REPORT
BRAMEDA RESOURCES LTD.
CASINO CREEK COPPER - MOLYBDENUM PROPERTY
DAWSON RANGE - YUKON TERRITORY

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April 11, 1970

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The President and Directors
Giant Mascot Mines Ltd.
7th Floor, 1177 West Hastings Street
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Gentlemen:

RE: REPORT
BRAMEDA RESOURCES LTD.
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DAWSON RANGE - YUKON TERRITORY

INTRODUCTION

This report has been prepared at the request of Mr. L.P. Starck, Vice-President, Giant Mascot Mines, to evaluate the above property and, from the information available, determine the advisability of proceeding from the exploration to the mine development stage, with the final objective of attaining large tonnage, open pit economic production. As much vital detailed information has yet to be obtained, assembled and assessed, most of which will come from future development, test work and other investigations, the following information must be considered as an estimate requiring further refinements and confirmation.

The property was first visited by the writer in June, 1969, when the first two drill holes were available for examination. Surface geology was just being mapped. The second visit was made April 7th, 1970, in conjunction with two days discussion with the resident geologist. Diamond drill core was examined, but snow cover prevented any surface observations. Prior to this visit, several days were spent on intensive study of all available data relating to the project. In this regard the assistance and co-operation of the Brameda personnel associated with the Casino project, and Mr. M.P. Phillips of Archer, Cathro & Associates Ltd., is sincerely appreciated.

The writer previously spent several years of mine operating in the Yukon and is quite conversant with general conditions that will obtain at this operation.

SUMMARY AND CONCLUSIONS

1. Exploration to date indicates that the property has the potential of a viable operation at an intial milling rate of 30,000 tons per day, with an estimated pay back of 5.35 years of pre-production and capital expenditures.

2. Confirmation of this opinion will be obtained by implementation of the proposed development program as in the first instance the reserve calculation is based on incomplete drilling information and cost estimates are of a very preliminary nature requiring a detailed feasibility study.

3. Reserves are estimated within a proposed pit area at 272,077,000 tons grading 0.274% Cu, 0.035% MOS_2 which, at 50¢ (U.S.) price for copper and \$1.72 (U.S.) per pound MO, results in a 0.345% Cu equivalent grade, using a 0.300% Cu equivalent cut-off grade.

4. The waste capping over the pit area, which is largely leached zone, amounts to 264,690,810 tons, giving a 0.97:1 waste:ore ratio.

5. The leached zone averages 0.053% Cu and assuming an economic leaching process can be developed, may contribute to the profitability of the operation.

6. Mineralization below the pit floor and walls, only partially tested to date, could contribute lower grade tonnage in the event that a 50,000 ton per day operation proved feasible.

7. Area "B", presently tested by seven widely spaced holes, indicates a possible ore grade to depths of 350 feet. The apparent low waste to ore ratio of .4:1 makes this area attractive for further exploration.

8. The proposed phased development program includes:

Phase 1

Drilling Pit Area "A"	Rotary	44,650 feet
	Diamond	32,950 feet
Total		77,600 feet
Exploration Drilling Area "B"	Diamond	12,000 feet
Total Rotary and Diamond Drilling		89,600 feet
Estimated cost		\$985,900.

Phase 2

Underground Development		
	Cross-cutting	3,000 feet
	Raising	1,000 feet
Total		4,000 feet
Pilot Mill Construction & Mill Testing		
	Estimated cost	\$775,000

Phase 3
Full feasibility study

Estimated cost	$500,000

Total cost of program:

Phase 1, 2, and 3	2,260,900
Administration, supervision, camp operation, etc.	1,250,000
	$3,510,900

9. Operating costs are estimated at $1.53 per ton milled. Smelting and refining costs, including concentrate transportation, are $0.60 per ton milled (0.345% Cu equivalent grade) and $0.81 per ton milled (0.469% Cu equivalent grade) for a total cost of $2.13 and $2.34 per ton milled respectively.

10. Net profit, before taxes and write-offs, is estimated at $0.70 per ton milled for lower grade ore and $1.51 per ton milled for the higher grade.

11. The estimated capital costs, including pre-production expenses, are estimated at $81,860,900. After allowing for anticipated Government assistance in the amount of $23,000,000, the total estimated cash requirements are $58,860,900.

Good

RECOMMENDATIONS

1. It is recommended that the Proposed Development Program be implemented, in anticipation that the property will attain a 30,000 ton per day production.

2. The services of a geologist, highly experienced in "Porphyry Copper" type deposits, on a consulting or permanent basis to complement the present geological direction, would be of considerable value.

3. Assays for oxide copper should be completed on all samples from the enriched zone, as appreciable percentages would seriously affect the economics of the operation.

4. At the present time whole diamond drill core is being sent for assay with specimens saved every ten feet. As the disseminated occurrence of sulphide mineralization precludes the possibility of biased sampling, it is recommended that core be split to provide complete geological information for continuing study.

LOCATION AND ACCESS

The property is located in the Dawson Range of the Yukon Territory, latitude 62°44' North, longitude 138°32' West, elevation 3000 - 4000 feet, approximately 187 miles by air north of Whitehorse. The terrain is mountainous, with generally moderate slopes and broad valleys marked by meandering streams and rivers and frequently extensive muskeg; the former conditions simplifying and the latter creating real problems in road construction. As a rule precipitation is light, but can be quite variable from year to year due to the depth of landward penetration of moist coastal air. Winters are cold and fairly long, but not sufficiently extreme to seriously effect an open pit operation.

At the present time access is by air summer and winter to an airstrip on the property, suitable for DC-3 aircraft. Transportation of freight is facilitated in the winter by a good winter road from the property to Burwash Landing, approximately Mile 1095 on the Alaska Highway, a distance of 140 miles. It is 178 miles by the Alaska Highway from this point to Whitehorse. In summer, freight may be moved by barge down the Yukon River to the confluence of Brittania Creek and the Yukon River, at which point an all weather road extends approximately 11 miles to the property.

The question of roads and access required for a production operation will be discussed in a latter section of the report.

HISTORY

The first lode claim in the area was staked in 1901. Placer mining for gold and tungsten over the ensuing years was carried out by various operators, particularly along Canadian Creek, and molybdenite was noted during these operations. The area was mapped in 1916 by D.D. Cairnes of the Geological Survey of Canada (1917 Report) who noted the intense leaching and alteration of the Casino stock. Galena veins were discovered on Casino Creek and explored from 1964 to 1967. The presence of a porphyry deposit was confirmed by diamond drilling in June, 1969, subsequent to indicative stream sediment sampling, followed by grid soil sampling and geological mapping.

GEOLOGY

The property is located along the northeast margin of the Klotassin batholith, which marks the northeast side of the Coast Range batholith. This intrusive varies in composition from a quartz monzonite to granodiorite and invades the older Yukon Group metasediments. The Klotassin and Yukon rocks are intruded by younger quartz rich stocks, with which most of the potentially economic mineralization is related.

A broad section of the Yukon, including the Casino area, was unglaciated during the last ice age and consequently all soils are residual and particularly amendable to geochemical surveying. This condition also explains the presence of a near surface leached zone, overlying an enriched zone before the primary or protore zone is reached.

The Casino stock, which is about 2000 feet in a north-south direction, by 5000 feet east-west is one of the quartz rich intrusive bodies. Dr. J.M. Carr, Brameda Resources geologist, has subdivided various phases of the Klotassin rocks and notes similarities of the quartz diorites and granodiorite with the intrusives in the Highland Valley camp and near the Brenda Lake stock. The stock itself is made up of quartz and feldspar porphyries and associated breccias. The quartz porphyry phase in the eastern half of the stock is apparently more readily brecciated than the feldspar porphyry and the breccia is considered to be of explosive origin.

The stock has been intensely hydrothermally altered and this alteration extends about 2000 feet into the surrounding granodiorite. The central core is typefied by the development of K-feldspar and biotite alteration, surrounded successively by argillic, quartz-sericite-pyrite and chloritic zones, common to "porphyry copper" type deposits.

The accompanying surface geological plan and vertical cross sections present a geological interpretation of the main area of interest based essentially on diamond drill hole information. The various rock types are shown in two main groups, the quartz and feldspar porphyries and the quartz diorite-granodiorite complex, for reasons of simplicity.

A number of interesting and possibly important relationships are indicated, the most striking being the apparent association of better grade mineralization, in both porphyry and quartz diorite and/or granodiorite, with well defined fault and fracture zones. The northern limit of the mineralized zone is bounded by a fault zone comprised of at least two main strands, striking approximately north 70° west. (Faults "A-A" and "B-B" on the plans and sections) "A-A" dips 70° south and "B-B" steeply north. These faults are recognized in diamond drill holes and correspond with faulting interpreted from geophysical work. Two north-south striking faults "C-C" and "F-F" probably have steep dips, cut through the main Casino stock and suggest post-intrusive movement. As noted above there is a spatial relationship between these faults and higher grade values, particularly along the strike and dip of "A-A". The intensity of K-feldspar and biotite alteration adjacent to this fault, the rough halo effect of the other alteration zones and the distribution of brecciated areas suggests post-intrusive movement along "A-A" and raises the interesting question of where the offset portion, if any, of the Casino stock may lie. Additional geological information is necessary to evaluate this possibility.

The southeast boundary of the stock may be bounded by fault "E-E", but there is only limited confirmatory evidence for this structural feature. Other faults shown on the maps, which generally parallel the two main trends are the result of geophysical surveying.

The Yukon series of metasediments was intersected in drill hole $^{D}1$ approximately 4000 feet west of the Casino stock, but its aerial extent is not known.

MINERALIZATION

The copper and molybdenum minerals in the Casino stock vary according as they occur in the leached, enriched or protore zones.

The leached zone contains jarosite, goethite and hematite, oxidation products of copper and iron sulphides, with minor quantities of copper oxide and silicate minerals. The depth of this zone varies from 0 - 500 feet, averaging 235 feet and may be of economic importance if a suitable leaching process can be developed.

The enriched zone is characterized by the presence of sooty secondary chalcocite which partly replaces and coats fractures in pyrite and chalcopyrite. Other minerals such as covellite, cuprite and oxidized iron minerals are present in varying minor amounts. Primary sulphide minerals are chalcopyrite, pyrite and molybdenite. Molybdite may be present as a very minor accessory.

The profile of the enriched zone has been interrupted by faulting and in the central portion of the map area is not present. The fracturing attendant with the main faults has apparently facilitated leaching and enrichment. It is interesting to note that higher copper grades in the enriched zone do not necessarily overly higher protore copper grades. The thickness of this zone, where present, varies from 20 - 300 feet, averaging 195 feet.

The protore zone mineralization is pyrite, chalcopyrite and molybdenite disseminated in the altered rocks and partly in vuggy fracture-fillings, with quartz, silicate minerals and carbonate. Pyrite is most abundant, amounting to about 1% by volume and locally up to 10%. All rock types contain sulphides. Magnetite, associated with chalcopyrite is found in fractures, locally. There is a suggestion of zoning with a slight decrease in copper and increase in molybdenum at depth, but in general the sulphide distribution is difficult to explain geologically, except that, in part, higher grade mineralization is associated with tectonic brecciation and fracturing related to faulting. There is no known termination to sulphide mineralization, which is still present to 1500 feet below surface.

WORK TO DATE

Stream sediment sampling indicated the specific area of interest, which was then subjected to grid soil sampling, geophysical surveying and geological mapping. A total of approximately 57.5 line miles was surveyed on a 400 foot grid pattern. The geochemical surveying was responsible for pinpointing the target area, while the geophysical work delineated certain structural features, which may be of considerable importance.

The drilling program is summarized as follows:

Diamond drilling - mostly NQ (1 7/8" diameter) core	35,802 feet
Rotary drilling - 4 7/8" diameter hole	4,427 feet
Total	40,229 feet

ORE RESERVES

Although sufficient detailed drilling has not been completed to calculate accurate ore reserves, or even determine the vertical or lateral limits of economic mineralization, it was decided that a reasonable approach would be to roughly outline a possible pit based on existing assay information. By this approach it was anticipated that a fairly reliable average grade of mineralization would be obtained and an approximate waste:ore ratio established. The outline of this approximized pit is shown on the accompanying map, Pit Layout and Proposed Development Drilling and the appropriate cross sections. The maximum depth of the pit is 1000 feet and final pit slope is 45°.

The following parameters were used in the calculation:

1. Cut off grade 0.300% Cu equivalent.
2. Polygonal Block Method for contiguous ore holes.
3. Sphere of influence of isolated holes (essentially lower grade), a circular area of 200 foot radius throughout the length of the hole.
4. The undrilled volume below the leached capping will have the same grade as the average of all the drill holes inside the pit.
5. One ton of ore or waste = 12 cu. ft. This is considered conservative for the leached capping tonnage calculation.
6. The following holes were used in the calculation "High Grade Holes": P-1, P-6, P-8, P-16, P-22, P-31, P-34, P-35, P-36, P-37, P-41, P-41A, R-1 and R-2.

A Summary of Ore Reserves tabulation appears on the following page and the details of the calculation are given in Appendix 3.

The following figures from the above summary and other calculations are particularly pertinent:

	ORE TONS	% Cu	% MOS_2	% Cu EQUIVALENT	WASTE CAPPING TONS
Ore Blocks - Enriched and Protore	146,048,000	0.317	0.042	0.404	58,839,890
Low Grade Holes, Enriched and Protore	46,006,000	0.136	0.011	0.159	30,050,921
Undrilled	80,023,000	0.274*	0.035*	0.345*	175,800,000
	272,077,000	0.274	0.035	0.345	264,690,810
Leached zone :	88,890,811	0.053	0.009	0.070	
NOTE * Assumed grade					
Low grade below pit floor and walls	77,573,992	0.155	0.021	0.198	
Unexplored to 300 feet below pit floor and walls	121,600,000	-	-	-	

It should be noted that of the total drilled tonnage in the pit, 76% averages 0.404% Cu equivalent. Only 30% of the total tonnage remains undrilled at 400 foot centres and has therefore been assigned the average grade of the drilled portion.

The waste:ore ratio is .97:1.

Several holes extend below the pit floor or walls to a maximum depth of 300 feet. Some interesting values occur in this area, but the average grade is low. No drilling of the area is planned in the proposed development program.

The calculated ore tons would provide a mine life of 15 years at 50,000 tons per day, or 25 years at 30,000 tons per day.

Additional tonnage potential is indicated in Area B to the northwest of the proposed pit. Using the same parameters as in the pit calculation, four widely spaced diamond drill holes indicate a .302% Cu equivalent grade, for an average of 5,000,000 tons per hole. Considerable more drilling is required to confirm the importance of the area, but while depths of possible economic mineralization only extend to about 350 feet, the indicated waste:ore ratio is in the order of .4:1. A total of 14 holes are planned to further explore the general area.

SUMMARY OF ORE RESERVES

ORE BLOCKS	TONS	% Cu	% MOS_2	% Cu Equivalent
Leached Zone	58,839,890	0.069	0.011	0.092
Enriched Zone	45,152,720	0.478	0.037	0.555
Protore Zone	100,895,100	0.245	0.044	0.336
Protore and Enriched Zones	146,048,000	0.317	0.042	0.404
LOW GRADE HOLES				
Leached Zone	30,050,921	0.020	0.004	0.026
Enriched Zone	13,647,500	0.172	0.016	0.200
Protore Zone	32,358,570	0.120	0.009	0.139
Protore and Enriched Zones	46,006,000	0.136	0.011	0.159
ORE BLOCKS & LOW GRADE HOLES				
Leached Zone	88,890,811	0.053	0.009	0.070
Enriched Zone	58,800,220	0.407	0.032	0.473
Protore Zone	133,253,670	0.215	0.036	0.288
Protore and Enriched Zones	192,054,000	0.274	0.035	0.345
HIGH GRADE HOLES				
Leached Zone	19,704,500	0.089	0.010	0.109
Enriched Zone	24,745,000	0.460	0.046	0.659
Protore Zone	53,529,000	0.260	0.058	0.381
Protore and Enriched Zones	78,274,000	0.324	0.054	0.469

METALLURGY

Preliminary bench testing has been carried out in Seymour Laboratories Ltd., North Vancouver, and by Colorado School of Mines Research Institute, Golden, Colorado, on core samples largely taken from the enriched zone in several diamond drill holes. Recently Britton Research Ltd. commenced bench testing for Pilot Mill design criteria.

The CSMRI testing was solely on enriched ore with satisfactory results, and indicated that the ore is amendable to conventional flotation treatment of copper-molybdenum ores. Recoveries from bulk copper-molybdenum flotation attained 85.8% Cu and 87.7% MOS_2. A third cleaner concentrate contained 35.3% Cu and 0.10% MOS_2. The ore contained 0.065% oxide copper.

Seymour Laboratories tested enriched ore, a mixture of enriched and protore and one test on protore alone. Recoveries of the enriched and the mixed enriched and protore varied from 78.5 - 90.0% for copper and 86.3 - 94.5% for MOS_2. The protore recoveries were 90.7% and 93.7% respectively for copper and MOS_2. Some difficulties were encountered in producing a satisfactory copper concentrate.

Based on the above work and being influenced largely by the Seymour results, who conducted the greatest number of tests, it would seem reasonable to assume 82% recovery for both copper and MOS_2 and concentrate grades of 20% for Cu and 52% for MOS_2 for the preliminary estimates in this report.

The following test work should be carried out as soon as practicable to provide data for a mill flowsheet. Determine:

1. Recoveries in the enriched zone. More assay data is required on the oxide content of this zone.
2. How the protore would respond to processes necessary for good recovery of the enriched mineralization.
3. The coarsest primary grind possible while obtaining good recovery. This is important in the separation of MOS_2 from copper minerals as MOS_2 is often attached to and in copper minerals in the minus 325 to 400 mesh range.
4. Quantity and type of collecting agent to obtain low tailing.
5. Type of frothing agent, as some difficulties encountered in obtaining good froth in early testing.
6. Locked cyclic testing to more accurately determine the grade of concentrate and recovery to be expected in plant operation.
7. A laboratory program to develop a process for the recovery of the molybdenite into a concentrate.
8. Leach tests on the upper leached zone, for recovery of the copper silicate and oxide minerals.

FUEL SUPPLY

It is anticipated that fuel supply will be tied into the concentrate haulage, using petroleum products for partial back-haul. Assuming that concentrates will be shipped by truck to Whitehorse, a distance of 235 miles, and by comparison with a contract rate on the Alaska Highway, a transportation cost of 4¢ per gallon could be expected. The average price of standard and low-pour fuels is 27.5¢ per gallon f.o.b. truck, Whitehorse. Thus a price of 31.5¢ per gallon is estimated in tankage at the mill.

ROADS

Brameda Resources Ltd. has had discussions with the Department of Northern Development and was told that a suitable road would be constructed, at Government cost, to the vicinity of the property. As in the case of Anvil Mines, it might be anticipated that further assistance, to the extent of 60% of the total cost, for the construction of a nine mile road from the mill to the townsite would be assumed by the Federal Government. However, capital provision in the amount of $200,000 should initially be included in overall estimates for the nine mile road construction.

The two probable road routes from present highways were flown by the writer during the recent visit to the property. The route from Carmacks, through the La Forma property and along the Big Creek and Hayes Creek valleys to Selwyn Creek and thence across higher country to the property, a distance of 121 miles, is favoured by the writer.

Some of the reasons are as follows:

1. Except for an estimated ten mile stretch along the Hayes River Valley, where rock work will be necessary, the route presents generally favourable construction conditions. There are no major river crossings, although the road will be on north facing slopes from La Forma westward, presenting perma-frost conditions.
2. The 35 mile road from Carmacks to La Forma is well known by the writer and can be upgraded to a suitable road at minimum cost.
3. The route will provide access to geologically favourable country, where exploration is now in progress.
4. There is the probability that White Pass and Yukon railhead will be extended from Whitehorse to Carmacks, which would become a transportation centre for this section of the Yukon.
5. The main disadvantage to this route is the 2000 foot climb from the Selwyn Valley to the property.

The route westward from Casino to Burwash Landing on the Alaska Highway is slightly longer, 140 miles, and is the present winter road route. It traverses broad areas of low ground, particularly east of Wellesley Lake and again in the valleys of Onion and Brooks Creeks, where muskeg conditions prevail. Following higher ground on the south slopes of these broad valleys would increase mileages. Bridge construction of major proportions would be required at the Kluane River about one mile east of the Alaska Highway. Grades throughout the route would be moderate.

This route would be required for truck concentrate haul to Haines, Alaska if ocean shipment from this port was selected in preference to Skagway, Alaska.

Based on current transportation costs, combined truck and rail, from Anvil Mines to Skagway, it is estimated that costs of concentrate haulage from the Casino property to tidewater will be $20.00/ton.

The following tabulation gives mileages between important road and rail junctions:

Westerly Route:	
Casino property - Mile 1095, Alaska Highway Present winter road	140 miles
Mile 1095 - Whitehorse	178 miles
Whitehorse - Skagway (rail)	110 miles
Total	428 miles
Easterly Route:	
Casino property - La Forma property	86 miles
La Forma - Carmacks (road to be upgraded)	35 miles
Carmacks - Whitehorse	109 miles
Whitehorse - Skagway (rail)	110 miles
Total	340 miles
Mileages for truck haul to Haines:	
Casino property - Mile 1095	140 miles
Mile 1095 - Haines Junction	80 miles
Haines Junction - Haines	151 miles
Total truck haul	371 miles

COMMUNICATIONS

Present communications are by radio via an intermediate station near Beaver Creek on the Alaska Highway, which is inadequate for an on-going operation. The ultimate communication link between the mine townsite to Carmacks or Minto would be installed by Canadian National Telecommunications. A link from the mill and mine to the townsite is a capital charge to the operation and is estimated to be $120,000.

TOWNSITE

The estimates for the probable size of townsite are based on the assumptions that a 30,000 ton per day operation will require a total labor force of 300 employees and a 50,000 ton per day operation a total of 500 employees, and that a high percentage of married personnel and good standard of residences is necessary to draw and hold a good labor force in this remote location. A town population, including services, etc., is estimated to be 900 for the lower tonnage operation and 1500 for the larger.

Services for the town are understood to be provided by the Federal Government at no cost to the operation. The following table outlines a very preliminary cost estimate.

	30,000 Tons per day	50,000 Tons per day
Bunkhouse @ $1500/unit (room)	$110,000	$190,000
Apartment suites @ $17,000/unit	1,200,000	2,000,000
Row-housing @ $17,000/unit	1,100,000	1,880,000
Single-family 2 b.r. @ $26,000/unit	1,300,000	2,080,000
Single-family 3 b.r. @ $30,000/unit	2,040,000	3,450,000
Sub-total	$5,750,000	$9,600,000
Community Space @ $25/sq.ft.	1,120,000	1,880,000
Services	615,000	920,000
Total development	$7,485,000	$12,400,000

At the new town, Faro, constructed for the Anvil Mines operation, approximately 2/3 of the construction costs were absorbed by Central Mortgage and Housing Commission. If this arrangement can be achieved for the Casino townsite, and considering the assistance by the Federal Government re services, then the total capital cost to the operation would be $2.3 million for a 30,000 ton per day operation and $3.8 million for a 50,000 ton per day operation.

TAILINGS DISPOSAL

The Casino Creek Valley has been reserved for tailings disposal in preliminary planning. Dams will be built at suitable points (please refer General Area Map) with rock-fill toe-dams upstream and downstream, utilizing mine waste, the body of the dam being sand fill. Tailings would flow by gravity to the pond where the coarse fraction is split out for dam building, the slimes being introduced into the pond. Reclaim water would be recovered by floating pump stations and then pumped to a reclaim water reservoir above the mill.

The following table summarizes preliminary cost estimates for a 30,000 and 50,000 ton per day operation:

	30,000 Tons per day	50,000 Tons per day
Estimated Costs		
Pumps	$400,000	$630,000
Electrical	500,000	600,000
Buildings and structures	400,000	600,000
Instrumentation	40,000	50,000
Reclaim pipe-line	510,000	600,000
Valves	70,000	100,000
Tailings Pipe-line	670,000	1,050,000
Dams	800,000	800,000
	3,390,000	4,430,000
+ 10% contingency	340,000	440,000
+ 10% engineering	340,000	440,000
Total estimated cost	$4,070,000	$5,310,000

WATER SUPPLY

There are three alternatives for developing an adequate water supply, which have been investigated in a preliminary fashion.

1. Collect surface water in the Dip Creek watershed and pump from the dam on Dip Creek to the mill.

Certain assumptions must be accepted for this approach, such as:

(a) That material for an earth-fill dam can be found near the confluence of Casino and Dip Creeks. This may be critical as early reconnaissance has not disclosed the presence of a suitable clay soil.

(b) That the valley of Dip Creek will present no serious foundation problems. Perma-frost conditions will almost certainly be met.

(c) That the excavation of a trench for a pipe-line from the dam to the mill is practical in the economic sense. This is not thought to be too serious a problem.

Meteorological data from Government records show that the flow in Dip Creek, just upstream from Casino Creek, is adequate to support a 50,000 ton per day milling operation. Detailed recording of flow should commence immediately as last summer was an exceptionally dry season and there has been abnormally light snowfall this winter. Water rights have been obtained for this watershed.

The water reservoir would be about seven miles from the mill and 1700 feet lower in elevation.

The following cost estimates have been developed by the Brameda staff:

ITEM	COST	
	30,000 Tons Per Day	50,000 Ton Per Day
Dam - 80 feet high, 2600 feet long Reservoir 20,000 acre-feet 1.5 years demand	$1.4 million	
- 95 feet high, 2800 feet long Reservoir 30,000 acre-feet 1.5 years demand		$2.2 million
Pumping Installation - 9500 H.P. 14,400 G.P.M. Dynamic Head 1930 feet	1.4 million	
- 16,500 H.P. 24,000 G.P.M. Dynamic Head 1940 feet		2.4 million
Pipeline - 30 inch diameter steel 35,000 feet long	1.2 million	
- 36 inch diameter steel 35,000 feet long		1.6 million
Powerline and Control Circuits	0.1 million	0.1 million
Contingencies and Engineering	0.8 million	1.2 million
Total capital costs	$4.9 million	$7.5 million
Operating costs	2¢ per ton milled	2¢ per ton milled

2. Pumping from the Yukon River at the confluence of Brittania Creek and the Yukon River, a distance of eleven miles and over a vertical difference in elevation of 3100 feet. A very rough capital cost estimate for this installation is $6.0 million for a 30,000 ton per day operation and $9.0 million for a 50,000 ton per day.

   Not included in the above estimates, but of major importance, is an installation around the pump intake, as protection against heavy flow ice in the spring and fall of the year.

3. The location of a subsurface aquifer. The presence of such underground sources is in evidence throughout the general area, and gravel deposits in valley floors could provide a reservoir of adequate size and permeability for the operation. Considerable study, followed by drill testing would be required.

POWER SUPPLY

The Department of Northern Development has indicated to Brameda Resources that primary electric power would be supplied at no cost to the operation. However, a transformer station at the mill and power distribution to the mine and plant facilities would be a capital cost. By comparison with Brenda, the stand-by power, terminal facilities and plant distribution cost is estimated at $1.0 million for a 30,000 ton operation and $1.5 million for 50,000 tons per day.

The power rates currently in force in the Yukon vary from 14 to 20 mils per KWH. Assuming a rate of 15 mils, the annual power bill for the mine and mill is estimated at $3.1 million for 30,000 tons per day and $5.2 million for a 50,000 ton per day operation, or 30¢ per ton milled. This is a major cost item and investigations into a thermal power plant, possibly utilizing coal from the mine at Carmacks, should be commenced at an early date.

FEDERAL GOVERNMENT SUPPORT

Several references to Federal Government support have been made in the above sections, and the estimated costs are summarized below. Anvil Mines obtained similar support in bringing the Faro operation into production, the repayment of same being contingent upon the establishment of a smelter, if found to be feasible. It is believed that such assistance would be forthcoming from the Federal Government when a decision is made to bring the Casino project into production.

ITEM	30,000 Tons Per Day	50,000 Tons Per Day
Road, approximately 116 miles @ $40,000	$4.6 million	$4.6 million
Power transmission		
Approximately 110 miles @ $50,000/mile	5.5 million	5.5 million
Power generation		
Additions to power station	7.7 million	7.7 million
Townsite development	5.2 million	8.6 million
Total estimated Government assistance	$23.0 million	$26.4 million

ECONOMIC CONSIDERATIONS

The estimated reserves indicate that a 30,000 ton per day operation should be planned initially, with provision made for increasing to 50,000 tons per day as the reserve potential dictates.

An area of higher grade tonnage, containing 78,274,000 tons averaging 0.469% Cu equivalent, will provide the bulk of production for the first two years, while it is assumed that the average pit grade of 0.345% Cu equivalent will be maintained thereafter. Obviously these are rather broad assumptions and will only be confirmed by future drilling, but must be made at this stage to project a possible cash flow.

1. Operating Costs
   Assumptions - 30,000 ton per day plant. Waste:ore ratio 1:1.
   82% recovery copper and MOS_2.

		Per Ton Milled .345% Cu Eq.	Per Ton Milled .469% Cu Eq.
Mining - Leached Capping	$ .13		
Ore	.35	$.48	$.48
Milling		.85	.85
Administration, Overhead		.20	.20
		$1.53	$1.53
2. Smelting and Refining			
Transportation concentrates		.35	.43
Smelting and refining		.25	.38
		$ .60	$ .81
3. Total Cost		$2.13	$2.34
4. Value		$2.83	$3.85
5. Net Profit (before taxes, write-offs)		$ .70	$1.51

CAPITAL REQUIREMENTS

1. Pre-production		
Development program	$3,510,900	
Pit preparation		
20,000,000 tons stripping leached capping @ $.13/ton	2,600,000	
		$6,110,900

2. Plant and Equipment

Mining equipment	$9,500,000	
Plant site excavation	1,750,000	
Milling plant - crushing, concentrating	25,000,000	
Plant services, tailings disposal, water power, housing, etc.	22,000,000	
Overhead costs	17,500,000	
		$75,750,000
Total		81,860,900
Less Assumed Government Assistance		23,000,000
Total Cash Requirements		$58,860,900

Cash Flow (before taxes or write-offs)

First year	$16,534,450	
Second year	16,534,450	
Third year	7,665,000	
Fourth year	7,665,000	
Fifth year	7,665,000	Pay back 5.35 years
Sixth year	7,665,000	

PROPOSED DEVELOPMENT PROGRAM

The proposed program has been divided into three main phases of which the latter two phases, underground development and pilot mill testing, and a full feasibility study for preparation of the property for production, would be contingent on continuing confirmation of tonnages and grades as outlined under "Reserves", during the first phase, which is essentially drilling. Early successes in the drilling program would permit overlapping of Phases 1 and 2.

PHASE 1 -

The fill-in development drilling in Area A, as shown on the accompanying map, entitled Pit Layout and Proposed Development Drilling, will provide assay and geological information on a 200 foot grid spacing over the main mineralized area within the limits of the proposed pit. Such closely-spaced drilling is considered necessary due to the marked variations in grade in a few closely spaced holes. Additional holes are spaced at 400 foot centres in the remainder of the area to the perimeter of the pit. Except where geological information is required, it is proposed to utilize rotary drilling to the top of the protore zone, followed by diamond drilling to total depth to ensure maximum reliable assay information. Taking into account anticipated broken ground conditions, some holes will be rotary drilled to total depth.

On this basis the following development drilling of 91 holes is proposed:

Rotary drilling	44,650 feet
Diamond drilling	32,950 feet

Appendix 1 details the location and rotary and diamond drill footages for each hole. In all cases total depth of holes is at the floor or walls of the proposed pit.

Area "B" as outlined on the above noted map has shown encouraging response to early widely spaced drilling and additional drilling at approximately 800' centres is proposed to check this and the ground to the north of faults "A-A" and "B-B". Geological information is important at this stage and consequently unless ground conditions dictate otherwise, it is proposed that the total footage will be by diamond drilling. The total footage is 12,000 feet for 14 holes.

Total drilling footage for Phase 1 - 89,600 feet.

Stream sediment sampling indicates the advisability of extending the geochemical and geological mapping coverage to the west and north of the presently known Casino stock. This would require a geologist and helper for one month.

PHASE 2 -

Underground exploration is required to provide representative ore for mill testing and give an indication of ground conditions, which information will be invaluable in ultimate pit design. Consideration was given to utilizing rotary drill cuttings for test work in a 5 - 10 ton per day test mill. Cuttings from 44,650 feet of 4 7/8" diameter hole would provide a total of 535 tons of material. Split core from diamond drilling would contribute another 26 tons. The total is not believed sufficient for adequate testing. The drill cuttings from rotary drilling are very fine and might not be entirely suitable for mill test purposes.

Rotary drilling of a six inch hole would provide about 4500 tons of cuttings, but the increased costs of rotary drilling and deepening by diamond drilling makes this approach impractical.

A program of 3000 feet of cross cutting and 1000 feet of raising would give adequate tonnages of both enriched and protore by collaring an adit of approximately elevation 3850. Some underground diamond drilling might be included in this phase.

A suitable test mill might be moved to the property by barge during the summer months, but in all probability such a move would be more expedient via winter road at the year end.

PHASE 3 -

This would entail a full feasibility report.

The following table summarizes the estimated costs:

Phase 1:					
	Development Drilling - Area A	Rotary	44,650 x $10.00 (direct)	$446,500	
		Diamond	32,950 x $12.00 (direct)	395,400	
	Exploration Drilling Area B	Diamond	12,000 x $12.00 (direct)	144,000	
					$985,900
Phase 2:					
	Underground Development		4,000 x $100.00	$400,000	
	Pilot Mill, equipment, construction and operation			375,000	
					$775,000
Phase 3:					
	Feasibility				$500,000
					$2,260,900
Administration, supervision, camp operation, etc.					$1,250,000
Total cost Development Program					$3,510,900

Respectfully submitted,

Walter E. Clarke

W.E. Clarke, B.Sc., P.Eng.
Consulting Engineer

April 11, 1970

PROPOSED DEVELOPMENT DRILLING

LOCATION:

SOUTH	EAST	WEST	ROTARY FOOTAGE	DIAMOND DRILL FOOTAGE	TOTAL FOOTAGE
48	22		400		400
	26		500		500
	34		300	300	600
	38		400		400
46	30		300	500	800
44	10		600		600
	17.5		600		600
	27		300	500	800
	32		300	500	800
	35.5		300	500	800
42	17		650	150	800
	22		700	200	900
	24		650	350	1000
	26		500	400	900
	28		500	300	800
	30		500	300	800
	32		500	300	800
	34		500	300	800
40	6		1000		1000
	10		800	200	1000
	14		700	200	900
	20		700	200	900
	24		600	400	1000
	27.5		400	450	850
	32		500	350	850
	38		400		400
38	6		700	300	1000
	10		800	200	1000
	14		650	350	1000
	16		700	400	1100
	18		800	400	1200
	20		700	500	1200
	22		700	500	1200
	24		600	450	1050
	26		500	450	950
	28		500	400	900
	30		500	350	850
36	2		600	350	950
	4		700	300	1000
	8		700	300	1000
	12		700	300	1000
	14			450(P-46B)	450
	16		700	500	1200
	18			650(P-42)	650
	20		800		800
	24		500	550	1050
	28		400	500	900
	34		500		500
	38		300		300

PROPOSED DEVELOPMENT DRILLING

LOCATION:

SOUTH	EAST	WEST	ROTARY FOOTAGE	DIAMOND DRILL FOOTAGE	TOTAL FOOTAGE
34	6		300	500	800
	10		600	300	900
	12		650	350	1000
	14		700	400	1100
	16		700	500	1200
	18		700	600	1300
	20		700	500	1200
	22		650	550	1200
	24		500	550	1050
	26		400	500	900
	28		400	400	800
32	2			500	500
	6			600	600
	8		500	300	800
	12		650	350	1000
	16		700	500	1200
	20		600	700	1300
	24		400	650	1050
	26			700(P-41A)	700
	28		300	400	700
	30		500		500
	34		300		300
30	10		400	300	700
	14		650	350	1000
	18		650	550	1200
	20		600	600	1200
	22		600	500	1100
	24		500	500	1000
	26		900		900
28	6			400	400
	10			600	600
	14			850	850
	18			1050	1050
	19.7		600	500	1100
	23		1000		1000
	27		550		550
26	21.5		600	300	900
	24.7		700		700
24	19			700	700
	22			800	800
	26			500	500
			44,650	32,950	77,600

PROPOSED EXPLORATION DRILLING

LOCATION:

SOUTH	NORTH	EAST	WEST	ROTARY FOOTAGE	DIAMOND DRILL FOOTAGE	TOTAL FOOTAGE
20			10		750	750
		2			750	750
		14			1000	1000
12			10		750	750
		2			750	750
		18			1000	1000
		26			1000	1000
4		2			750	750
		18			1000	1000
		26			1000	1000
	4		10		750	750
		2			750	750
		18			1000	1000
	8	10			750	750
					12,000	12,000

ORE BLOCKS

LEACHED ZONE

BLOCK NO.	HOLE NO.	DEPTH	LENGTH	%Cu	% MOS_2	% Cu. Eq.	TONS
1	P-19	0-360	360	.035	.004	.043	3,718,800
2	P-17	0-120	120	.053	.002	.057	1,359,600
3	P-10)						
	R-3)	0-385	385	.060	.011	.083	5,082,000
4	P-44	0-255	255	.052	.008	.068	3,167,100
5	P-45	0-255	255	.080	.013	.107	3,187,500
6	P-13	0-265	265	.138	.002	.142	3,445,000
7	P-42	0-465	465	.058	.027	.114	7,105,200
8	P-14(65°)	0-345	313	.016	.004	.024	2,513,390
9	P-16(75°)	0-210	203	.087	.008	.103	2,842,000
10	P-34	0-315	315	.071	.013	.098	3,962,700
11	R-1	0-321	321	.153	.011	.176	3,412,230
12	P-12(60°)	0-150	130	.141	.003	.147	952,900
13	P-1 (50°)	0-165	126	.057	.009	.075	1,485,540
14	P-36	0-150	150	.061	.009	.080	1,137,000
15	P-6(65°)						
	P-8)	0-60	60	.075	.001	.077	319,800
16	P-35	0-180	180	.069	.008	.085	2,354,400
17	P-22)						
	R-2)	0-35	35	.096	.006	.108	507,500
18	P-31	0-65	65				861,250
19	P-37	0-75	75	.045	.002	.049	838,500
20	P-41)						
	P-41A)	0-120	120	.103	.016	.136	1,983,600
21	P-38)						
	R-8	0-306	306	.010	.010	.031	3,941,280
22	P-46)						
	P-46B)	0-380	380	.065	.013	.092	4,662,600

NOTE: P-37 0-50 - Casing Grade 50-75 Assumed to surface

P-31 0-65 - Casing Grade is average for entire zone

ORE BLOCKS

ENRICHED ZONE

BLOCK NO.	HOLE NO.	DEPTH	LENGTH	% Cu.	% MOS_2	% Cu. Eq.	TONS
2	P-17	120-150	30	.235	.017	.270	339,900
3	P-10)						
	R-3)	385-658	273	.462	.039	.542	3,603,600
6	P-13	265-625	360	.274	.011	.297	4,680,000
7	P-42	465-585	120	.449	.045	.542	1,833,600
8	P-14(65°)	345-730	349	.297	.054	.408	2,802,470
9	P-16(75°)	210-525	304	.402	.044	.493	4,256,000
10	P-34	315-465	150	.465	.083	.636	1,887,000
11	R-1	321-573	252	.688	.076	.846	2,678,760
12	P-12(60°)	150-300	130	.401	.025	.452	952,900
13	P-1(50°)	165-381	165	.571	.034	.640	1,945,350
14	P-36	150-315	165	.368	.016	.401	1,250,700
15	P-6(65°)						
	P-8)	68-180	112	.365	.014	.394	596,960
16	P-35	180-300	120	.411	.007	.425	1,569,600
17	P-22)						
	R-2)	35-240	205	1.133	.022	1.178	2,972,500
18	P-31	65-260	195	.546	.099	.750	2,583,750
19.	P-37	75-255	180	.511	.041	.595	2,012,400
20	P-41)						
	P-41A)	120-301	181	.428	.028	.486	2,991,930
21	R-8	345-645	300	.396	.010	.417	3,864,000
22	P-46B	415-605	190	.448	.021	.491	2,331,300

ORE BLOCKS

PROTORE ZONE

BLOCK NO.	HOLE NO.	DEPTH	LENGTH	%Cu	% MOS_2	% Cu Eq.	TONS
1	P-19	360-450	90	.165	.022	.210	929,700
2	P-17	150-997	847	.242	.032	.308	9,596,510
3	P-10)						
	R-3)	658-857	199	.269	.028	.327	2,626,800
4	P-44	255-810	555	.276	.020	.317	6,893,100
5	P-45	255-1110	855	.196	.014	.225	10,687,500
6	P-13)						
	R-5)	625-1255	630	.179	.032	.245	8,190,000
7	P-42	585-737	152	.386	.049	.487	2,322,560
8	P-14(65°)	730-972	219	.250	.136	.530	1,758,570
9	P-16(75°)	525-1150	604	.207	.110	.434	8,456,000
10	P-34	465-1183	718	.147	.098	.349	9,032,440
11	R-1	573-1001	428	.229	.051	.334	4,549,640
14	P-36	315-945	630	.266	.027	.322	4,775,400
16	P-35	300-900	600	.178	.006	.190	7,848,000
17	P-22)						
	R-2)	240-604	364	.504	.047	.601	5,278,000
18	P-31	260-830	570	.282	.060	.406	7,552,500
19	P-37	255-795	540	.391	.033	.459	6,037,200
21	R-8	645-875	230	.202	.005	.212	2,962,400
22	P-46	345-459	114	.129	.017	.164	1,398,780

LOW GRADE HOLES

LEACHED ZONE

HOLE NO.	DEPTH	LENGTH	% Cu.	% MOS_2	% Cu. Eq.	TONS
P-2 (50^o)	0-197	151	.071	.002	.075	1,580,366
P-3 (50^o)	0-131	100	.036	.003	.042	1,046,666
P-4 (50^o)	0-187	143	.032	.007	.046	1,496,638
P-25	0-94	94	.026	.007	.040	983,804
P-23	0-180	180	.053	.004	.061	941,940
P-48	0-110	110	.060	.020	.064	1,151,260
P-47	0-383	383	.005	.002	.009	4,008,478
P-15	0-450	450	.005	.002	.009	4,709,700
P-20	0-390	390	.011	Tr.	.011	4,081,740
P-18	0-210	210	.010	Tr.	.010	2,197,860
P-32	0-155	155	.029	.006	.041	1,622,230
P-27	0-72	72	.026	.007	.040	753,552
P-29	0-54	54	.026	.007	.040	565,164
P-30	0-63	63	.026	.007	.040	659,358
P-28	0-90	90	.026	.007	.040	941,940
P-40)						
R-6)	0-354	354	.015	.003	.021	2,559,420

ENRICHED ZONE

HOLE NO.	DEPTH	LENGTH	% Cu.	% MOS_2	% Cu. Eq.	TONS
P-2 (50^o)	197-624	327	.186	.023	.233	3,422,380
P-3 (50^o)	131-453	247	.170	.023	.217	2,585,102
P-4 (50^o)	187-414	174	.177	.019	.216	1,821,084
P-23	180-300	20	.332	.014	.361	209,320
P-18	210-540	330	.124	Tr.	.124	3,453,780
P-32	155-290	135	.220	.008	.236	1,412,910
P-29	54-125	71	.191	.003	.197	743,086

PROTORE

HOLE NO.	DEPTH	LENGTH	%Cu.	% MOS_2	% Cu. Eq.	TONS
P-25	94-355	261	.167	.017	.202	2,731,626
P-23	300-730	430	.209	.030	.271	2,250,190
P-48	110-650	540	.223	.018	.260	5,651,640
P-20	390-525	135	.109	.002	.113	1,412,910
P-18	540-725	185	.043	.001	.045	1,936,210
P-32	290-755	465	.031	.004	.039	4,866,690
P-27	72-645	573	.117	.003	.123	5,997,018
P-29	125-500	375	.058	.003	.064	3,924,750
P-30	63-155	92	.045	.003	.051	962,872
P-40)						
R-6)	354-552	198	.136	.005	.146	1,431,540

AREA "B"

LEACHED ZONE

HOLE NO.	DEPTH	LENGTH	% Cu	% MOS_2	% Cu. Eq.	TONS
P-9	0-55	55	Casing-no assays			576,015
P-7	0-31	31	Casing-no assays			324,663
	31-151	120	.052	.006	.064	1,256,760
P-11	0-19	19	Casing-no assays			198,987
	19-125	106	.040	.002	.044	1,110,138
R-4	0-205	205	.033	.012	.058	2,146,965
P-5 (50^{o})	0-28	28	Casing-no assays			225,170
	28-253	225	.068	.009	.087	1,801,356
P-24	0-15	15	Casing-no assays			157,095
	15-225	210	.027	.008	.044	2,199,330
P-21	0-94	94	Casing-no assays			984,462
	94-210	116	.068	.006	.078	1,214,868
Totals & Averages (excluding P-7, P-21)			.041	.0085	.059	

Tons (excluding P-7, P-21)	With Assays	7,257,789
	Without Assays	1,157,267
	Total	8,415,056

ENRICHED ZONE

HOLE NO.	DEPTH	LENGTH	% Cu	% MOS_2	% Cu. Eq.	TONS
P-9	55-220	165	.284	.007	.298	1,728,045
P-7	151-241	.90	.198	.013	.225	942,570
P-11	125-320	195	.383	.012	.408	2,042,235
R-4	205-255	50	.631	.044	.722	523,650
P-24	225-330	105	.339	.053	.448	1,099,665
P-21	210-371	167	.161	.004	.169	1,748,991
Totals & Averages (excluding P-7, P-21)			.366	.022	.411	5,393,595

AREA "B"

PROTORE ZONE

HOLE NO.	DEPTH	LENGTH	% Cu.	% MOS_2	% Cu. Eq.	TONS
P-9	220-521	301	.173	.008	.189	3,152,373
P-7	241,554	313	.116	.021	.159	3,278,049
P-11	320-455	135	.247	.009	.266	1,413,855
	455-557	102	.193	.014	.222	1,068,246
R-4	255-495	240	.308	.041	.392	2,513,520
	495-687	192	.095	.014	.124	2,010,816
P-5	253-660	407	.311	.023	.358	4,262,511
P-24	330-488	158	.136	.022	.181	1,654,734
Totals & Averages (excluding P-7, P-21)			.225	.020	.266	16,076,055
Average Grades of Enriched & Protore			.260	.0205	.302	21,469,650

POTENTIAL LOW GRADE ORE BELOW PIT FLOOR AND WALLS

HOLE	DEPTH	LENGTH	% Cu.	% MOS_2	% Cu. Eq.	TONS
P-30	155-445	290	.035	.001	.037	3,035,140
P-29	500-857	357	.154	.003	.160	3,736,362
P-20	530-707	177	.112	.002	.116	1,852,482
P-18	740-888	148	.036	.001	.038	1,548,968
P-32	740-1200	460	.010	.003	.016	4,814,360
P-22	800-1200	400	.322	.066	.458	4,186,400
P-37	800-948	148	.209	.018	.246	1,548,968
P-27	610-891	281	.159	.010	.161	2,940,946
P-15	450-1187	737	.162	.016	.195	7,713,442
P-35	900-1506	606	.069	.005	.079	6,342,396
P-31	810-1231	421	.249	.058	.368	4,406,186
P-48	630-800	170	.170	.012	.195	1,779,220
P-19	440-585	145	.260	.024	.309	1,517,570
P-16(75°)	1140-1418	270	.221	.039	.301	2,825,820
P-36	950-1200	250	.190	.019	.229	2,616,500
P-23	730,908	178	.250	.041	.334	1,862,948
P-44	790-900	110	.177	.031	.241	1,151,260
P-45	1070-1290	220	.095	.019	.134	2,302,520
P-25	320-1004	684	.158	.035	.230	7,158,744
P-24	225-488	263	.215	.034	.285	2,752,558
P-12(60°)	300-696	340	.125	.018	.162	3,558,440
P-26	[illegible]0-757	757	.152	.013	.179	7,922,762

20 March 1970
CJK/t

Mr. M.D. von Mueller
ERZGESELLSCHAFT mbH.

6000 Frankfurt am Main

Re: BRAMEDA Resources Ltd./
Casino Copper-Molybdenum Project

Dear Mr. von Mueller,

Herewith my report on the Casino project. The BRAMEDA staff were most helpful in giving any information requested, and in making available the BRENDA feasibility report.

In spite of the many obvious difficulties due to the remote location, I feel this project represents an important opportunity in a plitically stable area, and warrants the major expenditure required to reach a proper evaluation.

With best regards,

Yours sincerely,

Colin Knight

Colin J. Knight

cc: Ca, DrGö

(Dr. Frohberg)

R E P O R T

on

BRAMEDA RESOURCES

Casino Copper Molybdenum Project

Yukon Territory, Canada

by

Dr. Colin J. Knight

Toronto,
20 March 1970

SUMMARY AND RECOMMENDATIONS

1. Drilling is too widely spaced to permit definite conclusions. Tonnage and grade calculations are very sensitive to one or two higher grade drill holes. Further drilling may change results significantly, positively or negatively.

2. However, making major assumption as to continuity of grade and dimension between widely spaced drill holes, 430 million tons might be inferred capable of supporting a viable operation of 50,000 to 60,000 tons per day for a life of 20 years. The potential is believed to exist for the larger tonnages necessary to support 75,000 to 100,000 t.p.d. operations.

3. No definite decision can be reached without expenditure of 4 to 5 million dollars necessary to complete and assess detailed drilling, bulk sampling and pilot mill operation.

RECOMMENDATIONS

1. It is essential that detailed drilling at 400 ft. centres or less should continue, with bulk sampling to check drill results, and provide pilot mill feed. As metallurgy is as critical as grade, pilot mill tests should not be delayed.

2. We should regard the $4 to 5 million cost of such a programme as the necessary price of information on which to base a final decision on a project of very large potential.

3. A visit should be made to the Casino property as soon as suitable weather conditions permit.

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BRAMEDA RESOURCES Ltd.

Casino Copper Molybdenum Project, Dawson Range, Yukon

INTRODUCTION

1. The Casino Silver Mines Ltd. deposit is a "porphyry copper"-type copper molybdenum deposit of large tonnage potential in Western Yukon. The agreement with Casino Silver Mines Ltd. provides in effect that Brameda Resources Ltd. will acquire 70% interest in the Casino property if more than $ 10,000,000 is expended in bringing it to production.

2. During Metallgesellschaft/Brameda meeting in Vancouver on March 1 and 2, the possibility of MG participation in the Casino project was discussed.

3. Brameda proposed that MG take a share interest in Brameda at a cost of about $ 4,250,000. The share interest would carry the right to earn a direct ownership interest in the Casino project by expending exploration funds to complete the feasibility studies and place the Casino property into production. MG could thus earn up to 50% of Brameda's interest in the Casino project (i.e.50% of 70% of the total project), in three stages:

   a) Completion of feasibility studies
   b) If the project is feasible, provision of half the debt financing for optimum mining and milling operation
   c) If found to be feasible, construction of optimum blister copper smelter.

(Further details are given in the Memorandum of Discussion, March 2, 1970, Petrenko/Austin).

4. The purpose of this report is to provide a preliminary technical assessment of the Casino project as a basis for further discussion.

5. It must be emphasized that large uncertainties are involved in attempting an assessment of a very large project at such an early stage of development, and many assumptions have been made.

UNITS EMPLOYED

Copper price	-	4 LME expressed in U.S.funds
Tons	-	short tons
Costs, cash flow,etc.	-	Canadian funds (US$ 1=Can.$1.075

LOCATION OF CASINO PROPERTY

Latitude	62°44'N
Longitude	138°32'W

Approximately 100 miles by winter roads from Carmacks, about 75 miles east of the Yukon/Alaska border, at an elevation of 3,000 to 4,000 feet above sea level, in the Dawson Range (Fig.1).

ACCESS

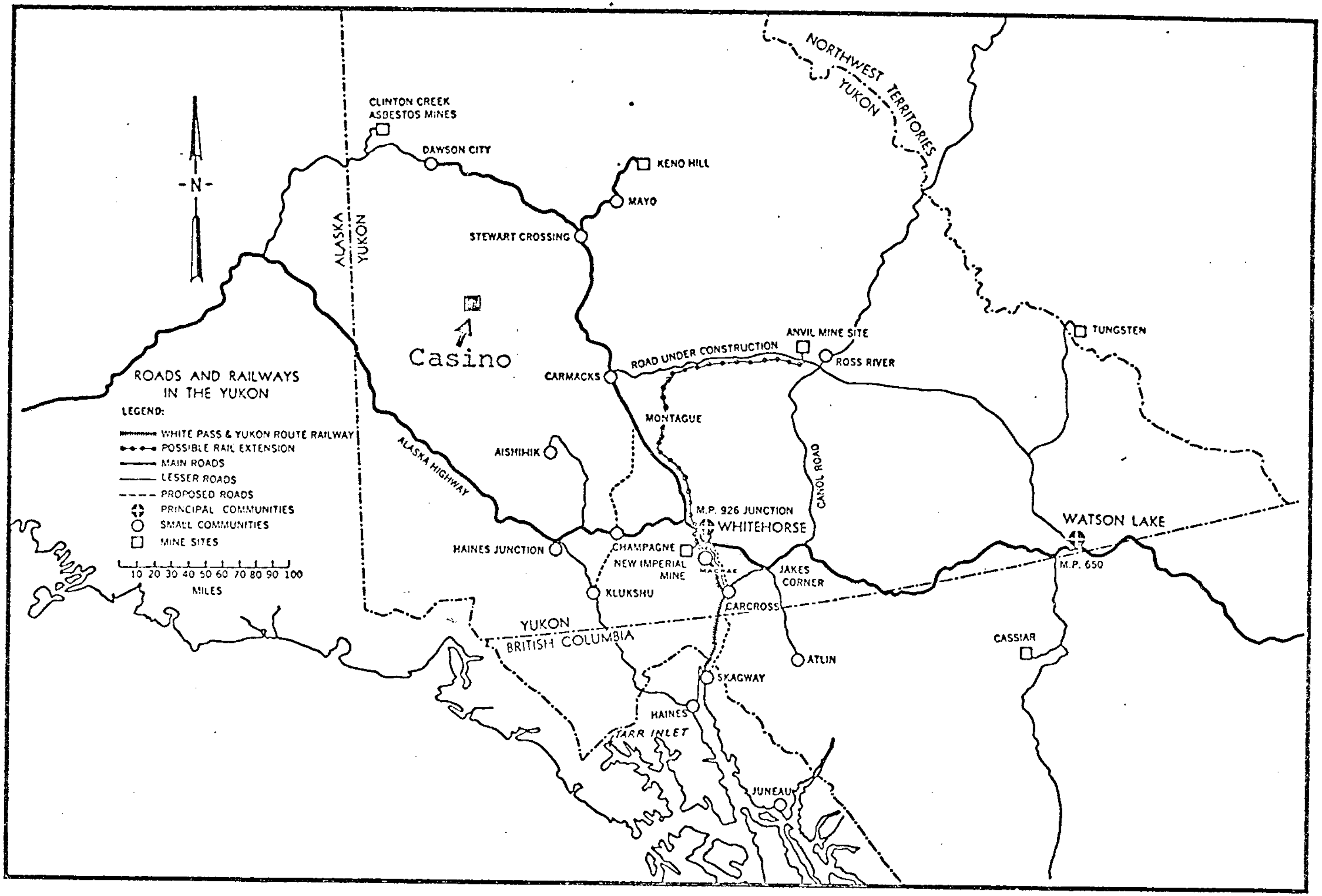
At present, by winter road from Carmacks. Airstrip for light aircraft on the property. Assuming that existing winter roads are subsequently improved; the road distance from Casino via Carmacks and Whitehorse to Skagway will be 340 - 350 miles.

GENERAL GEOLOGY

Casino lies within the Dawson Range belt of Triassic/ Jurassic intrusive and volcanic rocks.
The Casino mineralisation is associated with a Tertiary stock intruded into the Klotassin granodiorite. The stock is a complex intrusive with characterics commonly associated with "porphyry copper" deposits with a typical alteration zone and certain possible sub-volcanic features such as breccia pipes.(Fig.2)

DIMENSIONS OF MINERALISATION

Geochemical and geophysical results, and initial drilling results indicate that Cu and Mo mineralisation is present over an area about 6,500 x 3,500 ft.



ROADS AND RAILWAYS IN THE YUKON
LEGEND:
WHITE PASS & YUKON ROUTE RAILWAY
POSSIBLE RAIL EXTENSION
MAIN ROADS
LESSER ROADS
PROPOSED ROADS
PRINCIPAL COMMUNITIES
SMALL COMMUNITIES
MINE SITES
10 20 30 40 50 60 70 80 90 100
MILES
N
CLINTON CREEK ASBESTOS MINES
DAWSON CITY
KENO HILL
MAYO
STEWART CROSSING
ALASKA
YUKON
NORTHWEST TERRITORIES
YUKON
Casino
CARMACKS
ROAD UNDER CONSTRUCTION
ANVIL MINE SITE
ROSS RIVER
TUNGSTEN
MONTAGUE
AISHIHIK
ALASKA HIGHWAY
CANOL ROAD
M.P. 926 JUNCTION
WHITEHORSE
WATSON LAKE
M.P. 650
HAINES JUNCTION
CHAMPAGNE
NEW IMPERIAL MINE
JAKES CORNER
KLUKSHU
CARCROSS
YUKON
BRITISH COLUMBIA
CASSIAR
ATLIN
SKAGWAY
HAINES
TARR INLET
JUNEAU

Fig. 1

A zone of oxidation is present to an average depth of about 200 ft. It contains low values in Cu and Mo chiefly as oxides, and is presently regarded as waste. It will presumably be stockpiled for possible leaching experiments in the future.

The zone of secondary enrichment below is also about 200 ft. in average thickness, in which copper is mainly present as chalcocite.

The zone of primary sulphides carrying chalcopyrite and molybdenite extends for an unknown depth below the zone of secondary enrichment. A number of holes have been drilled into the primary zone for thicknesses of 500 to 1,000 feet, without any apparent marked decrease of grade within the primary sulphide zone.

DIAMOND DRILLING

About 49 diamond drill holes have been drilled ('BQ'wireline), about 25 of these at 600 to 1,000 ft. spacing, the rest are at 400 ft.spacing. Closer spacing is planned for holes in the critical higher grade section of the deposit. Core recovery was 90 - 93%; there was no return of drilling water, and so no sludge samples were taken. There is thus a possibility that a small portion of molybdenite values have been lost. A number of large diameter "down-the-hole" percussion drill holes are now in progress. In this case all the cuttings are blown out of the hole, providing a better sample than diamond drilling.

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ASSAYING

Three laboratories have been used:"Loring", "Coast-Eldrige", and"Seymour" all reportedly used atomic absorption techniques.

Seymour assays, which are used as the basis of "Inventory of Mineralisation" calculations, are consistently lower than "Loring" or "Coast-Eldridge" assays; particularly with respect to molybdenum.

RESERVES

(Brameda's consultants Chapman,Wood & Griswold refer to "Inventory of Mineralisation" rather than "Ore Reserves", in acknowledgment of the uncertainties involved at this early stage of investigation.)

Reserves have been calculated in two ways: -

1) By computer:"Trend Surface Analysis Technique"
2) By hand, conventional polygonal block method.

1. Computer "Trend Surface Analysis Technique"

The computer method divides the mineralised zone into 100 ft.cubic blocks; assigns the "most probable" grade to each block on the basis of a statistical treatment of variations of grade in three dimensions; and compiles total tonnage within a series of grade categories, also compiles tonnages above a series of cut-off grades.

While this technique is logical and consistent, there is a capacity to predict and use extrapolated grades which are higher than the highest real assay obtained in drilling. Because the calculations are based on relatively few and widely spaced drill holes, the statistics are correspondingly unreliable. For this reason I believe the computer result in tonnage and metal content is positively biased in comparison with the conventional "Polygonal Block" technique.

2. Hand Calculation, "Polygonal Block" Technique
Reserves are compiled within the two classes of mineralisation: Primary and Secondary.
Within each class the average grade and depth of drill holes is assigned to polygonal blocks extending 400 ft. or less from the drill hole at the center of each block (Category "A"). Category "B" constitutes reserves lying between blocks so defined as well as mineralisation extending 200 ft. horizontally beyond the polygonal blocks on the perimeter of the mineralized zone.

This calculation has similarly been based on results of relatively widely spaced drilling, and the resulting average grade is very sensitive to the effect of drill hole number P-22.

(P-22 block: -

Secondary 10,669,000 tons @ 1.019% Cu, 0.402% MoS_2
Primary 36,524,000 tons @ 0.402% Cu, 0.061% MoS_2

Several holes have been completed in the vicinity of P-22, since the above calculations were made. Of these, assays for P-31 are available. Although the grade of

P-31 is above average grade, it reduces the effect of P-22. Accordingly, Category"A" reserves have been recalculated, using P-31, to give "Revised Category A-Reserves". Table I summarizes "Computer reserves", "Polygonal Block, Category A and B Reserves", and "Revised Category A Reserves".

Table I

1. Computer Reserves

"Total above 0.20% Cu equivalent"

Tons	Average Grade
663,497,340	0.37% Copper Equivalent

2. Polygonal Block (assuming pit cut-off grade 0.19% Cu Equivalent)

	Tons	%Cu	$\%MoS_2$	%Copper Equivalent
Category A				
Secondary	153,977,000	0.373	0.027	0.429
Primary	278,018,000	0.219	0.043	0.309
Total A	431,995,000	0.274	0.037	0.352
Category B				
Secondary	63,846,000	0.251	0.017	0.287
Primary	79,152,000	0.174	0.027	0.230
Total B	142,998,000	0.208	0.022	0.255
Total A+B	574,993,000	0.257	0.033	0.327

3. "Revised Category A"

	Tons	%Cu	$\%MoS_2$	%Copper Equivalent
Secondary	152,395,000	0.365	0.0277	0.422
Primary	275,538,000	0.213	0.0420	0.300
Total	427,933,000	0.267	0.0369	0.340

For the purpose of this study the "Revised Category A Reserves" are regarded as "drill indicated reserve" for the basis of calculations. This involves the major assumption of continuity between widely spaced holes. The bulk of "Category B" reserves are considered to be too uncertain at present to be used as a basis of further calculations.

The higher tonnages and grades indicated by the computer results and by the Polygonal Block Category A and Category B should be regarded as including "potential reserves" which may subsequently be confirmed as development drilling continues.

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ANALYSIS

In a large low grade deposit of this type there obviously exists a range of reserve versus cut-off grade situations to be assessed for feasibility.

For the present purpose time permits only a simple study. Therefore, the calculations are based on the following reserves:

1. Secondary	152,395,000 tons	0.365% Cu	0.0277% MoS_2
2. Primary	275,538,000 tons	0.213% Cu	0.042 % MoS_2
Total	427,933,000 tons	0.267% Cu	0.0369% MoS_2

It is assumed that all of (1) could be mined before mining from (2) although this sequence might not correspond to the optimum pit design.

FACTORS: apart from grade and tonnage

1. Metallurgy
2. Smelter contract
3. Capital costs
4. Operating costs
5. Transportation
6. Off-site costs
7. Life
8. Taxation

Many factors can be approximately assessed by analogy with figures from the detailed feasibility study of Brenda Mines, which was placed at my disposal.

METALLURGY

Some bench-scale tests have been done on Casino drill core. While preliminary results show that recoveries of 90 to 93% can be achieved for Cu and MoS_2 in some cases, these recoveries have apparently been obtained at the expense of concentrate grade.

The recoveries used in the Brenda feasibility study were

Cu	88%
MoS_2	82%.

The Casino molybdenum grade is less than half that of Brenda, and it is assumed that Casino molybdenum recovery may be somewhat lower. Therefore, in this Casino study, the following recoveries are used:

Cu	88%
MoS_2	80%,

under the alternative conditions: -

a) Assuming that copper concentrate of grade of 26% Cu can be obtained for both primary and secondary ore

b) Assuming primary concentrate 26% Cu, but secondary concentrate 19%. (Casino bench tests indicate some problems in upgrading secondary concentrates.)

SMELTER TERMS (F.O.B.)

The following basis of payment is assumed: -

Pay (contained copper less one unit) @ 4 LME

Charges: Smelting US$ 15 per dry short ton

Refining US¢ 2.5 per lb.copper paid for.

Haulage from Casino to ship at Skagway assumed as for Anvil; at Can.$ 18.00 per short ton.

Note: Costs beyond Skagway have not been included.

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Table II

NET SMELTER RETURN PER SHORT TON MILLED

Price for Copper US-cents	35	40	45	50	55	60	65
	Can.$	Can.$	Can.$	Can.$	Can.$	Can.$	
a) Secondary Ore Concentrate 26%Cu, incl.payment for molybdenum concentrate Can.$ 1.72/lb. at mine	2.20	2.53	2.86	3.19	3.52	3.86	4.19
b) Secondary Ore Concentrate 19%Cu, incl.payment for molybdenum concentrate	2.01	2.34	2.66	2.99	3.32	3.64	3.95
c) Primary Ore Concentrate 26%Cu, incl.payment for molybdenum concentrate	1.70	1.89	2.09	2.28	2.48	2.67	2.87

(Payment for precious metal content of concentrates has not been considered.)

CAPITAL COSTS

The following capital costs of bringing an operation to production at Casino are assumed, including off-site costs.

Size of Mill	Can.$
20,000 t.p.d.	66,000,000 (66 million)
30,000 t.p.d.	87,000,000
40,000 t.p.d.	102,900,000
50,000 t.p.d.	112,500,000
60,000 t.p.d.	127,500,000
75,000 t.p.d.	150,000,000
100,000 t.p.d.	183,000,000

The 20,000 t.p.d. capital cost has been taken at Brenda costs + 10%; costs for larger plants have been extrapolated.

OPERATING COSTS

It is assumed that operating costs at Casino will be "Brenda cost + 5%".

It is assumed that the overall stripping ratio to mine/ 430 million ton reserve is 1.4 to 1, and that this ratio is constant throughout the life of the pit.

Brenda Costs (20,000 t.p.d.)	Brenda		Casino
Mine 20¢ per ton rock moved = 2.4 x 20	0.480		
Mill	0.855		
Overhead Administration	0.220		
	$ 1.56	+ 5% =	$ 1.64
	per ton milled		

Plant Size t.p.d.	Casino Operating Cost Can.$/ton milled
20,000	1.64
30,000	1.55
40,000	1.49
50,000	1.46
60,000	1.42
75,000	1.38
100,000	1.33

TRANSPORTATION

It is assumed that cost of shipping concentrate to Skagway, including handling to ship, will be the same as for Anvil (Can.$ 18.00/d.s.t.).
Anvil is almost exactly the same distance from Skagway. The road distance to Carmacks is about 100 miles./ Approximately 100 miles of access road is required from Carmacks to Casino. This would be provided by improving existing winter roads to "permanent access road" standards, and/or construction of a new road, for which Federal Government assistance could be provided up to 2/3 cost of construction, but not more than 15% of the investment in the operation prior to production. It is also possible that a new road joining Carmacks to the Alaska Highway would be routed so as to reduce the length of the Casino access road.

OFF-SITE REQUIREMENTS

WATER SUPPLY

Situation - similar to Brenda; the alternatives are: construction of 12 miles of pipeline or construct a dam near the minesite.

HYDRO-ELECTRIC POWER

Federal Government is likely to assume the capital burden; the cost to mine would therefore be based on consumption.

TOWNSITE

It is likely that Federal Government would provide serviced townsite, with lots to be sold to the company to build houses.

LIFE

It is apparent that a short-lived operation would be feasible in theory, mining only a relatively small high-grade portion of reserves. It is quite clear, however, that as an operation at Casino will be strongly dependent on Federal assistance, such assistance would not be made to support a "high grading" operation resulting in loss of low grade resources. Assistance for road, railway, townsite construction etc. are likely to be related to a minimum life of 15 to 20 years.

ASSESSMENT

A series of quick calculations were made of the time to return loan capital ("pay back time"), at 8 1/2 % compound interest, disregarding taxes, for 20, 30, 40, 50, 60, 75, and 100,000 ton per day operation.

Variables considered were:

Copper price from 35¢ by increments of 5¢ to 60¢ (US) per lb

a) Assuming concentrate grade of 26% Cu,

b) Assuming concentrate grade of 19% Cu.

RESULTS

1.) At copper prices of 55 to 60¢ "pay back" time is 6 years or less at 20,000 t.p.d. Under present scheme of taxation, such short pay back times will not be much affected by taxes.

2.) The effect of reducing concentrate grade from 26% to 19% is about the same as a 2.5¢/lb decrease in copper price.

3.) An increase in cost of haulage from Casino to ship at Skagway from $18.oo to $32.oo/s.t. similarly has the effect of a 2.5¢ decrease in copper price.

4.) At prices of 40 to 45¢/lb, an operation on the scale of 50,000 to 60,000 t.p.d. is indicated for "pay back" within ten years.

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METALL CRITERIA FOR OPERATIONS AT CASINO

Although as the Brameda proposal now stands Metall can earn an important equity position in Casino, the Casino project should perhaps be regarded largely as senior financing of a major, long-term source of copper concentrate or blister copper. In such a case a fast return of loan capital is of less immediate concern than a long life with a low degree of uncertainty.

It is suggested that the criteria for a Casino operation should be "pay back" within 10 years at copper prices not exceeding 45¢ (4 LME).

Accordingly, cash flow calculations were made for a 60,000 t.p.d. operation.

RESULTS OF CASH FLOW CALCULATIONS

		$
60,000 t.p.d.	Capital required	127,500,000.oo
	Equity assumed	6,000,000.oo
	Loan capital	121,500,000.oo

Depreciable Assets $91,000,000 (30% declining balance)
Preproduction Expenses $20,000,000.
Three year tax holiday, tax rate 39% of taxable income.

A. Price of copper 40¢ (US)/lb, copper concentrate grade 26% Cu.

1. Secondary ore of about 151 million tons is depleted in the 8th year of operations.
2. Primary ore of about 278 million tons is mined thereafter, at reduced net smelter return.

3. Loan capital is returned with interest at 8 1/2%, in 9 1/2 years of operation.

4. Thereafter net cash flow after taxation is about $ 6 million per year for 10 1/2 years.

5. Life: 20 years, based on total reserve of 430 million tons.

6. Production:

Years	1 - 7	67,450 tons Cu. 5,584,000 lbs Mo.
Year	8	44,713 tons Cu. 7,918,000 lbs Mo.
Years	9 - 20	39,362 tons Cu. 8,467,200 lbs Mo.

B. Price of copper 45¢, copper concentrate 26%

Loan capital is returned in about 5 1/2 years, thereafter net cash flow after tax is about $10.5 million per year for 14 1/2 years.

C. Price of copper 60¢, copper concentrate 26%

Loan capital returned in 3 years, thereafter net cash flow about $22.6 million for 17 years.

D. Price of copper 45¢, secondary ore concentrate grade 19% Cu.

Loan capital returned in about 6 1/2 years, thereafter net cash flow after tax is about $9.4 million for 13 1/2 years.

CONCLUSIONS (Note that these conclusions have been reached without benefit of a visit to the Casino project)

1. Diamond drilling completed so far is much too widely spaced to allow any definite conclusions as to reserves.

2. However, if the major assumption is made that there is continuity of grade and dimensions of mineralisation between widely spaced sampling points, 430 million tons might be considered drill indicated, at 0.267% Cu, 0.0369% MoS_2.

3. The grade quoted is very sensitive to the effect of one very high grade hole (P-22), and the result of a number of close-spaced holes to test the extent of the high grade zone will be critical. The nature of preliminary drilling so far is such that additional drilling may substantially increase reserves and grade, or on the other hand, it may substantially reduce grade and reserves.

4. It is not clear, at the moment, whether there is any structural control for the high grade mineralisation.

5. If subsequent drilling and bulk sampling confirms 430 million tons at the above grades, there appears to be a good chance that a 50 to 60,000 t.p.d. operation would be viable at copper prices (4 LME) 40 to 45¢ US, shipping concentrates via Skagway, with a life of 20 years.

6. The "computer" estimate of 664 million tons above 0.2% copper equivalent may be taken as some indication of potential. For example the Western portion of the mineralised zone has not been drilled to depth. There is a fair chance that further detailed drilling may indicate a total tonnage of the order of 700 million to 1 billion tons necessary to support a 75,000 to 100,000 t.p.d. operation.

7. However, it must be reiterated, that while tonnages of several hundreds of millions of tons are reasonably well assured, at this early stage of exploration/ development drilling a large uncertainty is still attached to the grade of these tonnages.

   It is therefore essential that detailed development drilling should continue particularly to delimit accurately the dimensions of higher grade sections.

8. It would be possible to complete the greater part of required drilling by the end of 1970 field season, and to delay until 1971 bulk sampling and pilot mill tests.

9. However, such a delay is not considered desirable or practical, because it is important that bulk sampling be undertaken as soon as possible to provide a calibration of drilling results.

10. It is apparent that metallurgy is as critical as grade, it thus is very important that pilot mill tests proceed more or less concurrently with drilling.

11. We have a particular interest in participating in a smelter operation. However, we should keep in mind that other "porphyry copper" discoveries are quite likely in the Dawson range, so that a smelter situated at Casino might not necessarily be dependent only on the Casino operation for concentrates.

12. The 60,000 t.p.d. operation was assessed on the basis of shipping concentrates via Skagway. On the assumption that cost per ton of shipping blister copper to Skagway is about the same as for concentrates ($18.oo) an on-site smelter would offer an advantage to the mine of 10 to 16¢ per ton milled. (This does not take into account costs beyond loading to ship at Skagway.)

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SUMMARY AND RECOMMENDATIONS

1. Drilling is too widely spaced to permit definite conclusions. Tonnage and grade calculations are very sensitive to one or two higher grade drill holes. Further drilling may change results significantly, positively or negatively.

2. However, making major assumption as to continuity of grade and dimension between widely spaced drill holes, 430 million tons might be inferred capable of supporting a viable operation of 50,000 to 60,000 tons per day for a life of 20 years. The potential is believed to exist for the larger tonnages necessary to support 75,000 to 100,000 t.p.d. operations.

3. No definite decision can be reached without expenditure of 4 to 5 million dollars necessary to complete and assess detailed drilling, bulk sampling and pilot mill operation.

RECOMMENDATIONS

1. It is essential that detailed drilling at 400 ft centres or less should continue, with bulk sampling to check drill results, and provide pilot mill feed. As metallurgy is as critical as grade, pilot mill tests should not be delayed.

2. We should regard the $4 to 5 million cost of such a programme as the necessary price of information on which to base a final decision on a project of very large potential.

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3. A visit should be made to the Casino property as soon as suitable weather conditions permit.

Toronto, 20 March 1970
CJK/t

CJK

APPENDIX

1. Summary of "Computer" tonnage and grade calculations.
2. "Polygonal Block" Inventory of Mineralisation.
3. Casino development programme 1970, with Sketch map showing proposed adit development.

New File
Budgets & Estimates

BRAMEDA RESOURCES LTD.

CASINO CREEK DEVELOPMENT PROGRAM

JANUARY 1, 1970 TO DECEMBER 31, 1970.

FORECAST OF TOTAL COSTS AS ITEMIZED BELOW

(Note: This does not include inventory through outright acquisition of property and assets thereon transacted in the summer of 1969. Other costs included and incurred preceding the period outlined are noted.)

MONTHLY EXPENDITURES

		JAN.	FEB.	MAR.	APRIL	MAY	JUNE	JULY	AUG.	SEPT.	OCT.	NOV.	DEC.
1. Fuels and Lubricants - all diesel, stove, propane, gasoline, and lubricants. (Includes cost of bulk transportation to site).	$ 210,000	12,000	19,000	167,000	1,500	1,500	1,500	1,500	1,500	1,500	1,500	1,500	
2. Air transportation between Whitehorse and site - (Includes the limited air freight)	175,000	10,000	10,000	12,000	14,000	20,000	20,800	20,800	20,800	20,800	15,800	10,000	
3. Winter Road Maintenance. Burwash Landing to Casino site. - (Includes estimated cost of Bailey bridge removal and transportation to Whitehorse, equipment rentals and operation (grader and D-8), and dispatch control costs).	52,000	15,000	16,000	21,000									
4. Airfield and Property Area Road Maintenance - grader rental, - labour and materials.	25,000		1,000	2,000	2,000	2,500	3,000	3,000	3,000	3,000	3,000	2,500	
5. Bulldozer (D-8) rental and operation on the property. - Rental, operator, and maintenance.	84,500	7,000	7,500	7,700	8,400	7,700	7,700	7,700	7,700	7,700	7,700	7,700	
6. Transportation Units on Site - trucks, pickups and bus. - Rentals, repairs, labour for operation.	37,000	2,500	2,500	3,000	3,000	3,500	3,750	3,950	3,750	3,750	3,750	3,550	
- Units purchased for same in November and December 1969	12,613	12,613											
7. Camp Operation - All catering costs, camp and equipment maintenance, and direct supervision	310,000	13,000	19,000	28,000	30,000	32,000	34,000	34,000	34,000	34,000	34,000	18,000	
- Generators - transportation, rental and maintenance	18,000	1,000	2,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	
- Pumps and other light equipment - purchases and maintenance	12,000	2,000	2,000	2,000	750	750	750	750	750	750	750	750	

	Total	JAN.	FEB.	Mar	Apr.	May	June	July	Aug.	Sep	Oct	Nov	Dec
8. Trailer dorms, wash trailers, kitchen-diner trailer and directly related equipment - 116 man camp													
- Dorms purchased (allows $6200 credit due from Churchill Copper for 2 trailer dorms)	43,900	43,900											
- Kitchen-Diner	18,500	18,500											
- Transportation and erection of all trailers	36,000	36,000											
(Note: Item 8 purchases and installations cover the November and December 1969 and January 1970 period).													
9. Exploration Diamond Drilling and Rotary Drilling													
(Allowing a maximum of 90,000' for 1970. This includes drilling for testing out plant site, tailing disposal, waste and stockpile areas).													
- Direct payments to contractors	1,000,000	46,000	108,000	108,000	108,000	108,000	108,000	108,000	108,000	108,000	90,000		
- Field supervision (geologist)	16,000		1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	
10. Bulk Sampling													
(a) Underground Development - payment to contractors plus cost of materials and related rentals (To commence on April 1, 1970 and terminate by September 30, 1970). About 3500 ft Dr & X-C, [illegible]	440,000			40,000	60,000	60,000	70,000	80,000	80,000	50,000			
(b) Construction and Maintenance of Mine Run and crushed product storage bins	50,000			20,000	20,000				10,000				
(c) Crushing Plant operation, maintenance, rentals and transportation of equipment costs, (5-month operation allowance),	130,000			1,000	14,000	18,000	20,000	20,000	21,000	21,000	1,000	14,000	
(d) Sample tower operation and maintenance (5-month operation allowance.	9,000					1,800	1,800	1,800	1,800	1,800			
(e) Material Handling - rental of 2 front-end loaders, operation and maintenance.	110,000	6,000	8,000	9,000	10,000	12,400	12,400	12,400	12,400	12,400	8,000	7,000	
(f) Direct Supervision, surveying and mapping	27,000		2,000	2,400	2,800	3,000	3,000	3,000	3,000	3,000	2,800	2,000	
11. Brenda complete pilot mill and sampling tower													
- Estimated purchase price	60,000		60,000										
- Dismantling, transportation and erection at Casino site	230,000		60,000	85,000	85,000								
- Grinding mill liner replacement plus additional equipment and alterations	35,000		10,000	25,000									
12. Pilot Mill Operation													
-(6-month operation allowance). Includes reagents, grinding media, maintenance, generator rental, labour, and direct supervision and staff	180,000						30,000	30,000	30,000	30,000	30,000	30,000	
12A. All Equipment Maintenance, Tools [illegible]	8,400	1,000	3,000	2,000	400	400	400	400	400	400			

		JAN.	FEB.	MAR.	APR.	MAY	JUNE	JULY	AUG.	SEPT.	OCT.	NOV.	DEC.
- Plant Fresh Water Supply	28,000			3,000			5,000	5,000	5,000	5,000	5,000		
- Main Access Road to Mine	30,000							7,500	7,500	7,500	7,500		
- Tailings Disposal	10,000								3,500	3,500	3,000		
- Townsite	20,000								7,500	7,500	5,000		
	4,330,563	293,548	438,125	584,497	400,885	331,642	425,150	454,630	468,846	423,055	311,630	178,380	20,175

(NOTE: (1) The forecast for December shows only the Whitehorse administration office rental and related service charges, and the retainer for C. W. & G.s services.

(2) The forecast includes transportation-out charges for the crushing plant and rotary drill.

(3) Should it be decided to discontinue activities on the property during the winter of '70-'71, it will require an additional outlay of approximately $75,000, to re-open the winter road from Burwash Landing during the latter part of November and early December and return leased and rental equipment to suppliers.)

		JAN.	FEB.	Mar	Apr	May	June	July	Aug	Sep	Oct	Nov	Dec
13. Assay Laboratory and Sample Preparation													
- All trailers, equipment, transportation and setting up at site.	46,000		46,000										
- Operation (10-month allowance). Includes labour, related expendables, generator rentals, and maintenance of equipment.	75,000		3,000	6,000	7,000	7,000	9,000	10,000	10,000	10,000	8,000	5,000	
- Additional outside assaying and transportation costs.	40,000	5,000	12,000	3,000	2,000	2,000	3,000	3,000	3,000	3,000	2,000	2,000	
14. Warehouse and First Aid													
- Building and shelving	9,000		9,000										
- Operation - labour and first-aid supplies (11-month allowance period)	13,000	1,200	1,200	1,800	1,200	1,100	1,100	1,100	1,100	1,100	1,100	1,000	
15. Service Building (Constructed on site).	16,000	14,000	2,000										
16. Office Building, core shacks, etc., constructed on site (Built in December)	12,000	12,000											
17. Dry change room - constructed on site.	6,000	6,000											
18. Direct other labour costs for general maintenance and field services.	95,000	9,000	9,000	9,000	9,000	9,000	9,000	9,000	9,000	9,000	9,000	5,000	
19. Whitehorse Office													
- Renovations	1,050		1,050										
- Office equipment	2,500		1,800	700									
- Space rental, janitor and light	7,000		500	650	650	650	650	650	650	650	650	650	650
- Staff wages	15,600		1,200	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	1,600	
- Office equipment rental plus stationery.	2,000		300	260	180	180	180	180	180	180	180	180	
20. Communications Installations	25,000						7,000	6,000	6,000	6,000			
21. Geological Consultant and other related services (Archer, Cathro & Associates, including services of M. Phillips)	50,000	8,000	7,000	5,000	5,000	5,000	5,000	5,000	5,000	3,000	2,000		
22. C. W. & G. Consulting and services	143,000	7,335	2,875	9,187	11,205	6,362	13,320	15,100	12,516	13,725	15,100	16,750	19,525
23. Other Consultant and Engineering Services													
- Pilot Mill Metallurgical	12,000	3,000	5,000	4,000									
- Pilot Mill redesign	7,500	1,500	4,000	2,000									
- Large Mill Metallurgical *	60,000					4,000	10,000	10,000	10,000	10,000	10,000	6,000	
- Large Mill and Plant Design *	200,000					15,000	30,000	35,000	30,000	34,000	30,000	30,000	
- Smelter *	75,000					5,000	10,000	15,000	15,000	10,000	10,000	10,000	