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1515 MINERAL SQUARE
SALT LAKE CITY, UTAH

S. D. MICHAELSON
CHIEF ENGINEER

ADDRESS REPLY TO
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RECEIVED
Scientific and Engineering Computer Center
11-26-69
WEISS
M.T.P.

November 25, 1969

Mr. Alfred Weiss, Director
Scientific and Engineering Computer Center
1356 Kennecott Building
Salt Lake City, Utah 84111

Dear Mr. Weiss:

Mr. C. T. Penney has requested that we work with him on developing long-range mining plans for Alice Arm. I understand the Computer Center has recently completed checking the ore reserve done by Mr. Eldon Bray, and has on file the assay values for 25 ft x 25 ft x 35 ft "mining blocks."

To carry the mining plan work further, as requested of us by Mr. Penney, we now need from you a schedule and price quotation for the following computer work:

1. Updating the Alice Arm file to include the five new diamond drill holes.

2. Re-compiling the 25 ft x 25 ft "mining blocks" to the standard 100 ft x 100 ft blocks, if this will save future computation time.

3. Simulation of four pits (Attachment 1) at pit limit cutoff grades of 0.24% MoS_2, 0.20% MoS_2, 0.16% MoS_2, and 0.10% MoS_2, with variable strip ratios in each case.

4. Ore inventory summaries of each pit (Attachment 1).

The above, if authorized, will be used along with our engineering cost studies as support for a possible expansion at B. C. Moly, and the timing of any expansion proposal will be contingent on doing the above work at an early date. Therefore, we would like to receive your cost and scheduling proposal as soon as you can let us have it, so a decision can be made on how best to proceed with this part of the expansion study.

Yours very truly,

S D Michaelson

SDM:djp
Attachment
cc: Mr. C. T. Penney
Mr. J. L. Halls

103P/06

KENNECOTT COPPER CORPORATION
METAL MINING DIVISION - ENGINEERING DEPARTMENT
1515 MINERAL SQUARE
SALT LAKE CITY, UTAH

ADDRESS REPLY TO:
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SALT LAKE CITY, UTAH 84111

INTEROFFICE MEMORANDUM

December 1, 1969

Mr. C. T. Penney, General Manager
B. C. Molybdenum Ltd.

Subject: British Columbia Molybdenum Limited - Ore Reserves

At the oral request of Mr. C. T. Penney, General Manager, B. C. Moly, I visited the property from November 5th through 11th to review B. C. Moly's latest ore reserve estimates. These ore reserves were prepared as a preliminary to determine whether an expansion of the operation would improve its economic prospects.

Summary

The ore reserves have been re-estimated both manually and by computer to incorporate recent drilling results and using costs which the mine estimates would be attainable on a 12,800 TPD operation. A break-even pit limit increment was determined using a pit limit cut-off grade of 0.113% MoS_2, variable stripping ratios and a stripping cost of $0.32 per ton waste. A pit limit slope of 45° was used. Close agreement was reached between the manually estimated and computerized ore reserves because the same distribution of values based on DDH assay results and geological predictions was used in both cases. The computerized ore reserves are:

> 119,419,326 tons at a grade of 0.181% MoS_2 with an overall strip ratio of 1.48:1 and a milling cut-off grade of 0.113% MoS_2. Within this pit there are 82,656,000 tons with a grade of 0.207% MoS_2 and a strip ratio of 2.6:1, using a milling cut-off grade of 0.16% MoS_2.

I checked the ore reserves by reviewing the level maps and sections and by manually estimating the stripping ratios at the pit limits on six sections and found that, with the parameters used, they are valid.

OPINION

I am concerned about the inclusion of the deep ore (amounting to roughly 20,000,000 tons below the 1360' level), in the positive ore reserves for the following reasons:

1. The stripping cost of $0.32 per ton used in the determination of the pit limit increment is, in my opinion, too low in light of the current mining and stripping costs of $0.43/ton.

2. Inclined holes have shown that grade cut-offs within the ore body are sharp and are vertical. These inclined holes are relatively shallow and although deep drilling has indicated that the ore body extends to depth, the distribution of values and the outline of the ore body, particularly below the 1360' level, have had to be determined mainly by geological interpretation. Deep and sparsely drilled ore reserves of this nature should be called "probable" ore.

The pit limit cut-off grade was derived from the mine's estimate of what the operating costs would be for a 12,800 TPD operation. Even if the parameters used were valid, the ore reserves would not be valid until approval has been given for the expenditure of funds on an expansion. For reporting purposes for January 1970, it is recommended that the January 1969 ore reserves, amended to take into account mining during the past year, be used. These ore reserves (1/1/69) were:

	Tons	% MoS_2
Positive Ore, 0.16% MoS_2 Cut-off	37,792,000	0.23
Probable Ore, 0.10% MoS_2 Cut-off	36,000,000	0.15
Total Ore, 1/1/69	73,792,000	0.19

Ore reserves estimated by using a pit limit break-even cut-off grade include an unknown quantity of ore which cannot support depreciation of the existing plant or depreciation of capital expenditures on new plant. They cannot, therefore, be used "per se" for expansion considerations, but must be broken down into mining periods to reflect the order in which the ore will be mined. The most convenient way of developing these mining periods is to compute ore inventories using a series of arbitrary pit limit cut-off grades, e.g. 0.24%, 0.20%, 0.16%, 0.14%, 0.12%, 0.10% and 0.08% MoS_2, giving rise to several essentially concentric pits. The highest cash flow pit, i.e., the one with the highest pit limit cut-off grade will be mined first followed by the one with next highest pit limit cut-off grade, and so on until the break-even pit limit increment is reached. Practical operating slopes are used in all the pits except the last when the slope is steepened to an engineered safe slope. The contents of the ore inventories in terms of different mill cut-off grades will be developed at the same time as the ore inventory is developed.

With the above data, fourteen financial evaluations can be prepared. In each evaluation, mining will be in the sequence of high cash flow pit through intermediaries to the break-even pit and each evaluation will include the most financially attractive method of handling depreciation. The fourteen financial evaluations would be based on:

1. Two production rates - the present 6,400 TPD and an expanded operation of 12,800 TPD.

2. For each production rate, seven milling cut-off grades, namely 0.24%, 0.20%, 0.16%, 0.14%, 0.12%, 0.10% and 0.08% MoS_2.

The evaluation which gives the highest net present value will be the one which indicates future action in the sense of whether an expansion is worthwhile and what the milling cut-off grade should be. The ore inventory which provides the highest net present value, providing the parameters used in respect to production rate and mill cut-off grade are implemented, will then become the ore reserves of the mine. In the meantime, it is recommended that the January 1969 ore reserves updated for operations during 1969 be amended to January 1970 reporting.

ORE RESERVE ESTIMATES

The ore reserves estimated both manually and by computer based on 0.113% MoS_2 pit limit cut-off and variable stripping ratios are:

	Tons Ore	Grade	Waste	S.R.
Manual	129,641,400	1.180% MoS_2	223,378,000	1.72:1
Computer	119,419,326	0.181% MoS_2	176,927,436	1.48:1

Parameters

In estimating B. C. Moly's ore reserves, the pit limit increment was determined on the valid basis that, in any open pit, mining will continue until such time as the mining and processing of an increment does not provide a cash flow. In this increment the revenue from a block of ore on surface is required to cover operating costs, through to sales, including its share of replacement capital necessary to maintain operations but excluding depreciation of existing plant. The grade of this block is the pit limit cut-off grade. A block of ore at depth is required to be of a higher grade such that the additional revenue over and above that of a cut-off grade block will cover the costs of stripping necessary to expose it for mining.

The parameters used in both manual and computerized reserves were:

Ultimate pit slope	45°
Minimum pit bottom width	120'
Pit limit cut-off grade	0.113% MoS_2
Stripping cost	$0.32 per ton waste

The ultimate pit slope is conservative. Studies by Mr. K. H. Rippere of MMD-ED indicate that the following safe ultimate pit slopes are feasible:

49° in the NE quadrant
55° in the SE quadrant
60° in the SW quadrant, and
38° in the NW quadrant.

The minimum pit bottom width is realistic.

The critical parameters which affect the ore reserve estimate are the pit limit cut-off grade and the stripping cost.

Pit Limit Cut-off Grade

The operating costs used for determining the cut-off grade are shown below together with B. C. Moly's operating costs for Jan. - Sept. 1969:

	B. C. Moly's Costs Jan. - Sept. 1969 $/ton ore	Costs Used for Cut-off Grade $/ton ore
Mining (w/o stripping)	0.453 (1)	0.430
Concentrating	1.040	0.800
General Plant	1.111	0.795
Depreciation	0.656	0.107 (2)
Sales @ $0.059/lb Mo	0.143	0.072
	$3.403	$2.204

(1) Includes $0.02 per ton deferred expense amortization.

(2) For replacement capital of $500,000 per year only.

The cut-off grade used, 0.113% MoS_2 would yield 1.22 lbs Mo/ton ore for sale assuming a 90% mill recovery at this grade. Revenue at $1.80 per lb Mo would, therefore, be $2.196 per ton ore which equates with the operating costs used.

The costs used for the cut-off grade are the mine's estimate for a milling rate of 12,800 TPD. The ore reserves developed are, therefore, on this one factor alone, only suitable for a 12,800 TPD operation and could not be justified for the present production rate.

For reporting purposes the ore reserves should be based on the present production capacity and associated costs.

Stripping Costs

The cost of stripping used in pit limit determinations has a profound effect on the magnitude of the ore reserves. A ton of ore which will support four tons of stripping at, say, 32¢/ton of stripping will only support two tons of stripping at 64¢ per ton of stripping.

It is my opinion that the stripping cost of 32¢/ton used in the determination of the ore reserves, compared with the current stripping cost of 43¢/ton, is too low.

The use of a lower than current stripping cost which is out of line with current KCC practice would:

1. Be suspect in the eyes of KCC's Management and almost impossible to defend. The first step in determining the best course of action for B. C. Moly and obtaining acceptance for such a course is to develop ore reserves which are defensible.

2. Include in the reserves, ore at depth which is mined toward the end of the life of the mine and which, because of the time value of money has little effect on the NPV. This ore, in any event, is sparsely drilled and its existence is due almost entirely to geological predictions.

For the above reasons I recommend using a stripping and mining cost of 43¢ per ton material, assuming that higher future haulage costs will be offset by future operating economies. This would follow the practice used in the other Divisions where current mining and stripping costs are used for long-range ore reserve calculations.

Manual Estimation of Ore Reserves

The ore reserves were prepared manually by Mr. Eldon Bray, Supervisor of Engineering and Geology at B. C. Moly. The method used was:

1. Sections of the ore body were prepared, from DDH assay information and using judgment to delineate ore containing 0.100% - 0.159%, 0.160% - 0.239%, 0.240% - 0.319%, and + 0.320% MoS_2 together with waste below 0.099% MoS_2. Inclined DDH's have indicated steep assay cut-off walls, and this information was used in the sections to delineate ore grades.

2. Leval maps were prepared at 35' bench height intervals, containing color contours of the different grades of ore and waste.

3. A stripping ratio graph was prepared (Exhibit 1) showing acceptable stripping ratios for the pit limit increment against the grade of the ore. For this purpose a pit limit cut-off grade of 0.113% MoS_2 (based on estimated operating costs for a 12,800 TPD operation of $2.204 per ton ore excluding stripping but including depreciation of replacement capital) and a stripping cost of 32¢/ton was used.

4. An ultimate pit slope of 45° and a minimum 120' pit bottom width were used.

5. Leval maps of segments of trial pits were drawn and the waste content compared with the c ntained ore grade. By a process of trial and error an ore reserve was developed which conformed with the stripping ratio graph.

<u>Computer Estimation of Ore Reserves</u>

Subsequently (Sept. 21, 1969), the ore reserves were estimated by computer using the following parameters:

1. Geological model and level maps as prepared manually.

2. A pit limit cut-off grade of 0.113% MoS_2 and a stripping cost of $0.32 per ton. (Same as manual.)

3. An ultimate pit slope of 45° and a minimum 120' pit bottom width. (Same as manual.)

4. Blocks sized 25' x 25' x 35' bench height.

The computerized ore reserves are:

0.113% MoS_2 Pit Limit Cut-off, Variable Stripping Ratios				
Mill Cutoff	Tons Ore	Grade	Tons Waste	S.R.
0.113% MoS_2	119,419,326	0.181% MoS_2	176,927,436	1.48:1
0.160% MoS_2	82,656,800	0.207% MoS_2	213,689,962	2.59:1

These are in close agreement with the manual estimates of ore reserves, but, because the geological model and interpretation of values were the same in both cases, it is simply a tribute to the meticulous accuracy and care used by Mr. Eldon Bray in his manual computation.

EXAMINATION OF ORE RESERVES

1. The files contain a substantial amount of correspondence between Mr. Eldon Bray and the S&ECC concerning DDH bearings, collar locations, depth of holes, etc. and it is accepted that the information used in this regard is correct.

2. For checking the ore reserves, the graph showing supportable stripping ratio vs. ore grade developed by the mine was used.

3. Using the ultimate pit contour map (Exhibit 2) and sections of the ore body prepared by Mr. Eldon Bray, six sections were drawn showing the relationship of the ore to the ultimate pit on E104,500; E105,000; E105,600; N106,800; N107,400 and N180,000 (Exhibit 3). The pit limit stripping ratio was determined from the sections so drawn and compared with the supportable stripping ratio graph. In each case, it was found that the ore at pit limits could support a larger amount of stripping than was indicated by the supportable stripping ratio graph used in the ore reserve estimation.

4. A detailed check of the grade colored contour level maps from the 2025' level to the 835' level was made to determine to what extent the geological predictions concerning the extension of the ore body away from drill holes influenced the magnitude of the reserves. A portion of the worksheet covering levels 1360' and below and the 1430' level map are attached (Exhibits 4 and 5) and reveal that:

   a. There is a dearth of DDH information between D47 on the west and D11 on the east; that is, over most of the NE, N, and NW portions of the ore body from the 1430' level down.

b. DDH or assay information is available from 13 holes on the 1360' level, reduced to 6 on the 1185' level, to 3 on the 975' level, and 2 on the 905' level and below.

Ore down to the 1360' level is well covered by drilling and can be termed "proven ore". Ore below the 1360' level should be classified as "probable ore."

DETERMINATION OF OPTIMUM ORE RESERVES

An improvement in operating costs has been recorded over the past year, such that B. C. Moly is now making a working profit, but this is insufficient to cover the interest payments on the debentures. An expansion of the operation to 12,800 TPD by mining ore to a lower cut-off grade with a consequent reduction of operating costs due to a bigger spread of the overheads is presently considered the most likely way of improving the economics of the operation.

However, care has to be taken in the use of ore reserves which have been estimated using a break-even pit limit increment for expansion purposes. Included in such ore reserves is an unknown quantity of ore which does not support depreciation of the existing plant, nor does it support the depreciation of capital expended on an expansion. From one pit it is impossible to estimate the quantity of this marginal ore. The only satisfactory way of determining "optimum ore reserves" for any property ("optimum" in the sense of providing maximum net present worth, maximum overall profits, maximum production of metal, or any other maximum required) is to estimate several ore inventories using a range of pit limit cut-off grades producing essentially a series of concentric pits. The pit with the highest pit limit cut-off grade is mined first followed by the others in sequence through to the break-even pit limit pit. Practical working slopes are used for all pits except the break-even pit limit pit where the slope is steepened to an engineered safe slope. Financial evaluations are then done at different milling cut-off grades and different rates of production, using the most advantageous method of handling depreciation and taxes. A comparison of the financial evaluations will reveal which of the ore inventories is most satisfactory for the "optimum" required, and this inventory then becomes the ore reserve for the time.

Important - For annual reporting purposes the ore reserves must be based on the current operating capacity of the mine. Ore reserves selected for expansion purposes must remain in embryo and used as supporting data only, until such time as the expenditure of funds on such an expansion is approved.

B. C. Moly Ore Reserves

In the case of B. C. Moly, the optimum ore reserves required are those that will, when milled at some to-be-determined-rate, yield the highest net present worth.

The recommended sequence for determining such ore reserves is as follows:

1. Computerize 7 ore inventories using the following parameters:

   a. Pit limit cut-off grades of 0.24%, 0.20%, 0.16%, 0.14%, 0.12%, 0.10% and 0.08% MoS_2.

   b. Stripping cost @ 43¢ per ton.

   c. Pit slopes of 45° throughout. Because of the configuration and the method of mining the ore body, an operating slope of 45° can be used.

2. Prepare 14 financial evaluations using the following parameters:

   a. Ore to be mined in sequence using the high cash flow (0.24% MoS_2 pit limit cut-off grade first).

   b. Two milling rates to be used - 6,400 TPD and 12,800 TPD.

   c. With each milling rate, milling cut-off grades of 0.24%, 0.20%, 0.16%, 0.14%, 0.12% 0.10% and 0.08% MoS_2 to be used.

   d. The best presently estimated capital costs for mill expansion, ancillaries, townsite, etc., and mine equipment to be incorporated where applicable.

   e. Determine the NPV of the mine in each case.

An examination of the financial evaluations will reveal whether or not an application for funds for the engineering of an expansion can be supported.

J. L. Halls

JLH:ff
Encl.
cc: Mr. S. D. Michaelson

Kennecott Exploration, Inc.

Exploration Services Department

Geologic Research Division

March 1, 1971

MEMO TO: Ralph C. Holmer

FROM: John C. Wilson

SUBJECT: Igneous Activity and Mineralization at B.C. Molybdenum, Ltd.

Bob Galbraith describes his current hypothesis for the relationship between intrusion and mineralization in the Lime Creek orebody at Alice Arm, British Columbia. Two intrusions, each with an associated triplet of vein types, form the portion of the stock observed at the surface and in the drill core. The first is a zoned diorite to quartz monzonite pluton, and the second is a system of closely spaced dikes of quartz monzonite.

Bob states that a third and youngest set of MoS_2-related veinlets is observed, but no intrusion is known to relate to this mineralization unless it is the so-called wipeout porphyry observed in DH-27, or the dike system is composed of two genetic parts. Ni and Co geochemistry are permissive of an additional nonoutcropping intrusion on the north-northeast side of the present pit.

This memo is a progress report on this research project and will be modified and fully documented as the investigation progresses.

John C. Wilson /gp

John C. Wilson

JCW:gp
Attachment

cc: R. M. Galbraith
D. L. Giles
C. T. Penney

2300 West 1700 South, Salt Lake City, Utah 84104 • Phone 801-486-6911 • TWX 910-925-5624

Kennecott Exploration, Inc.

Exploration Services Department

Geologic Research Division

February 24, 1971

MEMO TO: John C. Wilson

FROM: Robert M. Galbraith

SUBJECT: Igneous Activity and Mineralization at B.C. Molybdenum, Ltd.

The igneous intrusions and respective mineralization can be explained on the basis of experimental studies of the system $NaAlSi_3O_8$-$KAlSi_3O_8$-$CaAl_2Si_2O_8$-SiO_2. The only special condition necessary is that the magma be rich in calcium. This condition would enlarge the field in which plagioclase will crystallize and reduce both the K-feldspar field and the quartz field.

For purposes of this discussion, two intrusions will be referred to. The first will be called the original intrusion. This is the main stock in which mineralization occurs. It is bounded on the west, north, and east by hornfels. The south boundary is indefinite at present. It could be close to the south limit of mineralization or it could be the igneous hornfels contact farther to the south. Further study may indicate that the mineralization control on the south side of the orebody is an igneous contact. If so, the main stock described by earlier reports will have to be subdivided. The second intrusion referred to in this study is a porphyritic dike complex. The dikes are younger than the mineralization in the original stock but they are also mineralized.

The original intrusion is a classic zoned pluton. Chill effects along hornfels contacts have caused a diorite to crystallize. This is characterized by highly zoned plagioclase crystals growing in complex clusters which make up 70% of the rock. Hornblende and biotite make up 20% of the rock, with K-feldspar and minor quartz making up the rest. The K-feldspar appears to have crystallized slowly, for it often forms crystals up to three-eighths of an inch across. These crystals, however, do not produce a porphyritic texture. Instead, they are an intratelluric matrix surrounding plagioclase and mafic grains and having crystalline and optical continuity. These K-feldspars will not be seen in hand samples unless a cleavage surface reflection reveals its presence. Such crystals could only precipitate in a static, nearly isothermal, and relatively volatile-charged environment.

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The composition of the original intrusion grades to a more acid composition toward its center. The overall bulk of the intrusion is a granodiorite; nevertheless, toward the center, the composition verges on quartz monzonite. Interestingly, the K-feldspar grains seen in the diorite become more conspicuous, although they never developed to the point of interrupting the fabric and changing the texture of the rock from equigranular to porphyritic. The large K-feldspars usually include fine bits of plagioclase or mafic crystals. The inclusion of these bits causes the overall equigranular appearance of the hand specimen to be uninterrupted by the K-feldspars.

Magma differentiation continued until a residual liquid of virtually nothing but quartz, K-feldspar, metallic sulfides, and a very minor amount of plagioclase remained. The cooling crystalline shell around this residual liquid cracked, releasing the liquid which rapidly cooled, filling the cracks with alaskite dikes which sometimes contain minor plagioclase and some MoS_2 mineralization.

The residual liquid after removal of the alaskite fraction was comprised of K-feldspar, abundant quartz, and metallic sulfides. These were used up as subsequent fracturing of the cooling shell took place. First, quartz veins with the remaining K-feldspar were formed. It was probably during this period that most of the clay alteration took place. This was followed by the formation of quartz veins with MoS_2 mineralization, which in turn was followed by the last bit of liquid, forming quartz veins with sporadic pyrite mineralization.

An important question has arisen as to the ability of a diorite magma to generate this much quartz, K-feldspar, and water through fractional crystallization. The answer is that it probably wouldn't, at least not to the degree seen in this mineral deposit. The additional quartz, K-feldspar, and water must have come from the Bowser sediments. The intrusion must have been at least partially emplaced through assimilation of the overlying rock. Evidence of this is demonstrated by the many xenoliths of Bowser sediments seen in the pit walls and drill core. Not all the sediments are of the right composition to be susceptible to digestion by this magma; however, many xenoliths were seen to be partially or almost entirely assimilated. The second enrichment mechanism was most likely through the development of convection cells in the meteoric water in the Bowser sediments. Such cells would be developed due to the heat of the magma. These cells would introduce water that would be at least partially charged with potassium and silica into the magma.

The second intrusion is probably derived from a magma similar to the first and in the same general locus. This magma, however, produced a series of dikes rather than a zoned stock at the present exposure level. The dikes all have an aplitic K-feldspar-quartz groundmass. The variation in porphyritic crystal assemblages is gradational in much the same fashion as the original intrusion. This is demonstrated by three characteristic assemblages, but intermediate assemblages indicate a continuous sequence. First, very highly zoned plagioclase crystals form complex clusters, and many small plagioclase grains are in the matrix. The second assemblage has moderately zoned plagioclase crystals, small rounded quartz grains or eyes, and K-feldspar crystals that include mafic and plagioclase bits with the outer rim containing quartz blebs. The third assemblage has small plagioclase grains with very little zoning, large K-feldspar orystals, and larger quartz eyes up to 4 mm.

The dikes cut both hornfels and the original intrusion, but were not found to cut each other. It is postulated that they were emplaced almost contemporaneously and that variations in the porphyry assemblage are due to fractionation of the parent magma during emplacement. These dikes are mineralized by a triplet of quartz-K-feldspar, quartz-MoS_2, and quartz-pyrite veins. It is not possible to distinguish these veins from the ones related to the original intrusion without the aid of an igneous contact truncating the older veins.

This petrologic concept explains all the features seen in the pit except one. There is a third triplet of quartz veins which is younger than the set associated with the second intrusion. This triplet differs from the other two in that sometimes the quartz-MoS_2 veins show multiple banding and are up to 2 inches wide.

A third intrusion genetically related to this third triplet may exist. This intrusion may be what has been referred to as the "wipeout" porphyry and was first noted in drill hole 27 on the northeast side of the pit. However, since drill hole 27 does not encounter ore-grade mineralization in the hornfels, it is difficult to say this porphyry is displacing mineralization. The intersection of the igneous hornfels contact in drill hole 27 is some 250 feet northeast of the igneous hornfels contact exposed in the pit, and some 150 feet northeast of the ore zone. This composite porphyry appears to be little different from the dikes of the second intrusion. This similarity and the lack of mineralization above it casts doubt on this rock being the source of the third triplet.

A more distinct possibility is that the dikes of the second intrusion are actually related to two magmas. Contacts are difficult to see in the pit, and we could have missed crosscutting relationships. However, everywhere we looked, the younger dikes had two sets of crosscutting quartz veins. Thus, the source of the last period of molybdenum mineralization is unresolved.

The distribution of lower grade areas in the ore zone seems to be related to the distribution of younger intrusive dikes. These dikes are mineralized by only two triplet sets of quartz veins. The best ore grades are where the original intrusion is mineralized by the alaskite dikes and three triplet sets of quartz veins. The density of porphyry dikes is greatest in the northern third of the ore ring. This is also the area where ore grades drop off and the ore zone as presently evaluated is only 150 feet wide.

Geochemical anomalies in Ni and Co indicate the possible presence of an intrusion under the hornfels on the north-northeast side of the present pit. Perhaps this is the source of the younger intrusive dikes. The possibility also exists that this is a third intrusion and the source of the last mineralization triplet. Hopefully, the drill core and hand samples in shipment from B.C. Molybdenum will lead to a solution of this quandary.

Bob

Robert M. Galbraith

RMG:ms