82M002 MT COPELAND

004719

24

VOL. I

PROPERTY FILE

PRELIMINARY FEASIBILITY STUDY

COPELAND MOUNTAIN MOLY PROJECT

KING RESOURCES COMPANY. May 1969.

VOLUME I

KING RESOURCES COMPANY
PRELIMINARY FEASIBILITY STUDY
COPELAND MOUNTAIN
MOLYBDENUM PROJECT

REVELSTOKE MINING DIVISION

MAY 1969

TABLE OF CONTENTS

VOLUME II APPENDICES

I REPORT ON "BENEFICIATION OF A MOLYBDENITE ORE BY FROTH FLOTATION" by Colorado School of Mines Research Foundation Inc. March 17th, 1967

II PROGRESS REPORT NO. 2 ON "BENEFICIATION OF A MOLYBDENITE ORE BY FROTH FLOTATION" by Colorado School of Mines Research Foundation Inc. June 13th, 1968

III CONCENTRATOR AND CRUSHING PLANT ESTIMATES by Interior Engineering Services Ltd. September 16th, 1968

VOLUME III EXHIBITS

340-2-12	Section 350W	340-2-18	Section 650W
340-2-13	Section 400W	340-2-19	Section 700W
340-2-14	Section 450W	340-2-20	Section 750W
340-2-15	Section 500W	340-2-21	Section 800W
340-2-16	Section 550W	340-2-22	Section 850W
340-2-17	Section 600W	340-2-23	Section 900W

340-2-1	PLAN 6660 LEVEL
340-2-24	SURFACE PLAN
1223	BENCH STOPING SYSTEM
	FLOW SHEET STUDY NO. 1, Molybdenite Flotation Plant
	FLOW SHEET STUDY NO. 2, " " "

CERTIFICATE

I, Carl R.D. Miller of Vancouver, British Columbia do hereby certify:

1. That I am a geologist residing at 2732 Oliver Crescent, Vancouver, British Columbia.

2. That I am a registered Professional Engineer in the Province of British Columbia.

3. That I am employed by Chapman, Wood & Griswold Ltd., Consulting Mining Engineers and Geologists, 133 East 14th Street, North Vancouver, B.C.

4. That I have practised my profession for over 20 years.

5. That I have no direct, indirect or contingent interests in the King Resources Company or any of the properties controlled by this Company.

6. That I have studied the existing reports on this property and have reviewed the information with Messrs. Collins and Gallant but that I have not personally visited this property.

PROFESSIONAL PROVINCE C.R.D. MILLER BRITISH COLUMBIA ENGINEER

Carl R.D. Miller

May 6th, 1969

CERTIFICATION

I, E.S. Holt of North Vancouver, British Columbia, do hereby certify:

1. That I am a geologist, residing at 316-145 West Keith Road, North Vancouver, B.C.

2. That I am employed by Chapman, Wood & Griswold Ltd., Consulting Mining Engineers and Geologists, 133 East 14th Street, North Vancouver, B.C.

3. That I have practised my profession for ten years.

4. That I have no direct or indirect interest in the King Resources Company or any of the properties controlled by that Company.

5. That I have studied the existing reports and data of the Copeland Mountain molybdenum project and have reviewed the information with Messrs. Collins and Gallant, but that I have not personally visited the property.

E.S. Holt

E.S. Holt

May 1969

CERTIFICATION

I, John A. Wood, of West Vancouver, B.C., do hereby certify:

1. That I am a geologist, residing at 5019 Howe Sound Lane, West Vancouver, B.C.

2. That I am Vice-President of Chapman, Wood & Griswold Ltd., Consulting Mining Engineers and Geologists, 133 East 14th Street, North Vancouver, B.C.

3. That I have practised my profession for more than 30 years.

4. That I have no direct or indirect interest nor do I expect to receive any such interest from King Resources Company.

5. That I have personal knowledge of the Copeland Ridge property being explored and developed by King Resources Company.

John A. Wood

John A. Wood

May 6th, 1969

I

INTRODUCTION

At the request of King Resources Company, Chapman, Wood & Griswold Ltd., in October 1968, completed a preliminary feasibility study on the Copeland Ridge molybdenum deposit. At the request of Mr. B. Gallant, we have revised that report to include pertinent information obtained since that date.

The program in progress at the time of that report consisted of driving an adit in a northerly direction from the south side of Copeland Ridge to below the partially delineated mineralized body on the shear north side of the ridge (known as the Glacier Zone).

The purpose of this program was to:

1. Explore the geologically favourable area along strike and below the known mineralized area, and to furnish a base for exploring the broad east west zone on the north side of Copeland Ridge which is known to contain other molybdenum showings, and a partially delineated deposit estimated to contain several million tons at an approximate grade of 12 percent combined lead and zinc and about one ounce of silver per ton.

2. To provide access to the Glacier molybdenum Zone.

Our 1968 report concluded that the chances for encountering additional molybdenum mineralization and the exploration possibilities have sufficient value to justify a decision to put the property into production at a nominal rate of 200 tons per day.

Development of the property has continued throughout the winter. Geological sections have been revised to include diamond drill results not available in October and further information regarding capital and operating costs is now available.

This feasibility report re-examines the potential dollar return from exploitation of the estimated reserves taking into account only direct operating costs in relation to that portion of total capital outlay directly attributable to the development, mining and treatment of the Glacier Zone small molybdenum deposit.

II

TERMS OF REFERENCE AND CONTROLS

The present feasibility study of the Copeland Ridge molybdenum deposit is based on analysis of data furnished by King Resources Company as of May 1st, 1969.

The project is presently at the stage of ore definition by exploration diamond drilling, underground exploration and sampling, bulk sampling and bench scale metallurgical testing.

Preliminary estimates of volume and grade of mineralization, capital costs and operating costs have been prepared and are presented summarily and in detail in this report.

The following members of the staff of Chapman, Wood & Griswold Ltd., have made visits to the property prior to October 1968 and our report is in part based on their observations: E.P. Chapman, Jr., G.M. Hurd, David Scott, A Simon Malone and Herbert J. Toohey.

A. DATA UTILIZED IN THE FEASIBILITY STUDY

1. The Colorado School of Mines Research Foundation, Inc., at Golden, Colorado, has performed two series of bench scale metallurgical tests on bulk samples submitted to them by King Resources Company. We have reviewed and accepted the CSMRF results and conclusions regarding the amenability of the deposit to treatment to produce a molybdenum concentrate.

2. Interior Engineering Services, Ltd. of Kelowna, British Columbia, has prepared a preliminary report encompassing cost estimates of various sizes of crushing and concentrating installations, plant site locations, water supply and tailings disposal sites. We have reviewed the cost estimates contained in the report and have accepted them for use in our estimate of capital and operating costs. Property survey control was under the supervision of Interior Engineering Services, Ltd.

3. Crest Laboratories Ltd. of Edmonton, Alberta, have performed the chemical analyses of samples from the surface and underground exploration program directed by King Resources Company. We have accepted the results as being sufficiently accurate and have based our grade estimate upon these results.

4. Loring Laboratories Ltd. of Calgary, Alberta, performed chemical analyses of twenty-six check samples of duplicate pulps prepared by Crest. We consider the results to be sufficiently accurate and to compare favorably with those of Crest Laboratories Ltd.

5. The exploration programs at Copeland Mountain are now under the supervision of Mr. C. Collins, resident manager for King Resources. M.C. Robinson, Ph.D., P.Geol, P.Eng., and G.A. Wilson, P.Eng., consultants to King Resources, have directed most of the geological and sampling programs. Much of the data furnished by these gentlemen has been used in preparing this study.

6. The exploration program was under the overall direction of Mr. B.T. Gallant, Director of Mining Exploration, Canadian Division of King Resources Company. Mr. Gallant provided liaison between the mine staff and staff members of Chapman, Wood & Griswold Ltd. and expedited the transfer of data from the field.

B. ASSUMPTIONS AND CONTROLS

The following basic governing controls have been applied to evaluation of the Copeland Mountain project:

1. Mine life - 2.57 years, say 3 years
2. Milling rate - 200 tons per day, operating 350 days per year
3. Preproduction expense - $331,700
4. Working capital at startup - $285,600
5. Effective May 5th molybdenum metal price increased from \$1.62 to \$1.72 US per pound contained Mo in MoS_2. We have assumed a net realized price of \$1.83 Canadian per pound contained molybdenum in concentrates f.o.b. mine for purpose of a preliminary Cash Flow.

SUMMARY, CONCLUSIONS AND RECOMMENDATIONS

1. A partially developed lenticular body, in the "Glacier Zone" on the north side of Copeland Ridge, is estimated to contain approximately 127,600 tons averaging 1.85% MoS_2 in combined proven and probable reserves after allowances for mining dilution of 10% at 0.20% MoS_2 and 95% extraction. Possible reserves are estimated at 52,600 tons averaging 2.83% MoS_2, after applying the same factors of dilution and extraction.

2. The main access crosscut (6150 level) has been driven northerly from an adit on the south side of Copeland Ridge from which a vertical raise (timbered) has been completed to the 6660 level elevation. A connection to the upper north side workings will be completed in the immediate future. An inclined ore pass raise, also being driven to the 6660 level, is now 165 feet above the 6150 level and should be completed by June 1st, 1969.

3. The estimated capital required to put the property into production, including the cost of the ore pass raise, the installation of mining, milling and additional camp facilities and with provision for working capital, is approximately $2.4 million. This figure does not include expenditure on the property to date including the 6150 adit and access raise.

4. It is our opinion that, at a production rate of 200 tons per day, the presently known mineralized body would sustain an operating life of 2.6 years, and would generate a net cash flow of approximately $4,795,000 assuming no federal income tax is applicable.

5. Although the indicated profit of $2,379,500 (excluding exploration costs to date) would not be considered sufficiently attractive under normal mining investment standards, it does provide a means of expanding exploration and development to further test the potential for additional molybdenum mineralization. In particular, possible depth extensions of the Glacier Zone can be readily explored from various locations made available from the access raise.

6. We consider the chances for encountering additional molybdenum mineralization and the exploration possibilities to have sufficient merit to justify a decision to put the property into production at a nominal rate of 200 tons per day.

Respectfully submitted,
CHAPMAN, WOOD & GRISWOLD LTD

Carl R. Miller

C. Miller, P. Eng

E. Holt

E. Holt

John A. Wood

John A. Wood, Vice President

Distribution: B. Gallant (10)
E.P. Chapman, Jr
John A. Wood
C. Miller
E. Holt
File

May 6th, 1969

IV

GENERAL

A. LOCATION

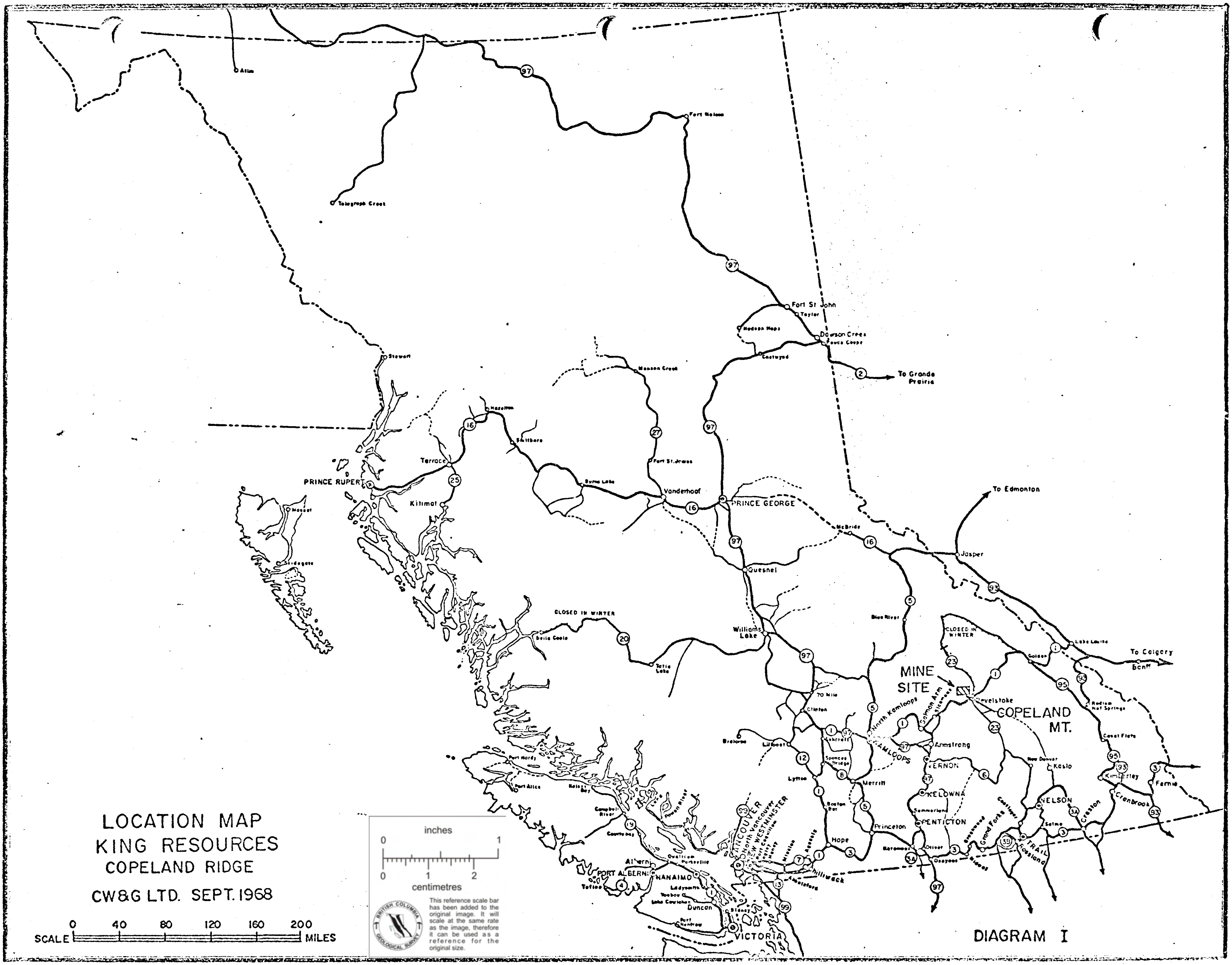
The proposed mine site on Copeland Ridge is located at approximately 51°8' North Latitude and 118°27' West Longitude, about 14 air miles north-west of Revelstoke in the Province of British Columbia. Revelstoke is some 250 air miles distant from Vancouver and some 430 miles by road and rail. Vancouver is the nearest sea-port. Calgary is 190 air miles distant.

The deposit is situated on the north side of Copeland Ridge at an elevation of some 6,500 feet above sea level. Access to the proposed mine is by means of an adit some 6,000 feet long, the entrance to which is on the southern slope of Copeland Ridge at an altitude of some 6,100 feet a.s.l. The mine area is referred to as the "North Side", while the access facilities are termed the "South Side" (See Diagrams 1 and 2).

B. ACCESS

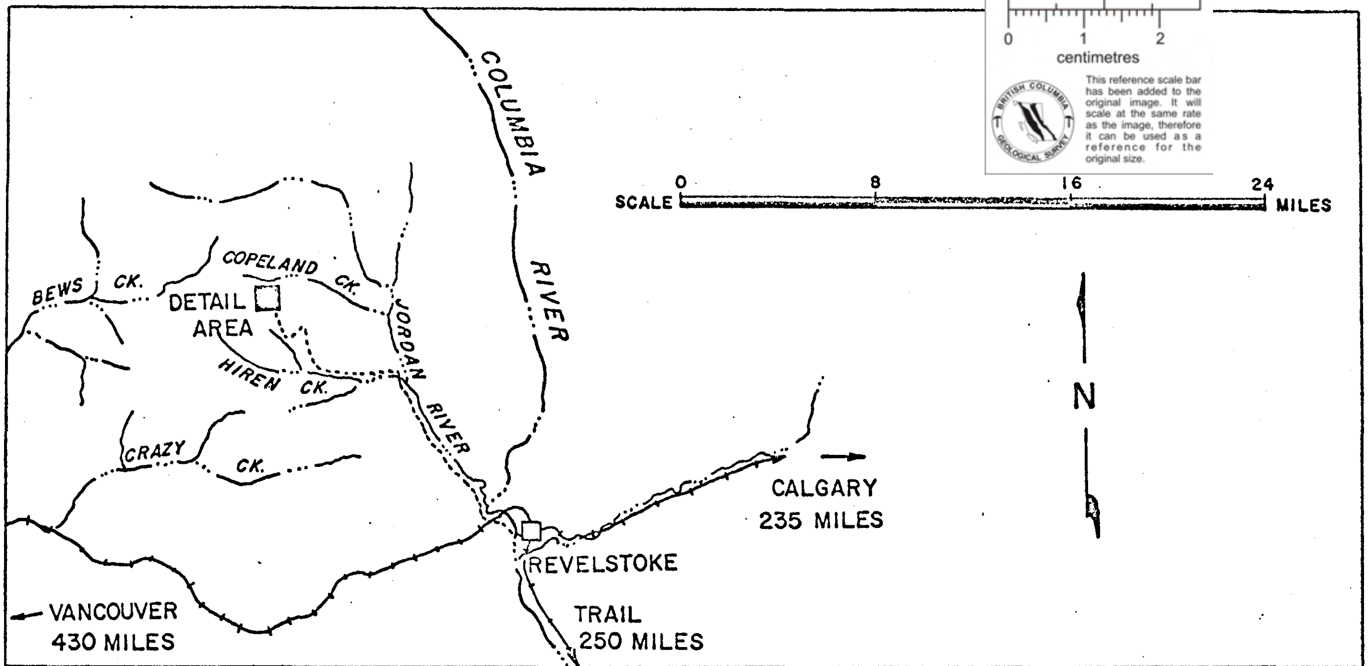
Proposed sites for mill and camp are on the south side of the ridge. Access to this side is by means of a "cat" road, well graded and in good condition, but which will need to be considerably improved once the mine gets into production. This road joins a logging road some 9.7 miles from the mine, and from this junction it is a further 9.2 miles to the Trans Canada Highway.

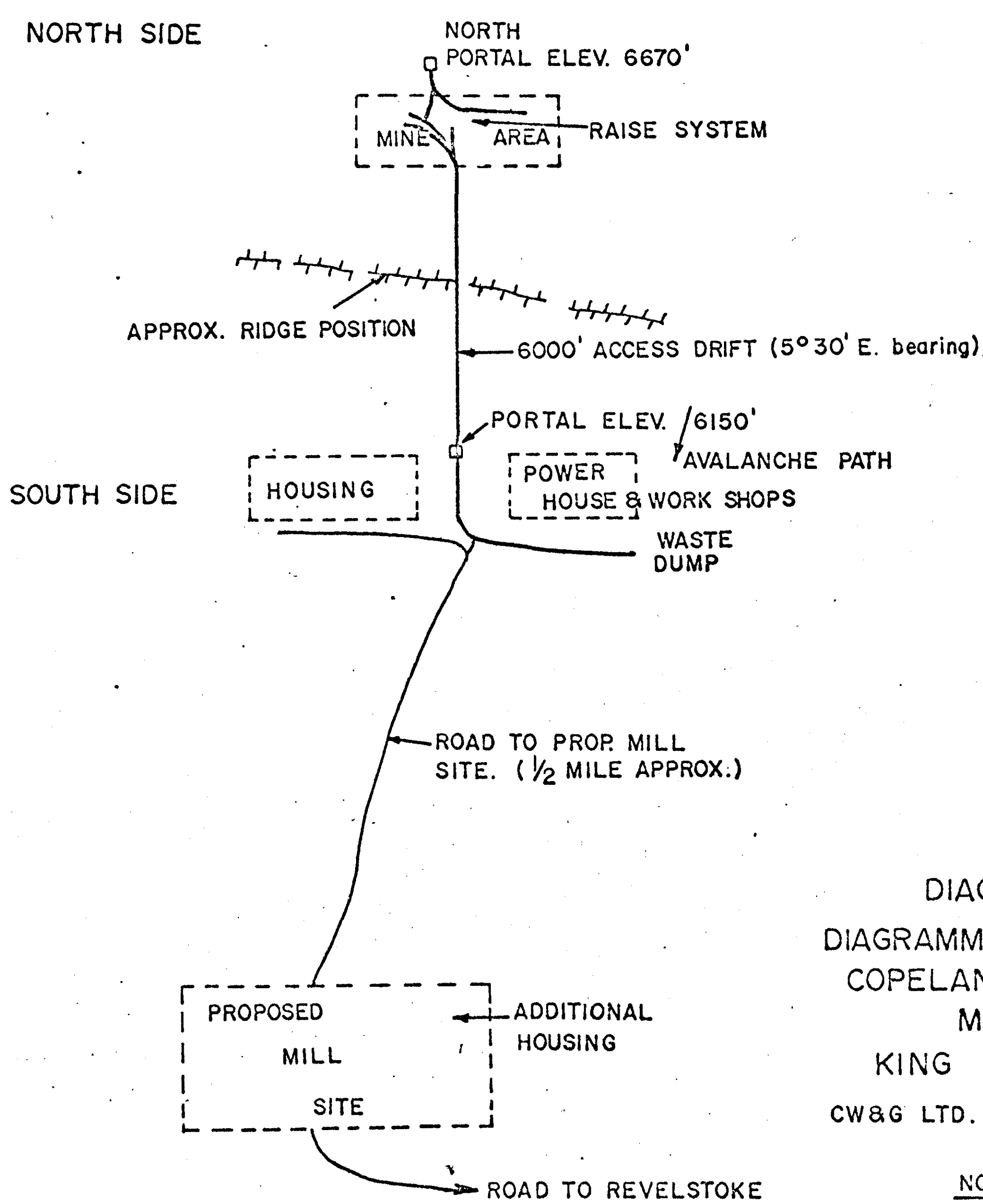
The north side is accessible only by helicopter or via the 6150 level and access raise now nearing completion to the 6660 level. During the winter months the mine road was maintained providing virtually continuous access. Where possible the road course has been routed to avoid potential slide areas, and unless conditions more severe than those of the past winter are encountered no snow sheds are planned.



LOCATION MAP
KING RESOURCES
COPELAND RIDGE
CW&G LTD. SEPT. 1968

DIAGRAM I





DIAG. 2

DIAGRAMMATIC LAYOUT
COPELAND RIDGE
MINE
KING RESOURCES

CW&G LTD. DRWG.NO. 1198

NOT TO SCALE

C. CLIMATE, TOPOGRAPHY AND VEGETATION

The general area of the mine including both the North and South Side camps is in the Gold Range of the Monashee Mountains which in this area is an east-west trending succession of high ridges at upwards of 8,400 feet elevation and steep sided youthful stream valleys. The valley floors are narrow with their long profiles having descents as steep as 3,000 feet in 6 miles.

The north facing mountain slopes are steep and rugged; and in many places depressions are occupied by glaciers above 6,500 feet elevation. The south slopes are more gentle and covered by alpine meadows. The tree line is at 7,300 feet elevation with a more dense growth below 6,500 feet. On the north slopes the tree line is slightly lower.

Average precipitation in the area is 42 inches per annum, the major portion being snow during the winter spring and fall seasons. Snowfall in the mine area itself was recorded at over 52 feet during the 1968-69 season to date.

D. WATER RIGHTS

Applications have been made for water rights totalling 500,000 gallons per day each from the Hiren Creek and Buse Lake systems. Verbal agreement in principle has been reached with formal agreements now pending.

E. CLAIMS

As at 2nd August, 1968, King Resources held 1244 staked mineral claims in the Copeland Mt. area.

The Walt, Knox, Ken and Hap Groups, cover most of the proposed mine area and possible extensions.

F. EXISTING FACILITIES

Two prefabricated portable camps were installed during the winter of 1968. The South Side camp has facilities for 40 men, the North Side for 14. The latter, being of a temporary nature, will be dismantled as soon as considered practical.

The South Side camp facilities include a dry, a small store, and some office accomodation, all of which are covered by a snow shed. A second snow shed covers the power house and workshop area including two sixty Kilowatt generators, two 750 H.P. Sullivan compressors, storage capacity for roughly 22,000 gallons of fuel, a battery charging unit, a small store and a mechanical workshop. A third snow shed shelters 8 trailers providing office and living-recreation facilities.

The equipment in use in the Adit drive includes: Two Eimco 24B mucking machines, Two $4\frac{1}{2}$ Ton Battery Locomotives, 12 mine cars and the ancillary equipment necessary for operating on 24" Gauge rail. This equipment, together with a great deal of smaller equipment, belongs to King Resources Company. Most of the remaining equipment is on a rental purchase basis and could be purchased if required.

The existing camp facilities will require expanding to accomodate an operating force of approximately 50 men.

G. HISTORY

During 1964 several claims were staked on the North Side of Copeland Ridge in the vicinity of the existing Camp. These claims were purchased by King Resources from Gulliver Mining and Exploration in January of 1965. Additional staking in the area of the existing workings was accomplished during the summer of 1965, and the surface geology was mapped by George A. Wilson.

M.C. Robinson, during 1966, further mapped and sampled the zone known as the "Peripheral Zone". In late 1966 the "Glacier Zone" was discovered, channel samples taken, and the zone was mapped. A bulk sample was shipped to the Colorado School of Mines Research Foundation for analysis and preliminary metallurgical testing.

The short field season did not allow time for further work on the property and exploration was suspended until 1967. Six diamond drill holes were put down in Mid 1967, the results of which led to a decision by King Resources to start an underground operation on the North Side only. Thus in 1967 about 700 feet of cross cutting and drifting on the North were developed and sampled. Winter conditions forced the operation's closure in mid-November of 1967. In the same period, due to some doubt about various claim boundaries, the claims were abandoned and relocated by Interior Engineering, who also provided geodetic and astronomic primary survey control of the area.

Since accessibility to the North Side was difficult at best, and of limited duration due to weather conditions, King Resources decided that the only manner in which a sustained exploration and development effort could be accomplished would be with access from the South Side by means of a 6,000 foot Adit cross-cut and a 600 foot raise. Work on the cross-cut commenced early in 1968. The present South Side camp, erected in February 1968, serves as a base. Work towards access to the peripheral molybdenum zone is continuing from the portal on the South Side.

The following consultants and contractors have prepared various reports and/or worked on certain aspects of the property.

1.	George A. Wilson	- Geology (1965 - Present)
2.	M.C. Robinson	- Consulting Geologist (1966 - Present)
3.	Interior Engineering Services Ltd	- Surveying, Road, Power, and Water Consultants (1966 - Present)

4.	Colorado School of Mines Research Foundation Inc.	- Ore Beneficiation (1966 - Present)
5.	Versatile Engineering	- General Contrators (1966 - Present)
6.	Rupert Drilling	- Underground Drilling Contractors (1967 - Present)
7.	Chapman, Wood & Griswold Ltd	- General advisory (October 1967 - Present)
8.	E.H.Bronson	- Mill Design (1967 - Present)

V

GEOLOGY

In describing the geology of the Copeland Ridge area of British Columbia we have in part drawn upon a report describing rock types, mineralization and structural geology written by George A. Wilson, dated November 28, 1965, completed under the auspices of King Resources Company. Additional structural data has been obtained from preliminary mapping by J. T. Fyles of the British Columbia Department of Mines (1965-66). Various staff members of Chapman, Wood & Griswold Ltd. have made visits to the property, examined underground exposures and diamond drill core, and have held periodic discussions with the mining staff of King Resources Company and their geologic consultant.

The geologic environment of Copeland Ridge is that of a syenite intrusive (?) generally occupying a topographically high area and surrounded by a complexly folded series of granitoid and metamorphic rocks of the Shuswap Terrane of the Monashee Group. The principal molybdenite-bearing unit is a lime silicate rock occurring as a generally persistent zone at the surface along the contact of the syenite core and brown-weathering gray gneiss containing a high percentage of calc-silicate and dolomitic beds. Outward from this central zone the metamorphic rocks vary in texture, mineralogy and apparent degree of metamorphism. Only minor amounts of molybdenum mineralization are reported to occur in the gneiss.

The Shuswap rocks are highly deformed and display at least two recognized phases of folding. The older phase is characterized by isoclinal recumbent folds with axial planes ranging from the horizontal to a dip of 50 degrees to the southwest. A second and later phase is that of more open folding that generally plunges to the southeast with northeast axes. Mineralization is contained in the later phase folds.

The major fault in the area is an east to west trending break generally expressed by the valley of Copeland Creek to the north of the ridge. Displacement is predominantly vertical with upward movement on the south side of several thousands of feet. At least four faults of intermediate magnitude on the south slope of Copeland Ridge trend north-northwest through the metamorphic rocks and possibly into the syenite core. Recent field investigations by the King Resources staff indicate that folding with a strong fracture expression may account for the apparent displacement. The relation of these structures is not yet clear; however, it has been suggested that the fractures are pre-mineral breaks that controlled mineralization or provided channels of ingress and were subsequently healed and then folded.

Within the Copeland Ridge area is the aforementioned molybdenite bearing calc-silicate zone surrounding the core which is locally referred to as the Peripheral Zone. To the north of the Peripheral Zone, some 500 feet, is a second zone of much smaller lateral extent known as the Glacier Zone and principally composed of molybdenum bearing aplite and pegmatite.

A. THE PERIPHERAL ZONE

Molybdenum mineralization of varying grades occurs within the lime-silicate Peripheral Zone over a total distance of 40,000 feet and to an undetermined depth. The distribution of molybdenum within the zone, as indicated by surface trenching and drill hole sampling, appears to be erratic and no potentially economic concentration has as yet been found.

Much of the zone is covered by permanent ice and snow fields and it has not as yet been sampled consistently over its entire outcrop.

B. THE GLACIER ZONE

The most significant molybdenum occurrence is that of the Glacier Zone to the north which has been observed and sampled at the surface by trenching and short inclined diamond drill holes and underground by channel samples in an adit crosscut, drifts and by diamond drilling. The Glacier Zone has been mapped as syenite, pegmatite and aplite occurring near or at the contact between nepheline syenite and mica schist.

In the underground exposures the zone has been observed to be a complex series of recumbent folds composed of an interfingered and infolded series of pegmatite and aplite units. In general, the zone forms a complimentary down-fold and adjacent up-fold with their axes inclined to the northeast approximately 30 degrees and resulting in a dip of 30 degrees to 50 degrees on the flanks of the folds. Along strike the zone has a known extent of approximately 500 feet in a northwest-southeast direction with somewhat less than this distance exposed at the outcrop some 200 feet up dip. The complementary folds plunge to the southeast and the plunge ranges from 40 degrees on the northeast to 20 degrees on the southeast and is conformable with the second phase folding. Limitation of the mineralized zone has been fairly well established at the northeast end by a drift and surface and underground diamond drilling. Here the zone is inclined upward at an angle of approximately 40 degrees and is seen in outcrop as thinning and fingering into the surrounding syenite. Limitation and configuration of the zone at the southeast end is not yet clearly understood; however, present evidence from the southeast drift and several diamond drill holes recently completed would suggest the zone in both the up-fold and down-fold portions is considerably less in thickness, lower in grade of mineralization and interfingered with the enclosing syenite. Considerably shattered and broken ground suggesting the possibility of faulting has been encountered at the end of the southeast drift. Because of the necessity of providing support by timbering to continue the southeast advance, it was decided to suspend the drifting and to diamond drill ahead in an attempt to gain geologic and grade information before proceeding. A drill hole was attempted, and although not successful in penetrating the bad ground, it did confirm that a major zone of shattered, broken and sheared rock of considerable extent does exist in this area.

Molybdenum mineralization in the Glacier Zone occurs as disseminated plates, clusters of plates, blebs (irregular but discrete particles with no apparent crystal form) and massive blebs tending to be elongate and parallel with fractures and foliation. The coarser particles are frequently surrounded by finer disseminated particles and flakes of molybdenite. Where accompanied by pyrite and pyrrhotite the molybdenite appears to cut across the accessory sulphides implying a later development of the molybdenite; however, no known investigation has been made of mineral paragenesis and this relationship is only implied.

The sulphides are contained in biotite feldspar syenite with bands of aplite and in pegmatite syenite and aplite syenite. The latter two may have been derived from the biotite feldspar syenite by replacement of the mica. Calc-silicate and aplite bands and lenses conformable with local folding occur frequently in the zone. Molybdenite was observed in the underground exposures to occur more frequently in visually greater amounts in a light green (possibly chloritic?) aplite.

Alteration minerals in the zone, and frequently associated with sulphide mineralization, are epidote, chlorite, calcite and pink and gray feldspars.

Several spectrographic analyses of samples from the Glacier Zone indicate no other metallic minerals of economic grade to be present.

Geologic conditions in the Glacier Zone have been observed in over 800 feet of drifting. The crosscut adit and raise from the south side to provide year round access to the upper workings has now been completed. It begins at a point near the downward projection of the south limb of the peripheal zone and would provide ready access for diamond drill stations to explore the extensions of the peripheral zone as well as the syenite core.

VI

RESERVES

The molybdenum-bearing reserves of the Copeland Ridge deposit have been estimated from information revealed by surface exposures, underground workings and diamond drilling. The reserves were calculated from blocks constructed on drill hole cross sections.

A. DEFINITION

Reserves have been divided into three categories based on their relation to assay data points and geologic information.

SEMI-PROVEN - Mineralization defined by diamond drilling and/or channel sampling, the block limits of which are not projected more than 25 feet up or down dip from assay data points, nor more than 25 feet horizontally along strike either side of a drill ring or channel sample section.

PROBABLE - Mineralization in blocks projected beyond or between SEMI-PROVEN reserve blocks, and which lie not more than 50 feet up or down dip from the assay data points, nor more than 25 feet horizontally along strike either side of a drill ring or channel sample section.

POSSIBLE - Mineralization in blocks projected beyond or between PROBABLE reserve blocks, the limits of which are not more than 100 feet up or down dip from assay data points, nor more than 50 feet horizontally along strike from drill ring or channel sample sections. These reserve blocks all lie within, and their shape is influenced by, the projections of the geologically favorable horizon.

The reserves have been further classified as GEOLOGIC RESERVES and MINEABLE RESERVES.

GEOLOGIC RESERVES	-	Reserves in place before applying factors of mining dilution and mining extraction.
MINEABLE RESERVES	-	Reserves available for milling after applying to Geologic Reserves the factors of mining dilution and mining extraction.

An overall mining dilution of 10% has been used for calculation of the mineable reserves. The somewhat erratic configuration of the stope blocks, the irregular pattern of the mineralization and the fact that some bad ground has been encountered in the mine area, indicate that 10% dilution will likely occur.

A dilution grade of 0.20% MoS_2 has been established by averaging the grade of all intersections immediately adjacent to the reserve blocks.

The mining method planned should achieve a high degree of extraction, hence an extraction factor of 95% has been used.

A density factor of 12 cubic feet per ton has been used for calculating the Copeland Ridge reserves. This conversion factor is based on several specific gravity determinations.

B. SAMPLING

Surface and underground channel samples and diamond core samples were taken under the supervision of the King Resources Company mine staff. We consider that the method and procedure is acceptable and that the sampling was done in accordance with good engineering practice.

The procedure of underground sampling in the mineralized zone was as follows:

1. Continuous vertical sidewall and horizontal back channel samples at 5-foot intervals along the drift.

2. Continuous horizontal channel samples at 5-foot intervals along both walls of the drift and at three levels along the wall, each approximately two feet apart.

3. Three horizontal channel samples on the face at the completion of each round.

4. One car sample for each round loaded.

Horizontal channel samples, horizontal chip samples and vertical channel sample weighted assays are compared with weighted core assays from a hole drilled along the same bearing and inclination prior to the advance of the west drift extension. Distance is 55 feet.

	Vertical Channel	Horizontal Chip	Continuous Horizontal Channel	DDH Core
Percent MoS_2	0.85	0.93	1.25	1.04

When due consideration is given to the nature of vertical channel and horizontal chip samples there is a favorable comparison between horizontal channel assays and diamond drill core sample assays. The drill core was split by a diamond saw and one half sent for assay and one half retained for logging and future reference. Assaying was done by Crest Laboratories Ltd. of Edmonton, Alberta.

C. DRILL HOLE SECTION METHOD

Geologic and assay sections were constructed along northeast-southwest trending lines normal to the trend of the zone at 50-foot intervals. Drill holes and assay data were projected to the sections, from which geologic interpretations were made and reserve blocks outlined.

Mineral grade cutoff was 0.50% MoS_2, except in rare cases where normal mining methods would include the extraction of some lower grade material.

The grade assigned each block was the weighted average of those assays within, or projected to the block.

The area of each block was measured by planimeter and converted to volume by application of the half-distance interval between sections. Blocks in each of the end sections were limited to 25 feet beyond each end section.

D. CHECK SAMPLING

Selected samples of duplicate pulps were submitted to Loring Laboratories Ltd. in Calgary, Alberta, for assay. These samples, a total of 25, were selected so as to represent the largest possible range in grade of MoS_2 and the widest distribution within the zone of mineralization. We have analyzed the results from the comparison check sampling and conclude that they compare favorably with the corresponding original assays.

E. RESERVE SUMMARY

	Tons	% MoS_2
GEOLOGIC RESERVES		
Semi-Proven	90,400	1.98
Probable	31,700	2.14
Total Semi-Proven and Probable	122,100	2.02
Possible	50,400	3.09
MINEABLE RESERVES		
1) Semi-Proven and Probable	122,100	2.02
Add: Dilution (10%)	12,200	0.20
	134,300	1.85
Subtract: Mining Extraction Factor (5%)	6,700	1.85
Total Mineable Semi-Proven and Probable	127,600	1.85
2) Possible	50,400	3.09
Add: Dilution (10%)	5,000	0.20
	55,400	2.83
Subtract: Mining Extraction Factor (5%)	2,800	2.83
Total Mineable Possible Reserves	52,600	2.83

TABLE VI-1

RESERVE SUMMARY BY SECTIONS

Classification	Section Number	Tons	% MoS_2
Semi-Proven	350 W	1,600	0.98
	450 W	800	0.90
	500 W	400	0.80
	550 W	10,400	2.58
	600 W	400	0.91
	650 W	24,300	2.06
	700 W	22,500	2.24
	750 W	4,900	1.14
	800 W	13,700	1.76
	850 W	9,300	1.80
	900 W	2,100	1.00
Total		90,400	1.98
Probable	350 W	1,500	0.98
	450 W	500	0.90
	550 W	3,900	5.05
	650 W	6,200	1.95
	700 W	2,800	2.24
	750 W	3,700	1.23
	800 W	5,400	2.20
	850 W	5,600	1.69
	900 W	2,100	1.00
Total		31,700	2.14
Possible	350 W	1,600	0.98
	400 W	1,300	0.94
	450 W	900	0.90
	500 W	12,300	4.96
	550 W	3,100	4.71
	600 W	15,500	2.44
	650 W	2,100	2.34
	750 W	5,600	2.35
	800 W	2,900	3.43
	850 W	2,600	3.12
	900 W	2,500	1.00
Total		50,400	3.09

TABLE VI-2

DETAIL RESERVE CALCULATIONS

Section No.	Classification	Block No.	Tons	% MoS_2
350 W	Semi-Proven	1	1,600	0.98
	Probable	2	800	0.98
		3	700	0.98
			1,500	0.98
	Possible	4	1,600	0.98
400 W	Possible	1	1,300	0.94
450 W	Semi-Proven	1	800	0.90
	Probable	2	400	0.90
		3	100	0.90
			500	0.90
	Possible	4	900	0.90
500 W	Semi-Proven	1	400	0.80
	Possible	2	7,900	6.17
		3	4,400	2.83
			12,300	4.96
550 W	Semi-Proven	1	2,300	6.17
		2	2,800	3.20
		3	2,600	0.77
		4	2,700	0.64
			10,400	2.58
	Probable	5	1,200	6.17
		6	1,200	6.17
		7	1,500	3.20
			3,900	5.05
	Possible	8	1,200	6.17
		9	1,900	3.79
			3,100	4.71

cont'd/

TABLE VI-2
DETAIL RESERVE CALCULATIONS
Cont'd

Section No.	Classification	Block No.	Tons	% MoS_2
600 W	Semi-Proven	1	400	0.91
	Possible	2	5,800	3.40
		3	7,900	2.11
		4	1,800	0.79
			15,500	2.44
650 W	Semi-Proven	1	2,200	0.89
		2	4,300	3.19
		3	3,100	1.55
		4	2,800	5.13
		5	1,300	1.14
		6	6,500	1.09
		7	2,300	1.11
		8	1,800	2.21
			24,300	2.06
	Probable	9	1,100	0.89
		10	1,300	0.89
		11	2,300	3.19
		12	1,200	1.61
		13	300	2.21
			6,200	1.95
	Possible	14	300	0.89
		15	1,800	2.58
			2,100	2.34
700 W	Semi-Proven	1	1,300	4.56
		2	1,900	1.41
		3	4,400	1.46
		4	5,000	4.30
		5	4,500	1.69
		6	3,200	0.96
		7	2,200	1.47
			22,500	2.24
	Probable	9	1,200	1.45
		10	1,100	3.05
		11	300	3.00
		12	200	1.32
			2,800	2.24

cont'd/

TABLE VI-2
DETAIL RESERVE CALCULATIONS
Cont'd

Section No.	Classification	Block No.	Tons	% MoS_2
750 W	Semi-Proven	2	2,000	1.17
		3	1,600	1.31
		4	600	0.33
		5	700	1.34
			4,900	1.14
	Probable	8	1,100	1.17
		9	400	1.34
		10	1,100	1.31
		7	1,100	1.17
			3,700	1.23
	Possible	1	1,400	3.31
		6	1,300	3.31
		11	300	2.32
		12	200	1.73
		13	900	1.34
		14	1,500	1.31
			5,600	2.35
800 W	Semi-Proven	1	3,300	3.43
		2	8,000	1.19
		3	2,400	1.40
			13,700	1.76
	Probable	4	1,500	3.43
		5	1,600	2.31
		6	1,500	1.33
		7	800	1.33
			5,400	2.20
	Possible	8	2,900	3.43

cont'd/

TABLE VI-2
DETAIL RESERVE CALCULATIONS
Cont'd

Section No.	Classification	Block No.	Tons	% MoS_2
850 W	Semi-Proven	1	3,100	0.21
		2	1,700	1.51
		3	400	0.74
		4	2,500	4.89
		9	1,600	0.63
			9,300	1.80
	Probable	5	2,100	0.43
		6	1,100	0.21
		7	1,000	1.51
		8	1,400	4.89
			5,600	1.69
	Possible	10	1,100	0.70
		11	1,500	4.89
			2,600	3.12
900 W	Semi-Proven	1	1,100	0.97
		2	1,000	1.03
			2,100	1.00
	Probable	3	1,100	0.97
		4	1,000	1.03
			2,100	1.00
	Possible	5	1,400	1.03
		6	1,100	0.97
			2,500	1.00

VII

MINING PLAN

A. INTRODUCTION

The present configuration and limited reserves of the Glacier Zone dictate a rather complicated and somewhat expensive mining system. In essence, three methods are proposed, all employing an off-track transportation means for delivering the ore through a central ore pass to the 6150 level. The ore is trammed on conventional track haulage to a transfer point at the portal from whence it is trucked to the mill.

This design is based on the configuration described in Section V. It is stressed that this is a preliminary design and that considerable detail drilling will be required before a final design is adopted.

B. GENERAL LAYOUT

Three main levels are envisaged for access and transportation. The present exploration level on the north side, termed 6660 level, and two further levels above and below the existing portal termed the 6760 level and the 6610 level. The present south side crosscut, 6150 level, is to connect all three mining levels by means of a two-raise system described in detail in subsection C.

All proposed systems utilize the "Scootcrete" units loaded either by scraper or overhead loaders. These carriers, three in all, one per level, tram along strike before dumping into the ore pass system some 100 feet south of the present southern limit of the orebody.

C. SHAFT SYSTEM

The shaft system now being developed under contract includes two raises; one handling ore only, the second, men, materials and services.

At the present time the service raise has been driven vertically from the 6150 level to the elevation of the 6660 level and a connection will be established in the immediate future. This raise has been timbered throughout in such a manner to facilitate conversion to a shaft.

The ore pass, a 7' x 8' inclined raise, is now 160 feet above the 6150 level and should be completed to the 6660 level by June 1, 1969. Alimak equipment is being used. The hoist chamber at the 6150 level is partially developed and negotiations for a suitable used single drum hoist are in progress.

When completed this system will facilitate the development of the Glacier Zone. At least two additional levels are planned, more can be added as warranted.

D. SHAFT COST

The expenditures required to complete the shaft system including the ore pass are estimated at:

Raise 7' x 8' 580' @ $40/ft	$ 23,200
Shaft, equipment and installation allow $50/ft	30,000
Two interconnecting crosscuts 50 ft @ $50/ft	2,500
Single Drum Hoist (used)	20,000
Ore chute equipment - timber, air cylinders etc	4,000
Hoist Chamber and Hoisting arrangements Slashing for HR, rope ways, foundations etc allowance	25,000
Grizzly Construction	2,000
Head all chutes on 2 levels	7,500
Shaft well 10 feet sinking	500
Stations at 6610 and 6760 levels	5,000
Plus 10% contingencies	12,000
	$131,700

E. VENTILATION

Natural ventilation is to be expected. The use of diesel equipment underground will require additional ventilation. Deadends will require forced air but until the natural draft is determined, and a more accurate evaluation of mining methods completed, no accurate assessment of ventilation requirements is possible. Some heating of the air, probably utilizing waste heat from the generating units must be provided.

Battery locomotives are recommended for use on 6150 level, thus minimizing ventilation requirements.

F. MINING SYSTEMS

As previously mentioned, three systems will be necessary to extract the ore. These methods are applied to their particular configurations as outlined below. It should be stressed that the methods are preliminary only and are based on present knowledge. The design of a complete mining system is not feasible at this time.

1. Bench Stoping System

This applies to the upper levels of the mine in which the zone dips at less than 40 degrees and is 17 to 20 feet thick. The stopes will retreat along strike from the western extremity of the deposit. Stope widths will be 70 feet with 10-foot pillars between stopes. Development will be by means of a raise on the hanging wall from the 6760 level. The raise, 70 to 80 feet long, will connect to an access and ventilation drift on strike. The stope will advance in two lifts of 10 feet each, with hanging wall face some 20 to 30 feet ahead of the footwall face. Major support is not anticipated. The ore will be scraped down-dip into a transfer chute beneath which "Scootcretes" will be loaded. Each blast will produce

$$\frac{80 \times 10 \times 5 \text{ ft}}{12} = 33 \text{ tons} \qquad 330 \text{ Tons}$$

Why in waste? Can develop it inside the O/C

Thus, assuming 70 tons per stope per day, the blasting interval will be four days. Production will be from all three levels but predominantly the 6760 level. Thus it is suggested that three stopes be developed on this level and blasted cyclically. This method is both labor intensive and costly but the alternatives available for a deposit of this size with a dip between 40 and 50 degrees involve extensive waste development, which this operation cannot support.

Two operating conditions are noted:

a) If possible a 10-foot surface pillar should be left to avoid an inflow of water.

b) It might be necessary to install a scatterwall for each blast which, although expensive, will confine the scatter to be expected in open bench stoping of this type.

Primary development requires a crosscut from the shaft system to the footwall on 6760 elevation. This tunnel is then drifted the strike length of the orebody, raises established at 80-foot centres through to 10 feet below surface, and the raises connected along strike leaving a surface pillar of some 10 feet. Poor ground conditinns should be anticipated as the surface is approached (Drawing No. 1216).

2. Sub-Level Stoping System

This is to be applied to the zone between 6760 and 6680 levels, and entails a sub-level through the centre of the zone from which conventional ring drilling will be done, blasting into a slot, retreating from the west along strike and drawing the ore off by means of access drifts and draw points. An underhand system is envisaged; consequently a horizontal pillar below 6760 level is not required. However, similar to the bench stoping method, vertical pillars should be left roughly evary 70 feet. Access raises are proposed between the 6680 level and the sub-level at 70-foot intervals.

Both the access facilities and draw points may be developed from the existing exploration drifts on the 6680 level. If the zone width becomes too large to extract all the ore, it might be necessary to duplicate draw point facilities in the footwall. This might encounter poor ground conditions beneath the glacier.

3. Modified Sub-Level Stoping System

This applies to the folded portion of the orebody sub-levels, between the 6610 and the 6680 levels, on 6645 elevation, spaced at 30-foot horizontal intervals. It will be used for ring drilling and blasting into a slot. Again an underhand retreat system from the western limit is visualized. Loading access will be necessary on the north and south sides of the fold. A crown pillar should be left below 6680 level. (See drawing No. 1216).

4. Shrinkage System

This applies to the southern arms of the folded system. The system is included in this report as section CC' (Ore Reserve calculations) indicates a zone width of some 10 feet as compared to 30 feet in other sections. This could either be mined as a vertical shrinkage stope or as a sub-level stope. Depending on zone thickness, the method will be a compromise between the two. Loading will be possible from the 6680 level by means of crosscuts to the zone.

Present indications are that most ground conditions will support an open stoping method. Design criteria are based on this supposition.

G. WASTE DISPOSAL

Construction of a waste pass system to the 6150 level is impractical. ?Why?

Two alternatives are offered:

1. Store waste or defer waste development until summer months when waste may be disposed of on the north side.

2. Develop an internal waste system serving the lowest level and develop a waste drift breaking through on the north side. It is submitted that vertical cliff faces or snow-protected areas exist where waste dumping throughout the year is possible. The waste is then dumped into the Copeland Creek valley.

   Removal of waste via the ore pass and 6150 level access drift is considered possible but should be avoided.

H. DEVELOPMENT COSTS

		Feet	Total
6760 Level	Crosscut from shaft	350	
	Haulage drift	400	
	Access drift below surface	200	
	Raises, 3x80 ft	240	1190
6660 Level	Crosscuts to zone 4x40 ft	160	
	Slot raises, 3x60 ft	180	
	Sub-level development	400	
	Access raises to sub-level	70	
	Second haulage drift	400	
	Draw point development	300	1510
6610 Level	Crosscut from shaft	250*	
	Haulage drift	400*	
	Access raises to 6660 level	140*	
	Sub-level development, 4x400'	800	
	Slot raises, 3x50 ft	150	
	Waste raise	100	
	Waste drift to north side	300	2140
			4840

*Primary development

Cost at \$55 per foot	=	\$266,200
Primary Development, 1540 feet	=	\$ 85,000

This represents a system from which production for the greater part of the mine's life could be derived while normal mine development continued.

For costing purposes, \$200,000 is entered as a capital cost. The remaining cost and any additional development costs to be met as a mining cost throughout the mine life. For taxation purposes roughly \$100,000 will be allowed as a depreciable asset.

METALLURGY

The Colorado School of Mines Research Foundation, Inc., conducted two series of metallurgical bench scale tests on samples submitted to them by King Resources Company. The first series, completed in March, 1967, was on a sample weighing some 500 pounds taken from surface exposures of the Glacier Zone. The second series, completed in June, 1968, was on one sample from the Glacier Zone and one sample from the Peripheral Zone.

E. H. Bronson, P.Eng., of Interior Engineering Services Ltd., Kelowna, B.C., was retained by King Resources Company to carry out a preliminary mill design.

Reports on the bench scale tests and on the preliminary mill design are appended, and the following is a summary discussion of results.

Tests were conducted to determine:

1. The optimum sized grind
2. The necessity for regrinding the rougher concentrate
3. Whether dispersants or depressants were required
4. The presence of impurities in the concentrate
5. An optimum pH modifier
6. The tailings effluent water composition
7. Tailings settling requirements.

Test results indicated the following:

a) Finer grind does increase recovery but 65 mesh size is sufficient to give desired concentrate grade if reagents are used in the flotation circuit for control of iron sulphides and gangue.

b) The ore is amenable to production of a high grade, high recovery concentrate using regrinding and dispersants but other tests produce similar results without regrinding. It is the opinion of E. H. Bronson that regrind of the rougher concentrates is desirable and allowance is made for this.

c) The use of dispersants and depressants is necessary to provide an acceptable concentrate grade.

d) Impurities are present, as indicated below, but are at a low enough level to satisfy market requirements.

	Cleaner Concentrate					
Zone	Cu %	Pb %	P %	Sn+As %	Insoluble %	Fe %
Glacier	0.04	0.014	0.003	< 0.01	1.59	0.86
Peripheral	0.026	0.050	0.004	< 0.04	4.54	1.91

A spectrographic analysis revealed the presence of silver in quantities too small to be of significance. Arsenic, bismuth and lead impurities are either absent or insignificant.

e) The use of soda ash as a pH modifier was shown to be superior to lime insofar as a higher concentrate grade is concerned.

f) A tailings water effluent test was conducted using the equivalent reagent quantities. The conclusions drawn were that the pine oil, sodium silicate and sodium carbonate are not added in sufficient quantities to be considered pollutants. The cyanide exists mostly as the radical $M(CN)_x$ and not as the iron $(CN)^-$, it resists decomposition, and is considered stable.

g) Utilization of a tailings thickener has been recommended to minimize groundwater pollution.

The metallurgy of the "Glacier Zone" ore appears to be relatively uncomplicated. Basic operating data are appended.

Apparent optimum grind	75%-80%; -65 mesh
Concentration ratio	25-30 to 1
Moisture content of concentrates	10%-12%
Concentrate production (200 ton mill)	approx. 8 tons per day, 240 tons per month
Percentage recovery	93%
Concentrate grade	90%-92% MoS_2

Reagents used:

Fuel oil	Standard petroleum product
Syntex L	Sulphated monoglyceride of coconut oil, a detergent
Pine oil	Terpineal derivative ($C_{10}H_{18}O$) from pine trees
Seperan MGL	High molecular weight synthetic polymer
Sodium cyanide	Depressant
Sodium silicate	Dispersant
Sodium carbonate	pH control and flotation agent.

Based on E. H. Bronson's design, the power requirements are:

	HP	Hours per day
Crushers	109	8
Concentrator	173	24
Miscellaneous	50	24
Total HP	332	

Water required is approximately 120 gpm.

IX

MILLING

A. MILLSITE

Two possible sites, depending on mill capacity, were considered:

a) At the South Portal, south of the present workshop-powerhouse complex on the waste dump.

b) At the temporary camp site, a relatively flat area safe from avalanche conditions, approximately 0.8 miles south of the south portal.

In view of the geologic potential of the area and limited aerial extent of the former site it is suggested that the mill be situated at the old temporary camp site. Ore will be transferred from mine cars to surge bins of 150 ton capacity at the portal and trucked from the portal to the coarse ore storage bin at the mill site.

B. MILL DESIGN

The mill was designed by E.H. Bronson and, as previously mentioned, this section summarizes the design. Based on present reserves and in accordance with a desire for the rapid generation of funds, a mill size of 200 tons per day is recommended.

Ore is dumped from the truck into a coarse ore bin, the capacity of which will be 200 live tons. Mine run ore is crushed by a primary jaw crusher and secondary cone crusher before feeding into the fine ore bins. The crushers will operate single shift, five days a week. The fine ore bins, covered but not enclosed, representing 640 live ton storage capacity, feed a pebble grinding mill. The -65 mesh material is then concentrated by flotation, the rougher concentrate being reground in closed circuit before cleaning. The cleaned concentrate is thickened before filtering, drying and packing. The preliminary design allows for a scavenger cell and at least four cleaning stages. Filtering, drying and packing will be on a single shift basis.

C. TAILINGS

The tailings will be thickened and led off to an area to the southeast of the mill. Initial estimates give a storage capacity of roughly six years. Reclaimed water will be diverted to the mill supply. Further expenditure on the tailings disposal should be anticipated for future years.

X

SERVICES

A. POWER

Presently the mine has two 60-KW generators on site. When production commences, considerably more power will be required. Tenders have been placed for two 500 KW units equipped with heat exchangers. After these have been installed the 60 KW units will be standby units. Both units will operate during peak capacity periods, i.e. the day shift; however, during slack periods a single unit will be adequate to handle camp and mill requirements. It is suggested that the generating plant be placed at the south portal to be combined in one building with the compressors.

1. Requirements

	Days per week	Hours per day	HP
Mine	5	8	1000
Crushers	5	8	173
Mill	7	24	109
Miscellaneous	7	24	50
Total HP			1332 = KW

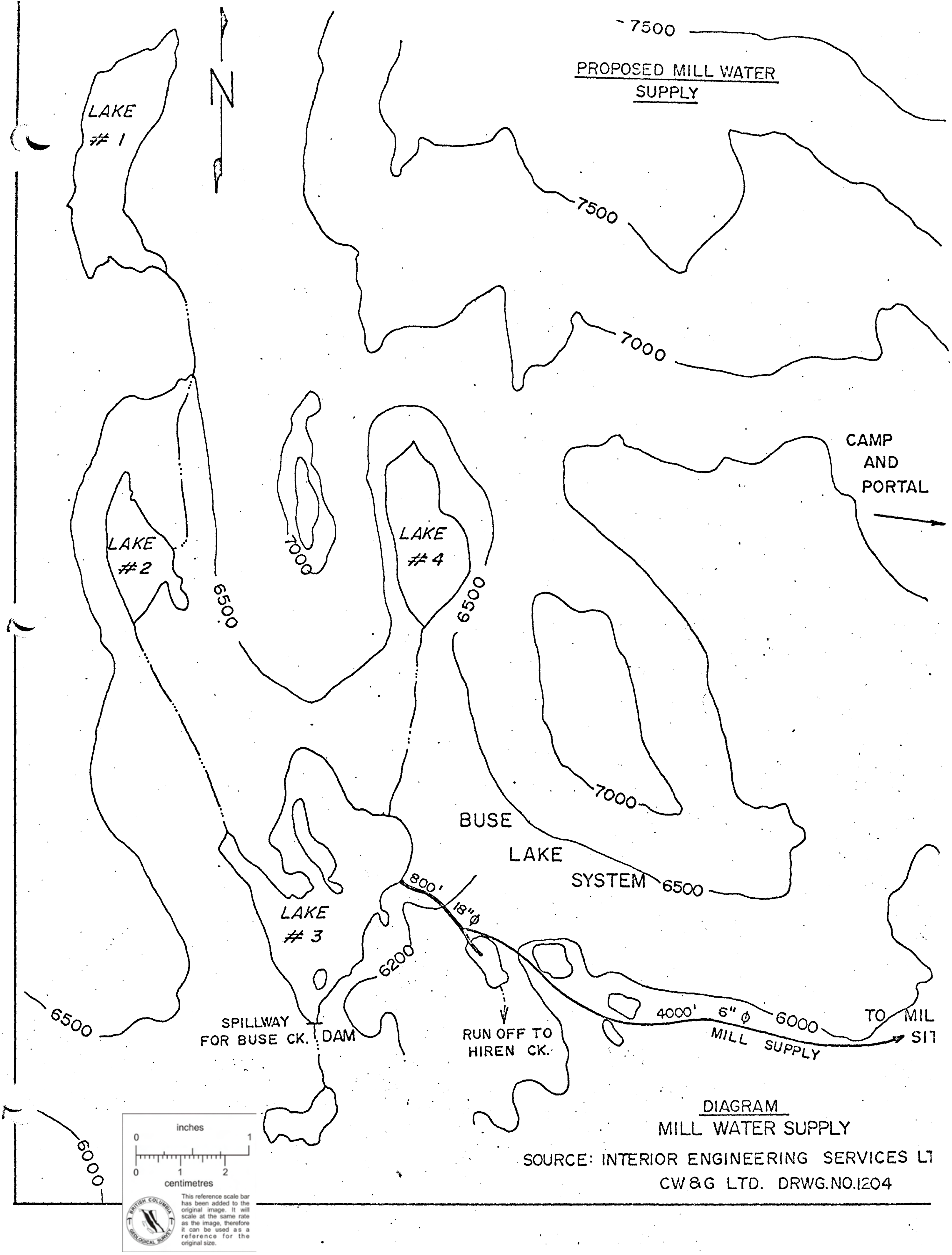
2. Cost

Single 500 KW unit uses	33 gph at .235 cents per gallon
Operating cost	20% over and above fuel cost
Hourly cost, 33 x .235 + 20%	$9.30 per hour
Daily cost $9.30 x 34	$316
Annual cost (350 days)	$110,600 = $1.58/ton

Cost distribution

Mine	46%
Mill	42%
Camp	12%

Consideration was given to erecting power lines leading from either a B.C. Hydro or the Revelstoke local system, but the cost involved was considered excessive. Moreover, the existing Revelstoke plant is presently at full capacity.


PROPOSED MILL WATER SUPPLY
N
7500
7500
7000
7000
7000
6500
6500
6500
6500
6200
6000
6000
LAKE #1
LAKE #2
LAKE #3
LAKE #4
CAMP AND PORTAL
BUSE LAKE SYSTEM
800'
18"ϕ
4000' 6" ϕ
MILL SUPPLY
TO MIL SIT
SPILLWAY FOR BUSE CK.
DAM
RUN OFF TO HIREN CK.
DIAGRAM
MILL WATER SUPPLY
SOURCE: INTERIOR ENGINEERING SERVICES LT
CW&G LTD. DRWG.NO.1204
inches
0
1
0
1
2
centimetres
This reference scale bar has been added to the original image. It will scale at the same rate as the image, therefore it can be used as a reference for the original size.

B. WATER AND COMPRESSED AIR

1. Water Supply

King Resources retained Interior Engineering Services Ltd to study water supply. Mill requirements are in the order of 120 US gpm. The original plan was to obtain this from the Buse Lake drainage system by raising the level of a portion of Buse Lake by a dam, diverting the water via an 800 foot pipe to a small lake and thence pumping the water 4000 feet to the proposed mill site. Mine and camp would be supplied by auxilliary pumping systems.

A more practical alternative is now under consideration. This involves a dam at the small lake and diverting reclaimed mill water and mine ditchwater to this reservoir. It is now estimated that these sources augmented by seasonal runoffs will provide ample supply for all purposes. Camp water supply must be purified.

2. Compressed Air

Two 750 HP Sullivan compressors are presently providing compressed air at 110 psi and 690 cfm per compressors. The mine requirements will be roughly 3700 cfm. Two 1000 cfm electrical units are now on order which will operate in conjunction with the existing compressor facilities.

The requirements are

	cfm
10 Air legs and machines @ 200 cfm/machine, factor 8.7*	1740
3 Overhead loaders, 350 cfm*	1050
25% safety factor	897
Total requirement	3687
Total supply	3750

*Altitude corrected

Operating costs of the compressors themselves are minimal; however, they do involve considerable power output.

Power requirements:

1150 cfm motor	200 HP
750 cfm motor	150 HP
Total requirement	700 HP

Spare compressors are not recommended as it is felt that four units should maintain continuity of supply.

C. ACCESS

As previously mentioned, access is to be via a logging and mine road from the Trans-Canada Highway. From the highway, the first nine miles of logging road will presumably be jointly maintained by the mine and the logging companies concerned. The following 10 miles of mine road, rising some 4100 feet in elevation, should be maintained in good condition throughout the year. During the winter months this will be difficult and the camp should be self-sufficient for periods of at least a week.

A bulldozer and a grader will be utilized for maintaining the road during both the winter and summer months, with the costs involved during the winter months being considerably higher than those of the summer.

Since the access road has been maintained through the past relatively severe winter, the expense of snow shedding portions of the road does not seem warranted; however, a contingency of $100,000 for such purposes has been included in the costing.

D. TRANSPORTATION

This aspect is intimately linked with that of the housing and access. It is felt that a large proportion of the miners would live in Revelstoke if the road were passable the year round. The road is likely to be passable 70 percent of the time but the steep grades etc suggest high maintenance costs therefore traffic should be kept to a minimum. The crew will probably live at the mine during the work week and be transported to Revelstoke or the Highway by company buses for their days off the property, thus housing facilities at the mine are provided for.

The mine will require a bus for transportation of men, a 5-ton truck for transportation of concentrate and supplies, and several general mine vehicles. One 10-ton conventional off-highway truck will be able to move the ore from the surge bins at the portal to the mill site, a distance of roughly 0.8 miles. It is recommended that the ore be moved on company account rather than on contract.

E. HOUSING AND LABOR

As well as during the week, residency at the mine camp should be provisioned for longer periods during the winter months. Accomodation for 50 people has been included.

Senior mine staff will be accommodated in 3 bedroom houses at Revelstoke.

It should be possible to draw upon local sources for labor. The mine and mill requirements based on a 200-ton per day operation are 40 men and a staff of six officials with additional clerical staff at Revelstoke.

XI

MARKETING

Some preliminary negotiations have been instigated, but no sales contracts finalized. Noranda Sales recently quoted prices ranging from $1.57 to $1.62 US per pound contained Mo in sulphide concentrates. A price increase of $0.10 US per pound Mo in sulphides was announced effective May 5th, 1969.

An average price of $1.83 Canadian per pound contained Mo has been assumed for this evaluation. The concentrate will be packaged in 33 gallon steel drums and palletized at 4 drums per pallet.

As is customary for this product, it is assumed that the market price of $1.83 Canadian per pound Mo will be f.o.b. the mine the customer providing containers and freight.

XII

SUMMARY OF COSTS

A. CAPITAL

	$
Mine and general equipment	330,900
Mine development	331,700
Power Plant and Compressors	220,000
Concentrator (including crushers)	600,800
Water System	60,000
Housing and roads	261,000
Working Capital	285,600
General Engineering Services	15,000
Contingencies 10% of above	210,500
Contingency for snow sheds	100,000
Total estimated capital	$2,415,500

These costs do not include work completed on the property prior to commencement of the ore pass raise, but in instances where equipment is being used on a rental purchase basis the original cost has been applied.

B. OPERATING COSTS

	$ Cost / Ton
Mine and surface plant	7.11 *
Milling	3.86 *
Camp and roads	2.55 *
Administration, including Headoffice	1.90
Exploration and development allowance	0.90
	$ 16.32

These are detailed and include an allowance of 50 cents per ton for exploration. Considerably more should be spent on exploration but in our opinion, this mine should not bear the total cost.

A development allowance of 40 cents per ton is included. A large portion of the development has been included in preproduction costs, but an allocation for further development will be required. As mining proceeds, development costs per ton may decrease.

* Power and compressor costs have been allocated to mine, mill and camp at 46:42:12 respectively.

All estimates are in Canadian funds.

DETAILS OF ESTIMATED COSTS

A. CAPITAL

Mine Equipment

	$	$
4 Scootcretes (or alternative units	32,000	
2 front end loaders	24,000	
12 - 5 ton cars	24,000	
1 Ring drilling rig	12,000	
3 - 50 HP scrapers	9,000	
12 drills (jack legs and stopers)	14,400	
1 - 10 ton truck	10,000	
1 transfer bin	10,000	
Ventilation fans 5 x 24"	7,500	
Spares	20,000	
		$162,900

General Equipment

1 - 10 ton flat bed	12,000	
1 - mine bus	15,000	
2 - mine trucks	6,000	
1 - grader (used)	25,000	
1 - Bulldozer (used)	60,000	
1 - Front end loader (used)	40,000	
Spares	10,000	
		$168,000

Power Plants and Compressors

2 - 500 KW generators with heat exchangers	160,000	
Powerhouse Building	10,000	
2 - 1000 cfm compressors	48,000	
Foundations	2,000	
		$220,000

Housing

5 - 8 man trailer units	30,000	
Cookhouse, wash house and dry	21,000	
5 - 3 Bedroom houses (Revelstoke)	100,000	
Recreational facilities	10,000	
Miscellaneous Buildings (powder magazine etc.)	10,000	
Office, warehouse (Revelstoke)	40,000	
		$211,000

	$	$
Road		
Allowances for further improvements		50,000
Mine Development		
Shaft system (hoist, installation & muck raise)	131,700	
Primary Development	200,000	
		331,700
Water System		
Dam at Lake	20,000	
Installation of 4000 ft of 6" pipe	30,000	
Pumps and ancillary equipment mine mill supply, fire and camp supply allow	10,000	
		60,000
Mill and Concentrator		
Crushing & screening equipment	76,930	
Concentrator equipment	115,100	
Assay and test labs	29,300	
Wood bins and tanks	31,565	
Miscellaneous constructions items	215,500	
Tailings disposal - initial	5,600	
Incidentals 15%	82,500	
Engineering fee	44,275	
		600,770
Working Capital - 3 months @ $16.32/ton		285,600
Engineering fees		15,000
Contingencies 10%		210,500
Road Contingency (for snow sheds)		100,000
Total Capital requirement		$2,415,470

B. OPERATING COSTS

Mine and Surface Plant

Labour	Men Req.	Daily Rate	Daily Cost	Monthly Cost	Annual Cost	Cost/ Ton
Underground Labor	7	50.24	352	7,392	88,000	
Miners	11	52.72	580	12,180	145,000	
Leadminers	1	54.24	54	1,134	13,500	
Mechanic	2	35.36	71	1,491	17,750	
Electricians	1	35.36	35	735	8,750	
Surface	2	28.24	57	1,197	14,250	
Truck Drivers	1	32.72	33	693	8,250	
	25		1182	24,822	295,500	4.22

Supplies	Daily Cost	Monthly Cost	Annual Cost	Cost/ Ton
Fuel (surface & UG Vehicles)	80	1,680	20,000	
Tires	25	525	6,250	
Spares (loco., loaders-shuttles)	150	3,150	37,500	
Bits and Steel	150	3,150	37,500	
Powder	120	2,520	30,000	
Timber & Rock Bolts	75	1,575	18,750	
	600	12,600	150,000	2.14
Total Mine & Surface Plant (exclude Power)				6.36

Mill	Men Req.	Daily Rate	Daily Cost	Monthly Cost	Annual Cost	Cost/ Ton
Mill Superintendent	1			1,200	14,400	
Assayer	1			900	10,800	
Crusher Operator	1	32.20	32.20	676	8,050	
Mill Mechanic	1	35.60	35.60	748	8,900	
Flotation Operator	4	32.20	128.80	2,705	32,200	
Mill Helpers	4	32.20	128.80	2,705	32,200	
Packer	1	28.24	28.00	588	7,000	
Laborer	2	28.24	57.00	1,197	14,250	
	15			10,719	127,800	1.83

Mill Operating Supplies	Monthly Cost	Annual Cost	Cost/ Ton
Grinding Media			0.25
Reagents			0.50
Maintenance			0.50
Assay and General Supplies			0.10
Total Mill (excluding Power costs)			3.18
Power & Compressor Costs all operations	9,216	110,600	
Compressor Maintenance	300	3,600	
	9,516	114,200	1.63

	Monthly Cost	Annual Cost	Cost/Ton
Administration (Engineering etc)			
Mine Manager	1,440	17,280	
Mine Superintendent	1,200	14,400	
Surveyor-Engineer	1,100	13,200	
Geologist - Sampler	1,050	12,600	
Helper	750	9,000	
Shift Boss	1,075	12,900	
Accountant Storeman -property	850	10,200	
" " - Revelstoke	825	9,900	
Secretary	350	4,200	
	8,640	103,680	
Engineering & Office Supplies	300	3,600	
	$8,940	$107,280	$1.53
Roads	$5,330	$ 64,000	$0.91
Camp Loss - 10/day for 40 men	$8,400	$100,800	$1.44
Overhead			
Transportation - men & supplies	400	4,800	
Communications	300	3,600	
Travel	500	6,000	
Insurance	420	5,000	
	$1,620	$19,400	$0.28
Allowances			
Exploration			$0.50
Development			$0.40
Total			$16.23
Head Office	$ 500	$ 6,000	$0.09
Total Operating Costs/Ton			$16.32

PRELIMINARY CASH FLOW PROJECTIONS

Assumptions:

Total Reserves	180,200 tons
Production per year	70,000 tons/year
Mine Life	2.574 years
Grade after diltuion	2.14% MoS_2
Mill Recovery	93%
Gross value per ton @ $1.83/lb of contained Mo	$43.70/ton
Operating Costs	$16.32/ton
Net Realization per ton	$27.38/ton
No loan is assumed	
Preproduction expenses	$ 331,700
Depreciable assets (Allowance for tax purposes taken at 30% of the undepreciated balance)	$1,783,200
Working Capital	$ 285,600

Under the Income Tax Act of Canada, income derived from the first 36 months production of a new mine need not be included in computing income taxes. We have, therefore, assumed no federal taxes applicable.

In the advent that the mine, by virtue of it's present life expectancy of less than 3 years, is not granted such an exemption, all expenditures on the property to date must be considered in determining a net cash flow.

C H A P M A N , W O O D , A N D G R I S W O L D L T D .

KING RESOURCES CO., COPELAND MNT. PROJECT, MO $1.83/LB

1000 CANADIAN $	1969	1970	1971	1972	TOTALS
METAL SALES	765.	3060.	3060.	993.	7876.
OPERATING COST	286.	1143.	1143.	371.	2941.
INTEREST INCOME	0.	0.	0.	0.	0.
TOTAL INCOME	480.	1917.	1917.	622.	
INTEREST EXPENSE	0.	0.	0.	0.	0.
PROV. DEPR. 1.00	0.	0.	0.	0.	0.
PROV. DEPR. .50	0.	0.	0.	0.	0.
PROV. DEPR. .30	480.	392.	274.	622.	1766.
PROV. DEPR. .25	0.	0.	0.	0.	0.
PROV. DEPR. .20	0.	0.	0.	0.	0.
PROV. DEPR. .15	0.	0.	0.	0.	0.
PROV. DEPR. .10	0.	0.	0.	0.	0.
PROV. PREPRODUCTION	0.	332.	0.	0.	332.
PROV. PFT. BFR. TX.	0.	1195.	1644.	0.	2838.
FED. DEPR. 1.00	0.	0.	0.	0.	0.
FED. DEPR. .50	0.	0.	0.	0.	0.
FED. DEPR. .30	0.	0.	0.	156.	156.
FED. DEPR. .25	0.	0.	0.	0.	0.
FED. DEPR. .20	0.	0.	0.	0.	0.
FED. DEPR. .15	0.	0.	0.	0.	0.
FED. DEPR. .10	0.	0.	0.	0.	0.
FED. PREPRODUCTION	0.	0.	0.	0.	0.
FED. PFT. BFR. TX.	0.	0.	0.	0.	0.
PROVINCIAL TAX	0.	180.	247.	0.	426.
FEDERAL DEPLETION	0.	0.	0.	0.	0.
FEDERAL TAX	0.	0.	0.	0.	0.
GROSS CASH FLOW	480.	1738.	1671.	622.	4510.
LOAN REPAYMENT	0.	0.	0.	0.	0.
NEW CAPITAL ASSETS	0.	0.	0.	0.	0.
WORKING CAP.RDMTN.	0.	0.	0.	286.	286.
NET CASH FLOW	480.	1738.	1671.	908.	4795.
CUM. CASH FLOW	480.	2218.	3888.	4795.	

A. M. COODE MAY 7 1969