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INSPIRATION CONSOLIDATED COPPER COMPANY

INTEROFFICE MEMORANDUM

TO: J. Burgess Winter, Deputy General Manager DATE January 12, 1979

FROM: H. H. Mellus, Supt. of Long Range Planning

SUBJECT: WHITEHORSE COPPER MINES LTD.

As per your request I visited the Whitehorse Copper Mines Ltd. operation located 6 miles south of Whitehorse, Yukon Territory, Canada during the week of January 1-7, 1979. The purpose of this trip was to advise their staff on open pit planning procedures for a small skarn type ore deposit (Cowley Park) situated approximately 10 miles south of their underground Little Chief Mine.

The underground mine has an estimated life of three years at 2500 tons per day production; the capacity of their present treatment facilities. Because of considerable molybdenum content in the Cowley Park deposit and the abrasive factor of its calcine silicate host rock--both items that differ from the present underground ore--it would be advisable to bring the Cowley Park deposit into production after the underground orebody has been depleted. Therefore, any economic open pit planning should be based on estimated 1982 cost data and metal selling prices.

Dave Tenney, Chief Geologist for Whitehorse Copper Mines Ltd., had designed a preliminary pit for the Cowley Park deposit in November 1978. Based on a .50% total copper ore grade cutoff, his calculated ore reserves were 975,000 tons @ 1.04% total copper (diluted) with a 2.3/1.0 overall stripping ratio. At 2500 tons per day this would result in a 390-day mine life, or approximately 13 months. The 0.50% cutoff was not based on economics, so it is doubtful whether all the 975,000 tons would be profitable to process through the present treatment facilities, with or without mining costs included. The pit perimeters had been designed by a visual observation of the pendant type skarn orebody as depicted on geological north-south sections drawn on 50' intervals. The inclined diamond drill development holes had been put down on this 50' spacing, so they represented the true intercepts of the orebody at these points, rather than projected ones. As the apparent strike of the skarn is only 10-15^{o} off true north, any drafting of pit slopes from these sections would be distorted only slightly when transferred to form a composite bench plan map.

Tenney's pit layout by visual observation of the section information was actually a modified final increment pit planning method (minus economics), as he had used a typical cut-out section with overall pit slopes established at 62^{o} (three 25' benches at 79^{o} plus a 25' catch berm). In this typical cross section he had first established a bottom bench elevation with a minimum operational mining width. By moving this typical bench outline section back and forth across the geological sections, he determined the position where it showed the least waste for the most ore. When projecting the pit slopes per bench established by these trial and error sections to plan, and then joining them to produce a composite toe plan, the irregularity of the orebody had presented problems that increased the amount of waste removal.

An examination of the sections and diamond drill hole data showed areas where information was definitely lacking to arrive at grade and tonnage calculations that could be considered confident. Additional drilling is definitely needed to more correctly define the ore zones and afford better interpolation between sections. The orebody is definitely not uniform in width or grade along its strike. However, until an economic analysis has been made based on this available information to determine whether it would be profitable to mine at 1982 projected costs and selling price of contained metals, the cost of further development drilling should be postponed. Tenney feels that further drilling could only increase the reserves approximately 10% where information is lacking on the outer perimeter.

On the basis of the above observations, I have recommended to Doug Linzey, General Manager, and Dave Tenney, Chief Geologist, that a breakeven ore grade be calculated as follows:

<u>Establishment of Breakeven Ore Grade</u>

The breakeven ore grade is that total copper grade (decreased by a precious metal and moly content copper equivalent) expressed in value per ton recovered that equals the ton costs for mining and treating a ton of ore plus the pound costs. This can be expressed by the formula:

$$(V \times 20 \times R \times S) - \left[\text{ton costs} + (V \times 20 \times R \times \text{pound costs})\right] = 0$$

where V = total copper grade R = recovery S = copper selling price

After V has been calculated, it should then be reduced by the copper equivalent of the value of the recovered precious metals and moly content per ton of ore. To arrive at the adjusted value for V, one can simply consider the recovered precious metal and moly value per ton as a credit against the ton cost, and solve accordingly for V.

The R coefficient can be determined from historic data at the Little Chief Mine or copper recovery from the four small pits that existed prior to 1971. Dave Tenney has estimated R as 90%.

Hudson Bay Mining has predicted (December 1978) that the selling price of copper (S) for 1982 will be 97¢ U.S. per pound. The question arises as to the currency exchange rate at that time. Presently it is .865. Will this be in effect in 1982? I have advised Doug Linzey to have Hudson Bay Mining make this decision.

Ton costs are those that apply to the mining, treatment, administration, etc., for one ton of ore. As Whitehorse Copper Ltd. underground mining costs would not apply in this situation, Chinook Construction and Excavation Ltd. were given parameters on rock type, loading, haulage distance, preliminary waste and ore tonnages, production requirements per day, etc., and asked to arrive at a mining cost per ton of ore and waste. Based on 1979 predicted costs for labor, supplies, equipment, etc., they came up with a cost of \$5.85 per ton of ore and \$2.20 per ton of waste. In conversation with Rod Bullock of Chinook, he felt that these costs could be lowered with more careful scrutiny of the proposed operation.

Whitehorse Copper's 1978 ton costs outside of mining were: milling \$2.89; plant .94; administration .83; and head office .69. These can be used for estimating purposes, except the milling costs, which should be increased 15% for the abrasive action of the Cowley Park material. Pound costs, which include freight, marketing, smelting and selling, averaged 30¢ per pound for 1978. All of the above costs should be escalated to 1982; in discussion with Doug Linzey, we agreed on a 9% per annum escalation factor.

On the above basis the following breakeven ore grade for the total copper grade without precious metals or moly value consideration can be established if today's currency conversion factor is used.

$$(V \times 20 \times .90 \times .97 \div .865) - \left[15.74 + (V \times 20 \times .90 \times .423)\right] = 0$$

$$\therefore V = 1.252$$

Approximately 50 samples have been composited from the +0.50 material in the diamond drill core from the Cowley Park deposit and sent to the assay lab for determination of Au, Ag and MoS_2 content. Until these results are known, D. Tenney has estimated that .005 oz. Au, .1 oz. Ag and .066% MoS_2 are reasonable figures to use for these calculations. He estimates gold and silver recovery at 90% (the same as copper), and Doug Linzey feels that moly recovery would be 70% with a cost of 50¢ (1979) per ton of mill feed. Hudson Bay Mining has forecast in 1982 that gold will sell for \$227 per oz. and silver for \$7.77--both quoted in U.S. currency. Molybdenum is now quoted at \$5.86 per lb. (U.S.)--an all-time high, I believe. As more moly projects are scheduled to come into operation in the next few years, I believe that this price should not be escalated for 1982.

On the above basis the recovered Au and Ag value per ton is \$1.99 and the moly (less treatment cost) is \$3.11. Subtracting this \$5.10 credit from the previous ton costs and recalculating for V:

$$(V \times 20 \times .90 \times .97 \div .865) - \left[10.64 + (V \times 20 \times .90 \times .423)\right] = 0$$

$$V = .846 \quad \text{(see Appendix A)}$$

Therefore, the copper equivalent for the precious metals and moly values estimated per ton of ore is 1.254 - .848 = .406. As this estimate of the precious metals and moly value per ton of ore has dropped the cutoff grade 33%, one can readily see that the forthcoming results from the 50 composite samples could have a significant effect on pit planning.

By introducing the $2.20 cost per ton for mining of the waste into the breakeven formula, you can derive a breakeven stripping ratio for any grade of ore above the .848 cutoff. For example, at a 1/1 stripping ratio the breakeven grade would be solved as follows:

$$\frac{(V \times 20 \times .90 \times .97 \times .865) - \left[10.64 + (V \times 20 \times .90 \times .423)\right]}{2.85} = 1$$

$$V = 1.073$$

For every ton of waste removed per ton of ore, the breakeven grade rises .227% total copper. Therefore, with the stripping ratio estimated at 2.3/1.0 by Tenney, the breakeven grade to support this would be .848 + (2.3 x .227) = 1.37. As his estimate showed a 1.04 total copper grade, this pit is not economical to mine unless some of the cost or selling price parameters can be changed.

One can argue that as long as one has to move a ton of material out of the pit to make another ton of ore available, the $2.20 waste cost per ton quoted by Chinook could be subtracted from the $5.85 ore ton cost, thereby lowering the breakeven grade to the plant by a similar .227% total copper. This is true in actual mining but should not be applied in the overall planning of the pit--only in the dispatching of ore if the pit planning proves economical.

Based on the above results, I have recommended to Dave Tenney that (1) the costs and selling prices in the above breakeven analyses be reviewed, (2) the pit slope angle be investigated, and (3) a new pit layout be initiated, establishing the ultimate pits on an economic basis as described in the paper entitled "Final Pit Limit Shell Generation--Hand Methods" by Benjamin C. Koskiniemi. This was presented at the Open Pit Mine Planning and Design Symposium in Denver, Colorado on February 23-25, 1978. A copy of the papers presented at this symposium has been left with the Whitehorse Copper Mines Ltd. staff.

Although Koskiniemi's methods are more adaptable on a final increment basis to a porphyry type orebody, he does show the pit limits established for a steeply dipping vein type deposit where one slope and the bottom is in ore--similar to the skarn deposit at Cowley Park. In conversation with Tom Couzens, Manager of Mine Planning for Pincock, Allen & Holt of Tucson, Arizona, he stated that he also had designed open pits for skarn deposits using this same final increment method.

I suggest that Tenney visit the Keno Hills Mine at Mayo in the Yukon Territory, where I understand they are now operating a small open pit. A comparison of their mining costs versus those submitted by Chinook might show that the $5.85 per ton of ore estimate could be lowered. A drop of $1.00 in ton costs would decrease the breakeven grade by 0.08 so it is important that this ton cost be reliable as possible.

The decision on an overall pit slope angle of 62° was based on previous bank slopes of the small pits at Whitehorse. Although I did not visit these sites, I understood that the walls were competent at this angle, which included room for a safety berm for every three benches. It might be expedient to have a slope stability expert give his opinion on this. The steeper the angle on the hanging wall side of the deposit, the less waste to be moved.

The cost escalation factor also has considerable effect on the calculated breakeven ore grade. Lowering from 9% to 8% per annum in this case would decrease the breakeven ore grade from .848% to .790% total copper. This 9% per annum escalation in costs, though, does not wipe out the benefit from the increase in copper selling price of 65¢ U.S. average 1978 to the 97¢ forecast by Hudson Bay Mining in 1982; 65¢ escalated 9% for four years would be only 92¢.

One cost that has not been considered in the above evaluation is the capital cost expenditure for the 10-mile haulage road from Cowley Park to the plant facilities at the Little Chief Mine. Apparently, seven miles of this would need only slight improvements--the remaining three miles are only a trail today. Each $100,000 of road construction costs by the contractor would add approximately 10¢ to the ton costs, affecting the breakeven grade by .008.

Without further work this preliminary study has shown that it is uneconomical to mine the Cowley Park orebody by open pit methods unless there is an increase in metal values, or costs can be drastically reduced. Dave Tenney shares this view also.

I wish to acknowledge the excellent cooperation and hospitality of Doug Linzey, General Manager of Whitehorse Copper Mines Ltd., and his staff, who supplied me with the information for this evaluation. I feel that, with the above suggestions and the pit planning parameters and procedures which were discussed with them, they are very capable to continue this investigation of the Cowley Park deposit. If necessary, I would be very willing to be available for further consultation.

Herbert H. Mellus

Herbert H. Mellus
Superintendent of Long Range Planning

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cc to: P. Steen D. Tenney
D. Linzey P. Richardson

BREAKEVEN CALCULATION FOR WHITEHORSE COPPER

Selling Price

Cu = \$ .97 per lb. U.S. in 1982)
Au = \$227. per oz. U.S. in 1982) Hudson Bay Mining Forecast
Ag = \$7.77 per oz. U.S. in 1982)

Mo = \$5.86 per lb. U.S. at Jan. 1, 1979

Present conversion factor = .865

Average Assays per Ton

Au = .005 oz. Ag = .1 oz. MoS_2 = .066%

Ton Costs

Mining including haulage per ton of ore = 5.85 Canadian (1979)
Mining including haulage per ton of waste = 2.20 Canadian (1979)

Milling = 2.89 Canadian)
Plant = .94 Canadian)
Administration = .83 Canadian) 1978 Whitehorse Mines Ltd.
Home Office = .69 Canadian)

Moly = .50 Canadian per ton of mill feed (1979)

Pound Costs

Freight, smelting, marketing, selling = 30¢ Canadian (1978)

Cost Escalation

9% per annum - plus initial 15% for milling for abrasive wear of Cowley Park ore

Recovery

90% for Cu, Au, Ag
70% for Moly

Recovered Values per Ton (except Cu)

Au = .005 x \$227 x .9 ÷ .865	=	1.181
Ag = .1 x \$7.77 x .9 ÷ .865	=	.808
Mo = .066 x .6 x 20 x .7 x 5.86 ÷ .865	=	3.756
		5.745
Less moly treatment .50 x 1.09^3		.648
Precious metal and moly credit		5.097

Breakeven Ore Grade Calculation

Formula = (V x 20 x R x S) - [ton costs + (V x 20 x R x pound costs)] = 0
where V = breakeven total grade   R = recovery   S = selling price

$$(V \times 20 \times .90 \times .97 \div .865) - \left[(2.89 \times 1.15) + .94 + .83 + .69\right] \times 1.09^4 - (5.85 \times 1.09^3) + 5.097 - (V \times 20 \times .90 \times .30) \times 1.09^4 = 0$$

$$20.185V - (5.7835 \times 1.09^4) - (5.85 \times 1.09^3) + 5.097 - 7.623V = 0$$

$$20.185V - 7.623V = 8.164 + 7.576 - 5.097$$

$$12.562V = 10.643$$

$$V = .847\ ^{*}$$

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* This is .001 different than quoted in the report, due to the rounding off of significant figures.

HHM/als
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