Cariboo Bell
671947

APPENDIX

APPENDIX

PROPOSED DRILLING

Zone 2
Zone 3
Summary

EXTRACTION - Tons and Grade

Summary
Details - (1) Pits 1, 2, 3 & 4.
(2) Subsequent Pits.
(3) Remaining Sulphides - approximate strip ratio and grade.

ECONOMICS -

Repayment of Capital - Alternative No. 1
" " " - " " 2
Estimate Operating Profit
Typical Smelter Schedule
Net Value Per Pound of Copper
Estimate Operating Costs
Estimate Capital Costs
Miscellaneous Calculations - (1) Credit re stockpiled ore.
(2) Cost - waste and "non-sulphide" removal - Pit #4.
(3) Total operating profit.

METALLURGY -

Letter Mr. S. Gray, Metallurgical consultant.
Metallurgical Summary - Japanese Group.
Metallurgical Test Results - Summary
- Pit 1)
Pit 2) Cominco, Department of Mines
Pit 3) and Galigher
Pit 4)

Metallurgical Test Report -
Summary and Conclusions - Galigher Co.

PROPOSED DRILLING

(A) ZONE 2.

SECTION	FILL-IN	DEEPENING	EXPLORATION
13,000			800
			800
13,200	900		
	900		
	900		
13,400	500		900
			900
			900
			900
			500
13,600	860	240	
	980	530	
		200	
13,800	800	180	800
	800		800
14,000	800	150	
	800		
14,200	500		
	500		
	500		
14,400	700		
	700		
Total	11,140	1,300	7,300

PROPOSED DRILLING

(B) ZONE 3.

SECTION	FILL-IN	DEEPENING	EXPLORATION
12,550	500 500	180.	
12,800	500 500	250 250 190.	
13,000	500 500	210 210	500 500.
13,200	500 500	190 190 200	
13,400	500	200 200	500 500.
13,600	500	190	~~[illegible]~~
13,800		250	500 500.
14,000	500	140 180 280.	
14,200		200	550 550.
14,400	500	500	
Totals	6,000	4,010	4,100

PROPOSED DRILLING

SUMMARY.

	FILL IN	DEEPENING	EXPLORATION	TOTAL
ZONE 2	11,140	1,300	7,300.	19,740
ZONE 3	6,000	4,010	4,100	14,110
ZONE 4.	—	—	3,000	3,000
TOTALS.	17,140.	5,310	14,400	36,840
GEOCHEM. ANOMALIES TO NORTH	—	—	8,000..	8,000
			TOTAL	44,840.

CARIBOO-BELL COPPER MINES LIMITED

SUMMARY EXTRACTION

Tons and Grade - % Copper

		SULPHIDES (Before dilution)			NON-SULPHIDES (Before dilution)			WASTE (Before adding back dilution	Extra for Ends & Roads @ 25%	TOTAL WASTE
		Tons	Grade	Units	Tons	Grade	Units			
Pits #1, 2 & 3										
	Preproduction	695,400	0.634	439,695	3,398,100	0.472	1,604,511	7,206,300	-	7,206,300
	Production	6,404,300	0.632	4,048,661	1,221,400	0.592	722,469	4,193,400	-	4,193,400
	Totals	7,099,700	0.632	4,488,356	4,619,500	0.504	2,326,980	11,399,700	-	11,399,700
Pit #4	Preproduction	-	-	-	1,255,700	0.400	502,169	8,208,700	-	8,208,700
	Production	2,962,400	0.612	1,813,095	-	-	-	3,529,200	-	3,529,200
	Totals	10,062,100	0.626	6,301,451	5,875,200	0.482	2,829,149	23,137,600	-	23,137,600
Subsequent Pits		6,312,700	0.464	2,926,749	100,000	0.350	35,000	8,100,800	2,025,200	10,126,000
	Totals	16,374,800	0.564	9,228,200	5,975,200	0.479	2,864,149	31,238,400	2,025,200	33,263,600
	Remainder	7,095,200	0.400	2,834,013	3,881,000	0.441	1,710,538	15,216,600	3,804,200	19,020,800
	Total	23,470,000	0.514	12,062,213	9,856,200	0.464	4,574,687	46,455,000	5,829,400	52,284,400
	Add Non-Sulphides	9,856,200	0.464	4,574,687						
	GRAND TOTAL	33,326,200	0.500	16,636,900						

EXTRACTION (Continued)   DETAILS

Under Alternative No. 2, it is estimated as follows that 7,809,600 tons averaging 0.587% copper and 0.015 ounces gold per ton could supply feed to a 6,000 ton per day concentrator for the first 3.72 years from three open pit areas by using a cut-off grade of 0.30% copper. Tables are available but not presented on this report to show similar figures for the other three alternatives.

1. PREPRODUCTION PERIOD - Pits 1, 2 & 3

	Tons	Grade	Units
"Sulphide" ore in place	695,400	0.634	439,695
Plus dilution allowance @ 10% using waste grade	69,500	0.165	11,676
Stockpiled pending treatment	764,900	0.590	451,371
"Non-sulphide" ore in place	3,398,100	0.472	1,604,511
Waste in place	7,206,300	0.165	1,189,855
Total - "Non-sulphides" and waste	10,604,400	0.263	2,794,366
Less dilution as above	69,500	0.168	11,676
Total	10,534,900	0.264	2,782,690

2. PRODUCTION PERIOD - Pits 1, 2 & 3

	Tons	Grade	Units
Sulphide ore in place	6,404,300	0.632	4,048,661
Plus dilution allowance	640,400	0.130	83,227
Fed to concentrator (in addition to sulphides above)	7,044,700	0.587	4,131,888
"Non-sulphide" ore in place	1,221,400	0.592	722,469
Waste in place	4,193,400	0.126	529,419
Total - "Non-sulphides" and waste	5,414,800	0.231	1,251,888
Less dilution as above	640,400	0.130	83,227
Total	4,774,400	0.244	1,168,661

Overall strip ratio $\frac{10,534,900 + 4,774,400}{764,900 + 7,044,700} = \frac{15,309,300}{7,809,600} = 1.96:1$

Strip ratio during production $\frac{4,774,400}{7,809,600} = 0.61:1$

A fourth open pit with a cut-off grade 0.30% copper would provide a further 3,258,600 tons averaging 0.566% copper and 0.015 ounces gold per ton during the next 1.55 years as follows:-

EXTRACTION (Continued)

3. PREPATORY WASTE REMOVAL - Pit #4

	Tons	Grade	Units
"Non-sulphides"	1,255,700	0.400	502,169
Waste in place	8,208,700	0.126	1,035,667
Total - "Non-sulphides" and waste	9,464,400	0.162	1,537,836

4. PRODUCTION PERIOD - Pit #4

	Tons	Grade	Units
Sulphide ore in place	2,962,400	0.612	1,813,095
Plus dilution allowance	296,200	0.111	32,878
Feed to concentrator	3,258,600	0.566	1,845,973
Waste in place	3,529,200	0.111	390,273
Less dilution as above	296,200	0.111	32,878
	3,233,000	0.111	357,395

Overall strip ratio $\frac{9,464,400 + 3,233,000}{3,258,600} = 3.90:1$

Strip ratio during production $\frac{3,233,000}{3,258,600} = 0.99:1$

5. SUBSEQUENT OPERATION - Succeeding Pits - 3.31 years

	Tons	Grade	Units
Sulphide Reserves - in place	6,312,700	0.464	2,926,749
Plus dilution allowance	631,300	0.100	63,130
	6,944,000	0.431	2,989,879
"Non-sulphide" ore in place	100,000	0.350	35,000
Waste in place	10,126,000	0.100	1,012,600
Total - "Non-sulphides" and waste	10,226,000	0.102	1,047,600
Less dilution as above	631,300	0.100	63,130
	9,594,700	0.103	984,470

Strip ratio $\frac{9,594,700}{6,944,000} = 1.38:1$

EXTRACTION - Continued

ESTIMATE STRIP RATIO AND GRADE -
REMAINING SULPHIDES (After first 8.58 years)

	Tons	Grade	Units
"Sulphides"			
In place	7,095,200	0.400	2,834,013
Plus dilution - 10% at waste grade	709,500	0.100	70,950
	7,804,700	0.372	2,904,963
"Non-sulphides"			
Necessary to remove	2,416,100	0.443	1,069,600
Waste	15,216,600	0.100	
Total	17,632,700		
Less dilution	709,500		
Allowance	16,923,200		

Strip ratio $\frac{16,923,200}{7,804,700} = 2.17:1$

REPAYMENT OF CAPITAL - ALTERNATIVE No 1.

Price of Copper - Cents U.S.	39			40			41			42		
Period of time	First 2.94 yrs.	Next 1.18 yrs.	Total 4.12	First 2.94 yrs	Next 1.18 yrs	Total 4.12 yrs	First 2.94 yrs	Next 1.18 yrs	Total 4.12 yrs.	First 2.94 yrs.	Next 1.18 yrs.	Total 4.12 yrs.
Pits considered.	1, 2 & 3	4		1, 2 & 3	4		1, 2 & 3	4		1, 2 & 3	4.	
Cutoff grade - % copper	0.40	0.40.		0.40	0.40		0.40	0.40		0.40	0.40	
Tons sulphide ore - millions.	6.1694	2.4805	8.6499	6.1694	2.4805	8.6499	6.1694	2.4805	8.6499	6.1694	2.4805	8.6499.
Diluted grade - % Copper	0.660	0.643		0.660	0.643		0.660	0.643		0.660	0.643	
Lbs copper fed to concentrator per ton of ore.	13.20	12.86		13.20	12.86		13.20	12.86		13.20	12.86.	
Lbs copper recovered @ 85% per ton of ore.	11.22	10.93		11.22	10.93		11.22	10.93		11.22.	10.93.	
Approx net value per lb of Copper cents Canadian	36.16.	36.16		37.19	37.19		38.23	38.23		39.26.	39.26.	
Revenue per ton of ore $ Canadian	4.057	3.952		4.173	4.065		4.289.	4.178		4.405	4.291	
Operating costs per ton of ore - $ Canadian	2.644	2.400		2.644	2.400		2.644	2.400		2.644.	2.400.	
Operating profit per ton of ore - $ Canadian	1.413	1.552		1.529	1.665		1.645	1.778		1.761.	1.891	
Operating profit per year thousands $ Canadian	2,967	3,259		3,211	3,496		3,455	3,734		3,698.	3,971	
Total profit over life of pits thousands - $ Canadian	8,723	3,846	12,569	9,440.	4,125	13,565	10,156.	4,406	14,562	10,872	4,686	15,558.
Capital Investment.			15,190			15,190			15,190			15,190.

Repayment of Capital - Alternative No 2.

Price of Copper - Cents U.S.	39			40			41			42		
Period of time	First 3.72 yrs.	Next 1.55 yrs.	Total 5.27 yrs.	First 3.72 yrs	Next 1.55 yrs	Total 5.27 yrs	First 3.72 yrs	Next 1.55 yrs	Total 5.27 yrs.	First 3.72 yrs.	Next 1.55 yrs.	Total 5.27 yrs
Pits considered.	1, 2 & 3	4		1, 2 & 3	4		1, 2 & 3	4		1, 2 & 3	4.	
Cutoff grade - % copper.	0.30	0.30		0.30	0.30		0.30	0.30		0.30	0.30.	
Tons sulphide ore - millions.	7.8096	3.2586	11.0682	7.8096	3.2586	11.0682	7.8096	3.2586	11.0682	7.8096	3.2586.	11.0682.
Diluted grade - % Copper	0.587	0.566.		0.587	0.566		0.587	0.566		0.587	0.566.	
Lbs copper fed to concentrator per ton of ore.	11.74	11.32		11.74	11.32		11.74	11.32		11.74	11.32.	
Lbs copper recovered @ 85% per ton of ore.	9.98	9.62		9.98	9.62		9.98	9.62		9.98.	9.62.	
Approx net value per lb of Copper cents Canadian	36.16	36.16		37.19	37.19		38.23	38.23		39.26	39.26.	
Revenue per ton of ore $ Canadian	3.609.	3.479.		3.712	3.578.		3.815	3.678		3.918.	3.777.	
Operating costs per ton of ore - $ Canadian	2.361	2.136		2.361	2.136		2.361	2.136		2.361	2.136	
Operating profit per ton of ore - $ Canadian	1.248	1.343		1.351.	1.442		1.454	1.542		1.557.	1.641	
Operating profit per year thousands $ Canadian	2,621	2,820		2,837	3,028		3,053	3,237		3,270.	3,446.	
Total profit over life of pits thousands - $ Canadian	9,750	4,371.	14,121	10,554.	4,693	15,247	11,357	5,017.	16,374	12,164.	5,341.	17,505
Capital Investment.			15,022			15,022			15,022			15,022

ESTIMATE OPERATING PROFIT

PITS SUBSEQUENT TO PITS #1, 2, 3 & 4.

Period of time - 3.31 yrs.

Price of copper - cents U.S.	39	40	41	42
Cut off grade - % copper	0.30.	0.30	0.30	0.30.
Tons "sulphide" ore - millions	6.944	6.944	6.944	6.944
Diluted grade - % copper	0.431	0.431	0.431	0.431.
Pounds copper fed to concentrator per ton of ore.	8.62	8.62	8.62	8.62
Pounds copper recovered @ 85% per ton of ore	7.33	7.33	7.33	7.33
Approx net value per lb of copper - cents Canadian	36.16	37.19	38.23	39.26
Revenue per ton of ore - $ Canadian	2.651	2.726	2.802	2.878
Operating costs per ton of ore $ Canadian	2.300	2.300	2.300	2.300.
Operating profit per ton of ore $ Canadian	0.351	0.426	0.502	0.578
Operating profit per year thousands $ Canadian	737	895	1,054	1,214
Total profit over life of pits thousands $ Canadian	2,439	2,962	3,489	4,018.

510 × 3
1530

CARIBOO-BELL COPPER MINES LIMITED

TYPICAL SMELTER SCHEDULE

A. Gold and silver content in concentrates -

Recovery copper = 85%

Ratio of concentration $= \frac{27.00}{0.66 \times 0.85} = \frac{27.00}{0.5610} = 48.13$

Gold content in concentrate = 0.015 × 48.13 × 80% recovery = 0.578 ounces per short ton concentrate or $\frac{2,204.62 \times 0.578}{2,000}$ = 0.637 ounces/MT.

Silver in concentrate = 0.05 × 48.13 × 80% recovery = 1.9252 ounces/ST.
or 1.10231 × 1.9252 = 2.122 ounces/MT.

B. Value of concentrates -

38¢ U.S. per pound of copper and ore averaging 0.660% copper with 0.015 ounces gold per ton and 0.05 ounces silver per ton.		$ U.S. Per D.M.T.
(a) Copper - 2,204.62 × (27.00 - 1.00) = 573.20#		
573.20 × (38.00 - 1.00) = ...	$	212.08
(b) Gold - 0.637 × 95% × $35.00 = ...		21.18
(c) Silver - 2.122 × 90% × $2.00 per ounce		2.82
	$	237.08
Less treatment		28.00
	$	209.08

C. Value per S.D.T. $= \frac{209.08 \times 2,000}{2,204.62}$ = $189.674 U.S.

+9.87 S.D.T. + anode charge = $135.—

		Cents U.S.	Cents Can.
D. Value per pound of copper (including Au & Ag credit)	$= \frac{189.674}{540}$	= 35.125 × 1.075	= 37.757

$3/lb Cu paid for sales commission

(Note: Factor $= \frac{37.759}{20.908} = .001806$)

Cost of smelting and refining per pound of copper	= 2.875 × 1.075 .... =	3.091
	Total....................	40.850

		P.S.D.T.-Can.		
E. Freight -	Mine to P. G. E. & load	$ 4.50		
	Rail to Vancouver	6.25		
	Bulk load Vancouver	3.50		
		14.25	or $\frac{14.25}{540}$ =	2.638¢

F. Total Deductions 5.729¢

G. Net Value per pound of copper including Au and Ag credit 35.121¢

CARIBOO-BELL COPPER MINES LIMITED

NET VALUE PER POUND OF COPPER

(1) Cases	(2) \$U.S. Per M.T. Value of Concs. re Cu content (1-\$1.00) x 573.20	(3) \$U.S. Per M.T. Gold & Silver Credit less treatment chgs.	(4) \$U.S. per M.T. Net 2 + 3	(5) ¢ Canada Net Value per pound of copper 4 x .001806	(6) ¢ Canada Freight Cost per pound of copper	(7) ¢ Canada Net value per pound of copper 5 - 6
0.38	212.08	(3.00)	209.08	37.759	2.638	35.121
0.39	217.82	(3.00)	214.82	38.796	2.638	36.158
0.40	223.55	(3.00)	220.55	39.831	2.638	37.193
0.41	229.28	(3.00)	226.28	40.866	2.638	38.228
0.42	235.01	(3.00)	232.01	41.901	2.638	39.263
0.43	240.77	(3.00)	237.77	42.941	2.638	40.303
0.44	246.48	(3.00)	243.48	43.972	2.638	41.334
0.46	252.21	(3.00)	249.21	45.007	2.638	42.369

ESTIMATED OPERATING COSTS – 6000 T.P.D.
PITS #1, 2, 3 & 4

Assumption – waste stripping & ore mining performed by independent contractor.

Alternative.	(1)	(2)	(3)	(4)	(5)
Pits considered	1, 2 & 3	1, 2 & 3	4.	4	Subsequent Pits.
Cut-off grade % copper.	0.40	0.30	0.40	0.30.	0.30.
Waste to ore ratio during production –	1.01	0.61	1.62.	0.99	1.38.

COSTS PER TON OF ORE MILLED.

	$				
Mining @ 42¢ per ton moved.	0.844.	0.676.	1.100	0.836	1.00
Credit re stockpiled ore.	0.037	0.039	–	–	–
Waste & non sulphide removal re mining # 4 Pit	0.537	0.424	–	–	–
Milling	0.900	0.900	0.900	0.900	0.90
Administration & overhead	0.400	0.400	0.400	0.400	0.40
Total – operating cost / ton milled.	2.644	2.361	2.400	2.136.	2.300

ESTIMATE CAPITAL COSTS – 6000 TPD.

Description – waste stripping & ore stockpiling performed by independent contractor.

Alternative	(1)	(2)
Cut off grade – % copper	0.40	0.30.
Preproduction waste including "non sulphides" – tons	10,726,900	10,534,900.
Preproduction "sulphides" – tons	572,900	764,900.
COSTS – THOUSANDS $ CANADIAN		
(1) Waste stripping @ 35¢/ton.	3,754.	3,687
(2) Ore stockpiling @ 40¢/ton.	229	306.
(3) Mine costs.	67	67.
(4) Concentrator	6000	6000
(5) Power 30 miles @ 20,000/mile plus 100,000 switchgear etc	700	700
(6) Camp	750	750
(7) Water supply	750	750
(8) Tailings	500	500
(9) Shops & other facilities	250	250.
Total.	13,000.	13,010.
Working capital – 3 months operating costs (54000 tons)	1,428	1,275.
Inventory	500	500
	14,928.	14,785
Less cash for equity	2,000	2,000
	12,928	12,785
Interest – avge of 3½% x 5 yrs.	2,262	2,237
TOTALS. – THOUSANDS $ CAN.	15,190	15,022

MISCELLANEOUS CALCULATIONS

Credit re stockpiled ore

	Alternative No 1.	No 2
Cut off grade – % copper	0.40	0.30.
Tons stockpiled	572,900	764,900
Cost @ 40¢	$229,160	305,960
Tons milled – life of 1, 2 + 3 Pits.	6,169,400	7,809,600
Cost per ton milled.	$0.037	0.039.

Cost – Waste + Non sulphide removal – #4 Pit.

Tons – waste + non sulphides.	9,464,400	9,464,400.
Cost @ 35¢/ton	$3,312,500.	$3,312,500.
Tons milled – life of 1, 2 + 3 Pits.	6,169,400	7,809,600.
Cost per ton milled.	0.537¢	0.424¢.

Total operating profit @ 40¢ U.S. per # of copper + 0.30% copper cutoff grade.

Period.	Profit.	
First – 3.72 yrs.	$10,554,000.	
Next – 1.55 yrs.	4,693,000.	
Next – 3.31 yrs.	2,962,000.	
Total – 8.58 yrs.		$18,209,000

Less capital investment incl interest, working capital + inventory.	15,022,000
	3,187,000.
Add back working capital	1,275,000
Total profit before taxes	$4,462,000

Stanley Gray, P.Eng.
METALLURGICAL CONSULTANT
PHONE: FAIRFAX 7-1208

196 WEST 44TH AVENUE
VANCOUVER 18, B. C.

JAN 15 1968

(1)

January 12, 1968.

Mr. J. J. Crowhurst,
Vice-President of Operations,
Cariboo-Bell Copper Mine,
300 - 999 West Pender Street,
VANCOUVER 1, B.C.

Dear Sir: SUBJECT: Tests on Cariboo-Bell Copper Mine

A study of all the flotation tests run on this ore by the Japanese, Galligher and Cominco gives very similar results on the copper flotation on ore showing a low percentage of copper oxide. Gold and silver assays on the head samples have been run on a number of tests and also on some of the copper concentrates. However, most of the cu. concentrate was on the rougher concentrate or on a single cleaned concentrate assaying from 8 to 16% cu. From all available data head samples of .5 to .6% copper will assay .03 to .025 oz/ton of Au. and about .1 oz. Ag.

Japanese tests show two 28% cu. cts. assay to average 1.06 oz. Au. and 2.3 oz. Ag. Galligher figures for three rougher tests with cu. at 10.5% show .375 Au. and .67 Ag., while Cominco shows a cleaned test of 16.6% cu. to assay .72 Au. and 1.04 Ag. Based on these figures I estimate that flotation of unoxidized ore will give the metallurgy shown on the attached notes.

Yours truly,

"Stanley Gray".

SG:PH
Attachment

2 cc's to Mr. Crowhurst.

Stanley Gray, P.ENG.
METALLURGICAL CONSULTANT
PHONE: FAIRFAX 7-1208

196 WEST 44TH AVENUE
VANCOUVER 15, B. C.

January 12, 1968.

ESTIMATED METALLURGY:

		Assay			Distribution		
	% Wt.	% Cu.	Au.	Ag.	Cu.	Au.	Ag.
Feed	100.0	.53	.025	.10	100.0	100.0	100.0
Cu. cts.	2.0	24.0	.75	1.1	90.0	60.0	22.0
Tails	98.0	.051	.01	.07	10.00	40.00	78.0
	100.0	.53	.025	.10	100.0	100.0	100.0

The copper metallurgy, as estimated above, is based on Cominco's test of Groups 35 and 36. There is no doubt that the feed assay of .53 will be considerably higher on much of the tonnage of the ore treated and at .6% to .7% recoveries as shown above will be made, with concentrate grades up to at least 25% Cu. The gold assay will vary and may run as high as 1 oz. per ton.

A grind of 80% minus 200 M. is indicated as being the optimum and in many tests a coarser grind of around 70% gave equal results. Reagents used varied on different tests, but standard reagents such as Xanthate Z6, Dowfroth 1012, or Dow Z200 at PH of 9 gave good results.

Examination of tailings from tests on ore giving very low recoveries of 25 to 60%, by Professor Whelan, of Utah University, indicated the copper mineral present was largely conichalcite, a copper arsenate.

"Stanley Grey"

SG:PH

2 cc's to Mr. Crowhurst.

METALLURGICAL SUMMARY - JAPANESE GROUP. (3)

1. INTRODUCTION

CARIBOO-BELL MINE (CARIBOO-BELL COPPER MINES, LTD.), B.C., CANADA is being prospected by boring on the joint prospecting option of the three companies, SUMITOMO METAL MINING CO., LTD., MITSUI MINING AND SMELTING CO., LTD. and NIPPON MINING CO., LTD.

In this connection flotation test of the CARIBOO-BELL ore was requested by the Dept. of Mining, central office of Sumitomo in order to gather data for investigating the workability.

The ore samples can be classified into six categories according to the part of boring of the ore body. Basic tests were performed on the sample obtained from the upper part of the eastern ore body (Sample No. 1) which can be regarded as the most important in view of the most abundant ore reserves. The results of these basic tests made us to establish the optimal flotation conditions for the comprehensive test works which followed. For the other five samples flotation tests were conducted on the similar conditions to the optimal conditions for the Sample No. 1 (Part of the flotation tests for determining the optimal conditions for Sample No. 1 were made using Sample No. 4 instead because of the limited quantity (18 kg) of sample No. 1 which was available.)

The results of flotation tests and microscopic analysis are described below.

2. ORE SAMPLES

The ore samples which weigh ca 83 kg in the total of six kinds, were sent from the Central Research Institute of MITSUI METAL MINING AND SMELTING CO., LTD. and received as of April 28, 1967.

Each sample was as fine as -3/8 (9.423 mm) containing 2 - 6% of moisture. Specification of the samples are given in Table 1. The chemical analysis of the samples are indicated in Table 2, while the results of spectroscopic analysis for the samples No. 1 and 2 are shown in Table 3.

Table 1: Ore Samples as received

Sample No.	Spot of Sampling	Weight as received (kg)	Moisture (%)
No. 1	Upper part of the Eastern Ore Body	18.6	4.0
No. 2	Western part of the Western Ore Body	10.7	2.5
No. 3	Eastern part of the Western Ore Body	10.2	6.0
No. 4	Lower part of the Eastern Ore Body	23.0	4.0
No. 5	Off the Western block of the Western Ore Body	3.2	2.3
No. 6	Off the Eastern block of the Western Ore Body	17.1	6.5

JAPANESE GROUP.

REPORT ON MILLING TEST OF CARIBOO-BELL ORE

May, 1967

General Description

The ore samples with which we were provided (-3/8", 272 bags 1 lb per bag) were classified into six kinds (No.1 - No.6) and divided into halves by Mitsui Mining & Smelting Co., Ltd. and Sumitomo Metal Mining Co., Ltd. And the milling test was carried out at the laboratories of the two companies, respectively.

As for the ore, besides the sulphide copper minerals, the oxide copper minerals existed in so large percentage that the recovery in the flotation was very low.

According to the test results, the grade and recovery of the copper concentrate are estimated as follows:

A. Case of low grade ore

Crude ore	cu 0.41 %	Au 0.02 OZ/t	Ag 0.08 OZ/t
Concentrate	cu 28.0 %	Au 1.02 OZ/t	Ag 2.28 OZ/t
Recovery	cu 60 %	Au 45 %	Ag 25 %

B. Case of selected ore

Crude ore	cu 0.64 %	Au 0.03 OZ/t	Ag 0.10 OZ/t
Concentrate	cu 28.0 %	Au 1.11 OZ/t.	Ag 2.36 OZ/t
Recovery	cu 65 %	Au 55 %	Ag 35 %

Details of Test

1. Assays of ore (Mitsui and Sumitomo)

(Sample No.1 - No.6)

	Au(g/t)	Ag(g/t)	Cu	Sol.Cu	S	Fe	Pb	Zn	Mo
Mitsui	1.0 - 0.2	3 - 1	0.72 - 0.33	0.32 - 0.08	0.84 -0.23	8.77 -4.73	0.15 -0.08	0.08 -0.03	<0.01
Sumitomo	0.6 - 0.2	3 - 2	0.69 -0.36	0.20 - 0.02	0.78 -0.28	8.49 -5.24	0.29 -0.09	<0.01	<0.01
	SiO_2	Al_2O_3	MgO	CaO	Na_2O	K_2O	BaO	As	V
Mitsui	56-53	19-17	3-2	5 - 3			<0.01	<0.01	
Sumitomo	55-51	16-15	3-2	5 - 3	5 - 3	6-4			<0.1

2. Wi measurement (Mitsui)

Sample No. 1 Wi = 12.8 Sample No. 2 Wi = 12.8

3. Size distribution of ore (Mitsui)

Sample No.	mesh	+65	+100	+150	+200	+250	+325	-325	Total
No. 1	Weight %	0.1	3.3	10.1	14.2	9.3	13.5	49.4	100.0
	cu %		0.30	0.28	0.35	0.50	0.57	0.69	0.55
No. 2	Weight %	0.1	3.1	9.4	15.3	8.0	15.1	49.0	100.0
	cu %		0.30	0.26	0.31	0.35	0.41	0.96	0.64

4. Microscopic analysis (Sumitomo)

	No.1	No.2	No.3	No.4	No.5	No.6	mean
Oxide copper	◎	△	◎	○	△	○	◎
Chalcopyrite	◎	○	◎	◎	◎	◎	◎
Chalcocite	○		◎	○	○		○
Bornite	○		◎	○			○
Covellite	○		◎		○		○
Native copper			△		△		△
Molybdenite		△			△		△

Copper minerals in tailing

In the tailing, chalcopyrite in fine size combined with gangue remains. In the tailing of Sample No. 3, besides chalcopyrite, the middling of sulphide copper minerals also remains.

5. Flotation test (Sumitomo)

Group ① Group ②

Sample No.	No.1	No.2	No.3	No.4	No.5	No.6	mean
Sampled from	Eastern lode Upper part	Western lode Western part	Western lode Eastern part	Eastern lode Lower part	Western lode Out of western part	Western lode Out of eastern part	
Head total cu %	0.45	0.69	0.63	0.36	0.45	0.45	0.54
Sol.cu %	0.20	0.025	0.17	0.084	0.046	0.10	0.13
S %	0.22	0.78	0.35	0.33	0.42	0.41	0.45
Au g/t	0.6 (1.1)*	0.6 (1.1)*	0.2	0.2	0.2	0.2	0.32
Ag g/t	2	3	2	2	23	2	2.3
Conc. weight %	0.5	2.3	0.9	0.8	1.4	0.9	1.15
total cu %	35	25	40	30	25	30	28.9
Au g/t	108	28	–	–	–	–	
Ag g/t	62	62	–	–	–	–	
Tail total cu %	0.26	0.007	0.24	0.11	0.09	0.16	0.19
Sol. cu %	0.20	0.017	0.12	0.07	0.03	0.08	0.12
Recovery cu	39	83	58	65	77	60	61.3
Au	45	53	–	–	–	–	–
Ag	15	45	–	–	–	–	–

(Remark) * Parenthesized figures represent calculated grade.

(Optimum condition)

1. As for the collector, Z-200 (20 g/t) and KAX (50–60 g/t), and for the frother, Dowfroth #250 is recommendable.
2. The ore size in roughing stage would only have to be about –200 mesh 50 %. The flotation time requires 15 minutes, and P.H. natural.

3. The rougher concentrate requires regrinding, and the cleaning requires to be done 2-3 times.

4. The albite which floated at roughing stage, is hard to be excluded even by cleaning, and especially this is remarkable in the samples No.2, No.4 and No.5.

6. Assays of concentrate (Sumitomo)

	Au(g/t)	Ag(g/t)	Cu	Sol.Cu	S	Fe	Pb	Zn	Mo
No.1	100.8	62	36.37	0.71	31.53	31.22	0.00	0.07	0.008
No.2	28.0	62	26.37	0.37	31.34	26.04	0.00	0.08	0.055
	SiO_2	Al_2O_3	MgO	CaO	Na	K	Mn	Sb	Ti
No.1	2.04	0.94	0.19	0.22	-	-	0.009	0.09	-
No.2	4.34	1.64	0.37	0.45	0.29	0.23	0.018	-	0.12

CARIBOO-BELL COPPER MINES LIMITED

METALLURGICAL TEST RESULTS - PIT NO. 1

LOCATION	LABORATORY	GROUP NUMBER	HOLES	ELEVATION RANGE OF SAMPLE - FT.	CALCULATED % CU. (Lab.)	% RECOVERY	% NON SULPHIDE COPPER	CORRESPONDING TOTAL COPPER ASSAY %	RATIO OF NON SULPHIDE COPPER TO TOTAL COPPER
UPPER PART	Galigher	38	1,2,3,4,5,28,105,109,110	3660 - 3810	0.437	52.3	0.264	0.437	0.6041
	Cominco	33	2,3,5,28,105 and 110	3700 - 3800	0.370	58.6	N/A	N/A	-
	Cominco	41	106	3700 - 3820	0.46	63.3	0.28	0.577	0.4853
	Ottawa	1	3,4,5,6,8,11 and 12	3690 - 3820	0.39	51.5	0.226	0.390	0.5795
	Ottawa	8*	5	3690 - 3720	0.42	59.7	0.20	0.640	0.3125
	Ottawa	5	1, 4, 5 and 109	3770 - 3820	0.51	38.0	0.226	0.485	0.4660
	Ottawa	6	1, 3, 4, 5 and 105	3750 - 3780	0.64	50.0	0.233	0.686	0.3396
	Galigher	39	2,3,4,5,28,105,109,110	3510 - 3720	0.554	73.4	0.189	0.554	0.3412
	Cominco	34	2, 3, 28, 105 and 110	3620 - 3690	0.31	62.6	N/A	N/A	-
	Cominco	42	106	3560 - 3700	0.40	48.4	0.216	0.556	0.3885
	Ottawa	2	2,3,4,5,6,8, 11 and 12	3370 - 3690	0.415	76.3	0.138	0.415	0.3325
	Ottawa	9*	2	3650 - 3680	0.37	79.4	0.02	0.395	0.0506
	Ottawa	10*	3	3620 - 3650	0.48	63.7	N/A	N/A	-
	Ottawa	11*	3	3590 - 3620	0.52	73.8	N/A	N/A	-
	Ottawa	9*	4	3640 - 3670	0.53	22.1	0.670	0.970	0.6907
LOWER PART	Ottawa	11*	105	3600 - 3630	0.24	84.4	0.015	0.573	0.0262
	Ottawa	12*	105	3510 - 3600	1.00	95.4	0.050	0.822	0.0608
	Ottawa	8*	109	3690 - 3720	0.30	84.2	N/A	N/A	-
	Ottawa	9*	110	3660 - 3690	0.30	84.2	0.06	0.490	0.1224
	Ottawa	10*	110	3630 - 3660	0.34	85.7	0.05	0.465	0.1075
	Ottawa	9*	28, 105 and 110	3660 - 3690	0.64	44.6	N/A	N/A	-

*Part only of samples in Group taken for test.

N/A - Not assayed.

CARIBOO-BELL COPPER MINES LIMITED

METALLURGICAL TEST RESULTS - PIT NO. 2

LOCATION	LABORATORY	GROUP NUMBER	HOLES	ELEVATION RANGE OF SAMPLE - FT.	% COPPER	% RECOVERY	% NON SULPHIDE COPPER	CORRESPONDING TOTAL COPPER ASSAY %	RATIO OF NON SULPHIDE COPPER TO TOTAL COPPER
Upper Part	Galigher	48	21	3590 - 3680	0.756	89.7	0.018	0.756	0.0238
Lower Part	Galigher	45	21	3470 - 3560	0.796	86.1	0.069	0.796	0.0867

CARIBOO-BELL COPPER MINES LIMITED

METALLURGICAL TEST RESULTS - PIT NO. 3

LABORATORY	GROUP NUMBER	HOLES	ELEVATION RANGE OF SAMPLE - FT.	CALCULATED % CU. (Lab.)	% RECOVERY	% NON SULPHIDE COPPER	CORRESPONDING TOTAL COPPER ASSAY %	RATIO OF NON SULPHIDE COPPER TO TOTAL COPPER
Ottawa	13*	23	3785 - 3810	1.17	90.3	N/A	N/A	-
Ottawa	13*	23	3750 - 3785	1.50	94.6	N/A	N/A	-
Cominco	35	15 & 23	3750 - 3870	0.47	92.0	0.069	0.850	0.0812
Ottawa	14*	15	3730 - 3760	0.82	92.4	N/A	N/A	-
Ottawa	14*	15	3700 - 3730	0.71	86.8	N/A	N/A	-
Ottawa	15*	15	3670 - 3700	1.41	93.8	N/A	N/A	-
Ottawa	15*	29	3660 - 3690	0.37	87.7	N/A	N/A	-
Ottawa	15*	23	3660 - 3685	0.51	90.9	N/A	N/A	-
Ottawa	16*	23	3625 - 3660	0.55	93.4	N/A	N/A	-
Ottawa	16*	29	3630 - 3660	0.70	92.2	N/A	N/A	-
Ottawa	16*	15	3640 - 3670	0.76	86.7	N/A	N/A	-
Cominco	36	15,23 & 29	3600 - 3640	0.58	98.3	0.032	0.531	0.0603
Galigher	37	15,23 & 29	3540 - 3840	0.63	91.7	0.063	0.630	0.1000

*Part only of samples in Group taken for test.

CARIBOO-BELL COPPER MINES LIMITED

METALLURGICAL TEST RESULTS - PIT NO. 4

LOCATION	LABORATORY	GROUP NUMBER	HOLES	ELEVATION RANGE OF SAMPLE - FT.	CALCULATED % CU. (Lab.)	% RECOVERY	% NON SULPHIDE COPPER	CORRESPONDING TOTAL COPPER ASSAY %	RATIO - NON SULPHIDE COPPER TO TOTAL COPPER
Outside edge of Upper Part	Galigher	47	112 and 201	3710 - 3800	0.491	25.6	0.409	0.491	0.8330
Upper and Lower Part Pit 4 Plus Hole 21	Ottawa	3	21, 48 & 202	3370 - 3750	0.60	75.7	0.163	0.611	0.2668
Lower Part	Galigher	46	111	3530 - 3590	0.894	93.1	0.078	0.894	0.0872
Lower Part Pit 4 Plus Underlying Material Plus Hole 39	Ottawa	4	39, 48 & 202	3030 - 3410	0.50	81.0	0.113	0.485	0.2330

THE GALIGHER COMPANY
ESTABLISHED 1901

CABLE ADDRESS
GALSAL
TELEX 038-536

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TELEPHONE
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AREA CODE 801

January 9, 1968

Mr. J. J. Crowhurst
Vice President
Cariboo-Bell Copper Mines Ltd.
300-999 West Pender Street
Vancouver 1, B. C.
Canada

Dear Jack:

Laboratory Report on Preliminary
Testing of Your Copper Samples
Our Lot Nos. 1737 A-G

In accordance with the arrangements made with you, we are pleased to submit our laboratory report covering the preliminary tests conducted on your copper samples designated as our Lot Nos. 1737 A through G.

I. Sample Preparation and Analyses

Two separate suites of drill core samples were received at our laboratory on July 19 and September 18, 1967. The various samples were composited on representative weight bases to form composites for testing. The compositing instructions were contained in your letters to us dated July 12, July 19 and September 14, 1967.

The various composite samples were each stage-crushed through 20 mesh with a laboratory roll crusher and vibrating screen, each sample being thoroughly mixed before splitting out head samples. The head samples were submitted to the Union Assay Office of this city for analyses, they reporting the following:

Our Lot No.	Group No.	Holes	Assays			
			% Cu	% OxCu	oz Au	oz Ag
1737-A	37	Mixed*	0.630	0.063	0.015	0.09
1737-B	38	Mixed*	0.437	0.264	0.025	0.10
1737-C	39	Mixed*	0.554	0.189	0.027	0.10
1737-D	37	112 & 201	0.491	0.409	0.010	0.10
1737-E	38	21	0.756	0.018	0.025	0.10
1737-F	39	21	0.796	0.069	0.030	0.20
1737-G	40	111	0.894	0.078	0.023	None

*Refer to your letter of July 19, 1967.

With the exception of samples 1737-C and D, the average calculated head assays, based on test product assays, for the remaining samples were about 0.05% Cu higher than those shown in the above table. We are at a loss to explain these discrepancies since repeat assaying of various products resulted only minor changes from the original assays.

II. Investigation Objectives

The initial objectives of our work were discussed in a meeting with you in late June of this year, these objectives being as follows:

1. Determine the maximum copper recovery which can be obtained, by flotation, from each of the samples. Rougher flotation testing only was to be conducted.

2. Check the effects of both grinding and lime concentration regarding the floatability of the copper minerals.

It was observed, in some instances, that low copper recoveries were obtained because of the presence of non-sulfide copper minerals. Our work was then extended to include preliminary testing regarding attempts to recover these non-sulfide copper minerals.

III. Summary and Conclusions

The testing of these seven samples has shown that the sulfide copper minerals can be easily recovered using conventional sulfide copper flotation methods. The indicated sulfide copper recoveries in rougher flotation are shown in the following tabulation:

Lot No.	Test No.	% of Cu as Sulfide Cu*	% Total Cu Rec.	% Rec. of Sulfide Cu**
1737-A	5	90.0	92.0	102.2
1737-B	1	39.6	52.3	132.0
1737-C	2	65.9	73.4	111.3
1737-D	1	16.6	13.6	81.9
1737-E	5	97.6	88.4	90.6
1737-F	5	91.3	85.3	93.4
1737-G	5	91.3	92.2	101.0

*Based on the total copper and oxide copper assays for the head samples.
**Determined by dividing the % total Cu recovery by the % of Cu as sulfide Cu.

In the above tabulation it is seen that more than 100% recovery of the sulfide copper is indicated with some samples. These questionable figures could be due either to assaying discrepancies with the head samples, or some of the oxide copper floating with the sulfides. Nonetheless, the above figures, plus our visual observations, indicated that the sulfide copper was "fast floating".

The addition of lime to the grinding circuits, to a pH of about 8.5 to 9.0, was observed to be beneficial, generally speaking, regarding the recovery of the copper. A grind of about 45% minus 325 mesh was found to be satisfactory for rougher flotation for most of the samples tested. Finer grinding, to about 65% minus 325 mesh, showed more favorable copper extractions with the 1737-E and F samples.

Only very limited testing was conducted regarding sulfide copper flotation, the main problem being that of developing a recovery process for the oxide copper minerals. Preliminary testing for oxide copper recovery was conducted exploring the possibilities of concentration by flotation, gravity concentration and acid leaching. Gravity concentration (tabling) yielded completely negative results. Oxide copper flotation recovered the visible, green oxide copper minerals, but additional copper recovery was very low using this process. Acid leaching probably would result in the greatest oxide copper extraction but, the single test which was conducted here indicated an acid consumption in excess of 7 pounds of acid per pound of copper extracted, recovery being about 40% (Lot No. 1737-D, Test No. 2). Further testing would possibly show greater extractions. Microscopic examinations, conducted by Dr. J. A. Whelan of the University of Utah, of the high copper tailing samples failed to reveal the identity of the oxide copper minerals remaining therein.

In accordance with your request, no further testing of the seven samples will be conducted.

IV. Discussion of Test Results

A. Testing of Sample 1737-A. Eight tests were conducted with this sample investigating the effects of varying grind and lime concentrations on sulfide flotation. The head assays for this sample (0.630% Cu and 0.063% OxCu) indicated that 90% of the copper occurred as sulfide copper.