operate 30% higher than the preparation plant capacity. An arrangement to by-pass the blending plant will be provided for use when required.

5.2 Coal Preparation Plant

The preparation plant flow diagram is shown in Drawing 151-SK39. App II.

5.2.1 1 1/2" x 28 Mesh Treatment Section (Heavy Medium Cyclone)

The coal preparation plant feed is distributed into two streams. Each stream feeds, in turn, three sieve bends and three 6' x 12' desliming screens with 28 mesh openings. The screen oversize (1 1/2" x 28 mesh) of each stream is discharged into a heavy medium cyclone, where the coal is mixed with magnetite slurries of pre-determined specific gravity. The coal and the magnetite mixture is then pumped to a set of 24' diameter heavy medium cyclones. The cyclone overflow which contains the desired clean coal product is discharged to a set of sieve bends and drainage screens where the coal is washed with water sprays to remove the adhering magnetite. After being discharged from the screen, the coal is further dewatered by a set of centrifuges. The centrifuged coal is then finally conveyed to the thermal dryer.

The underflow from the heavy medium cyclones is passed over a set of sieve bends to recover the entrained medium. The coal is then fed into a secondary cyclone feed cone where it is mixed with magnetite at higher specific gravity than in the separation described above. The coal and magnetite mixture is again pumped to another set of heavy medium cyclones. The cyclone overflow, which contains the thermal coal, is passed over a set of sieve bends and drainage screen, and finally conveyed to the power plant.

The cyclone underflow (refuse) from above is drained through a set of screens to recover the magnetite. The refuse is discarded as waste and trucked to the waste pile.

5.2.2 28 Mesh x 0 Treatment Section

The 28 mesh x 0 coal from the 28 mesh desliming screen mentioned above is pumped to a set of Primary hydro cyclones (or water only cyclones). The primary hydro cyclone underflow is retreated by a set of secondary hydro cyclones. The underflow is discarded as refuse to a 250' diameter thickener. The overflow is recycled as dilution to the primary hydro cyclone feed.

The primary hydro cyclone overflow is pumped to a set of sieve bends with 100 mesh openings. The + 100 mesh material is discharged for dewatering to a set of three 12' - 6" x 14 filters.

The - 100 mesh material is pumped to a set of thickening cyclones. The cyclone overflow is recycled to black water storage for reuse. The underflow is conditioned with kerosene and subsquently floated with MIBC in two banks of flotation cells. The flotation concentrate which contains clean metallurgical coal is discharged to the three disc filters mentioned above.

The tailings from the flotation circuit is discharged to a 250' diameter thickener.

5.2.3 Clean Coal Drying

The disc filter product joins the 1/2" x 28 mesh clean metallurgical coal to be conveyed to the thermal dryer.

The dryer is a fluidized bed type. Hot air utilized to dry the coal is produced by a series of steam coils and electric heating elements. This is covered under the "utilities" section of the report.

Fine coal particles in the air stream will be removed by a series of mechanical separators followed by a venturi scrubber. The equipment is designed to comply with the objectives of the B.C. Pollution Control Branch.

The dryer product including the separated fines will then be conveyed to the storage silos.

5.2.4 Refuse Disposal

Both the flotation tailings and the secondary hydro cyclone underflow are discharged into a 250' diameter thickener. The thickener underflow is dewatered by a set of four solid bowl centrifuges. The centrifuge cake is conveyed to a storage pile and subsquently trucked to waste dumps.

The concentrate which still contains small amounts of suspended solids is clarified by two 4' x 6' x 100 frame filter presses. The filter cakes join the centrifuges cake mentioned above to be disposed.

The thickener overflow water and the filter press filtrate is recycled for process use.

5.2.5 Medium Recovery

Correct medium in the heavy medium separation section will return by gravity to the individual feed boxes.

The dilute medium from all plant sources will be received into a dilute medium tank, from where it is pumped to a primary magnetic separator. The magnetite is therefore collected and discharged into an overdense tank.

The effluent from the primary magnetic separator is collected in a slime tank and then pumped to a battery of thickening cyclones. The cyclone overflow is recycled as a spray water on deslimed screens. The cyclone underflow is fed to a secondary magnetic separator to ensure that all the magnetite is recovered. The reclaimed magnetite is discharged into the same overdense tank mentioned above.

5.2.6 Medium Control

The magnetite slurries in the overdense tank will be pumped continuously to a splitter box from where the magnetite slurries are fed by gravity into all individual heavy medium separation circuits. The flow to each circuit is regulated by pneumatic cylinders which are in turn automatically controlled by the medium density controllers. By means of this device the pre-determined density of separation medium (magnetite slurries) can be automatically and precisely controlled.

5.2.7 Reagent Storage and Distribution

Flotation reagents, kerosene and MIBC will be stored on ground level and pumped to overload distribution tanks from where they are regulated by reagent feeders to points of addition in the flotation circuit.

Settling agents will be stored and mixed on ground level and fed to the 250 foot diameter refuse thickener as required.

Magnetite will be received in bulk from pneumatically unloaded truck tankers into a magnetite storage hopper. A variable speed pneumatic feeder will extract magnetite and deliver it at pre-determined rates to a mixing tank, where water would be added to give a medium of approximately 2.1. S.G. It is pumped to the over-dense tank as required.

5.3 Clean Coal Storage and Loadout

This system consists of a set of four 10,000 ton storage silos and a 200 ton loadout bin over the rail line, each served by belt conveyors. See drawing 151-SK43 App. III. Three of the four silos will handle a mixture of clean coal from seams 4 and 5 while the fourth silo will handle clean coal from seam No. 2. The loadout system described below will mix these clean coals from the silos to a set specification before the final product is loaded into the cars.

A system of 30" conveyors carries dried coal from the drying building to the top of the storage silos at a rate of approximately 400 TPH, where it is distributed by means of a tripper. The two types of coal will be fed simultaneously to a 72" wide conveyor for delivery to the loadout bin at a rate of 3,500 TPH, where it is loaded into rail cars. Totally enclosed conveyor galleries are pro-provided to ensure against wind losses, and the effects of rain & snow.

Storage for four days' production is provided by the four 10,000 ton silos, as required by CP Rail to meet possibilities of unit train tie-up in winter. In an emergency an additional one and one half days of production can be stored in two open stockpiles, filled from chutes from the distributing conveyor on top of the silos. Reclaim is by front end loader onto the 72" conveyor.

Silos are of slip form concrete construction, each with five 700 TPH mechanical vibrating feeders loading onto the 72" conveyor underneath. A variable speed drive is provided on the conveyor so that coal can be loaded into rail cars, via the 200 ton steel loadout bin, at a rate to match the rail car loading operation.

A unit train will be pushed by its locomotives under the loadout bin at a constant speed. A retractable chute on the bottom of the bin drops into each car as it passes underneath, loading the required amount of coal. Total load in each car is checked by a rail track weigh scale located adjacent to the bin. Manual operation of the load-out system has been chosen at the present time as fully automatic operation has proved to be difficult to achieve. An accuracy of 2% is required on the load in an individual car and 1% on the overall train load; 10,000 ton unit train loading will be completed in approximately 3 1/2 hours.

The system described above has been chosen from several alternatives on the basis of best knowledge available at this time. There is the possibility that the general method of loadout or some details could change as engineering progresses and further information comes to hand.

5.4 Quality and Blendability of the Coal Seams

The quality and characteristics of the North and South Hill coal seams are similar to the present producing Western Canadian coal mines.

The work carried out in the 1974 Summer program has shown that the raw ash content and the coking propensity has not changed throughout the South and North Hills from the evaluation results in 1972 and 1973.

Coking tests have been carried out on each individual seam and seams #2, #4 lower and #5 have good coking properties, whilst #4 upper seam has a high proportion of inert material, causing this seam to have poorer coking properties. This characteristic in #4 upper seam tends to reduce the coking strength of the other three seams when blended.

Below is a table to show the coking qualities of each seam:

	#2 Seam	#4 Upper Seam	#4 Lower Seam	#5 Seam
Ash	8.0	9.0	9.0	9.0
Volatile Matter	24-28	22-24	25.0-26.0	25-26
F.S.I.	7.0	4 1/2	6.	6 1/2
Reactives/Inerts	70/30	60/40	65/45	68/32
D.D.P.M.	20/60	3/10	5/12	100/300
Stability	50	44	50	52
Hardness	69	64	68	70
Percent Breeze	3.0	4.5	3.5	3.0
Di $\frac{30}{15}$ I.I.S.	92.5	89.8	92.5	93.5

It can be seen from the table that #4 Seam has the poorest coking properties, and tests have shown that when this seam is blended with other seams the quality is reduced almost to the quality of #4 Upper Seam. It may be necessary to modify the blend ratio to arrive at the ultimate product for the market. This matter will be developed to a firm conclusion during projected quality and marketing investigation over the next several months.