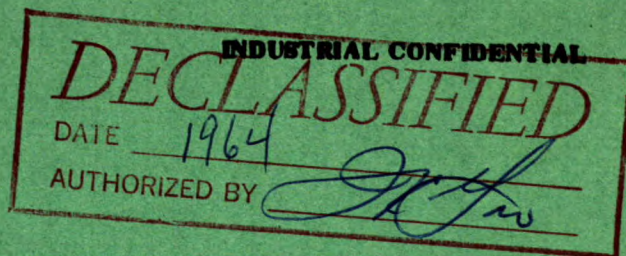


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DEPARTMENT OF MINES AND TECHNICAL SURVEYS

OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 63-23

**INVESTIGATION OF AN OXIDIZED COPPER
ORE FROM THE PROPERTY OF FRIDAY
CREEK DEVELOPMENT CO. LTD.,
NEAR PRINCETON, B. C.**

by

R. P. BAILEY

MINERAL PROCESSING DIVISION

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MARCH 31, 1963

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INVESTIGATION OF AN OXIDIZED COPPER ORE FROM
THE PROPERTY OF FRIDAY CREEK DEVELOPMENT CO. LTD.,
NEAR PRINCETON, B.C.

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R. P. Bailey^{*}

SUMMARY OF RESULTS

The valuable metals in this ore were copper (1.09%), gold (0.045 oz/ton), silver (0.26 oz/ton), and palladium (0.016 oz/ton). About 60% of the copper occurred as sulphide mineral, principally bornite with a little chalcopyrite. The other 40% was in oxidized form, chiefly as malachite. Gold, silver and palladium were mainly associated with sulphides.

No satisfactory recovery of copper, gold and silver could be made by tabling. Most effective procedure was two-stage flotation which produced a rich sulphide copper concentrate containing gold, silver and palladium, and a low-grade oxide copper concentrate. From the latter, cement copper was produced by the leach-precipitation technique. Typical analyses of the final concentrates, and overall recoveries were as follows:

Product	Cu %	Au oz/ton	Ag oz/ton	Pd oz/ton
Sulphide concentrate	41.2	2.5	10.0	0.84
Cement copper "	56.3	-	-	-
Recovery %	82.4	80.5	78.2	57.6

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INTRODUCTION

In a letter dated November 6, 1961, Mr. Douglas F. Hamelin, Manager, Friday Creek Development Co. Ltd., 1055 Eyremount Drive, West Vancouver, B.C., asked the Mines Branch to do an investigation on a sample of copper-gold ore from the Company's property.

Location of Property

The Company holds under option a group of claims alongside Friday Creek, about 14 miles south of Princeton, B.C., on the east side of the Hope-Princeton highway at an elevation of about 3,200 feet.

History^{*}

As far back as 1929 work was being done on these claims by A. E. Wheeler and associates. At that time ore was reported to have been found in two of several tunnels and in an open cut on the opposite side of the creek. In 1958 Phelps Dodge Corporation of Canada Limited extended bulldozer trenches to north and south of Friday Creek for distances of 1400 and 3600 feet, respectively. Work done in 1960 by the present company included bulldozer trenching, 1000 feet of diamond drilling and a geophysical survey. Further trenching and another 710 feet of diamond drilling were done in 1961, as well as stripping to bedrock over a wide area on the south bank of the creek.

Purpose of Investigation

The company proposed to treat ore in a 50 ton/day table concentrating plant and wished to know if satisfactory recovery of the copper might be expected. If the investigation showed that table concentration was unsatisfactory, an alternate flowsheet was requested.

Shipment

Two bags of crushed ore, weighing about 229 pounds, were received on November 16, 1961. This sample was reported to have been taken over a width of 25 feet from three separate locations throughout the length of the stripped area. Maximum size was about $\frac{1}{2}$ inch, but 10-15% of the material was very fine.

*From B.C. Minister of Mines Annual Reports, 1929, page 277; 1960, page 56; 1961, page 56.

Sampling and Analysis

The ore as received was split into two equal parts. One part was crushed to minus 10 mesh and head samples were riffled out for mineralogical examination, chemical analysis and semi-quantitative spectrographic analysis. The remainder of the minus 10 mesh fraction was set aside for test work.

Chemical analysis* of the head sample gave the following results:

Gold	-	0.07 (0.045) oz/ton	Copper	-	1.09%
Silver	-	0.28 (0.26) "	Iron (soluble)	-	5.86%
Palladium	-	0.016 "	Sulphur	-	0.27%
Platinum	-	not detected	Insoluble	-	74.45%
Rhodium	-	" "			

*From Internal Reports MS-AC-61-1009, 62-55 and 62-1225.

() Average calculated value from all tests.

Elements indicated by semi-quantitative spectrographic analysis* are listed below in approximate order of decreasing abundance:

I	Si	(principal constituent)
II	Fe, Ca, Mg, Al, Na	(5 - 1%)
III	Cu, Mn, Ti, Ba	(1 - 0.2%)
IV	Sr, Cr, V, Ni, Mo	(0.05 - 0.01%)
V	Co, Ag, Sn, Bi, Zn	(0.005% - trace)

*From Internal Report MS-61-954, December 1, 1961, by Miss E. M. Kranck.

MINERALOGICAL EXAMINATION**

A portion of the head sample, crushed to minus 10 mesh, was submitted to the Mineralogical Section of the Mineral Sciences Division for microscopic examination.

Results of Examination

A -100+150 mesh fraction of the head sample was separated into fractions by heavy liquids and a hand magnet. The mineralogy of each fraction was determined by microscopical studies and X-ray diffraction analyses. The mineral content, in per cent by weight, was as follows:

**From Mineral Sciences Division Internal Report MS-61-867, by W. Petruk, December 14, 1961.

<u>Metallic</u>		<u>Non-Metallic</u>	
Bornite	0.5%	Calcite	4%
Chalcopyrite	0.2%	Sphene	8%
Malachite	0.15%	Biotite	12%
Pyrite, sphalerite,)	0.15%	Quartz	12%
pyrrhotite, ilmenite)		Feldspar	25%
Magnetite, hematite	4%	Pyroxene	34%

Two polished sections were prepared from seven selected pieces of $\frac{1}{2}$ inch ore and the metallic minerals were identified by microscopical and X-ray diffraction studies.

Polished section studies indicate that the bornite, covellite and sphalerite occur in pegmatite, and the magnetite and hematite occur in pyroxenite and diorite. The other metallic minerals present in the head sample were not observed in the polished sections.

The bornite occurs in massive forms and as irregular shaped grains that range between 0.02 mm and 1.0 mm in diameter. It is partially altered to covellite along fractures and grain boundaries, and contains several inclusions of sphalerite that range between 5 microns and 100 microns in diameter.

The magnetite occurs as subhedral grains that range between 20 and 300 microns in diameter and is partially replaced by hematite.

OUTLINE OF INVESTIGATION

Because of Mr. Hamelin's hope that gravity concentration might be effective, several shaking table tests were done. Low recovery due to loss of oxide copper mineral in the table tailing prompted leaching tests on this material and on the original ore, but acid consumption was prohibitively high. A high intensity wet magnetic separation test effected no practical concentration. All subsequent tests involved differential flotation for recovery of separate sulphide and oxide concentrates and acid leaching of the latter. Limited work was done on recovery of copper from solution by precipitation with iron powder.

DETAILS OF INVESTIGATION

Test 1, Tabling

In this preliminary test, a 4000 gram sample of minus 10 mesh ore was first deslimed on a 24" x 48" Deister shaking table. The coarse fraction was stage ground to minus 48 mesh and tabled to recover a bulk concentrate from which magnetite was removed by treatment on the Crockett magnetic separator. The non-magnetic fraction was re-tabled to produce a cleaned concentrate and a tailing. Products were analyzed for copper only.

TABLE 1:

Results of Preliminary Tabling Test

Product	Weight %	Assays* Cu, %	Distn % Cu
Cu cl conc	2.0	17.95	30.7
Cl tailing	36.7	0.84	26.4
Magnetics	5.1	0.25	1.1
Ro tailing	37.9	0.72	23.3
Slimes	18.3	1.18	18.5
Feed (calcd)	100.0	1.17	100.0

* From Internal Reports MS-AC-61-1009, 1049 and 1055.

Test 2, Magnetic Separation and Tabling

Because of the tendency of the magnetite to report in the table concentrate, a 4000 gram sample of minus 65 mesh ore was first treated on the Crockett wet magnetic separator to remove magnetics. The non-magnetic fraction was then tabled on the laboratory Deister to produce a copper concentrate and a tailing.

TABLE 2

Results of Magnetic Separation and Tabling

Product	Weight %	Assays ^{**}			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Magnetics (Fe, 50.0%)	5.2	0.10			0.5		
Cu conc	1.5	23.33	1.98	5.27	31.5	66.0 [*]	30.4 [*]
Table tailing	93.3	0.81			68.0		
Head (calcd)	100.0	1.11			100.0		

* Calculated by difference from average calculated head assays.

** From Internal Reports MS-AC-61-1049 and 62-55.

Test 3, Tabling and Magnetic Separation

By tabling 4000 grams of minus 65 mesh ore a gravity concentrate was obtained. Magnetite was removed from this by Crockett wet magnetic separation to produce a copper concentrate. Table tailings were collected separately as coarse and slime fractions.

TABLE 3

Results of Tabling and Magnetic Separation

Product	Weight %	Assays ^{**}			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Magnetics (Fe, 63.0%)	3.2	0.2			0.5		
Cu conc	1.1	29.38	1.97	6.55	27.8	48.2 [*]	27.7 [*]
Table tailing - coarse	64.9	0.72			40.2		
" " - slime	30.8	1.19			31.5		
Head (calcd)	100.0	1.16			100.0		

* Calculated by difference from average calculated head assays.

** From Internal Reports MS-AC-61-1049, 62-55 and 105.

Test 4, High-Intensity Wet Magnetic Separation

A 2000 gram sample, ground to about 70% minus 200 mesh, was treated in a Jones high-intensity wet magnetic separator, first at minimum intensity to remove strong magnetics, and then at near-maximum intensity to produce a concentrate, middling and non-magnetic tailing. As shown by the results in Table 4, no effective concentration was achieved.

TABLE 4

Results of High-Intensity Wet Magnetic Separation

Product	Weight %	Assays **			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Mag, 0 amp (Fe, 37.0%)	6.0	0.61			3.0		
Mag, 25 amp	18.5	1.65	0.055	0.37	25.6	22.6*	26.3*
Midd, 25 amp	40.8	1.29			44.1		
Non-mag tailing	34.7	0.94			27.3		
Head (calcd)	100.0	1.19			100.0		

* Calculated by difference from average calculated head assays.

** From Internal Report MS-AC-62-346.

Test 5, Leaching of Table Products and Ore Feed

The low copper recovery by tabling in Test 3 and the high copper loss in the table tailing, prompted sulphuric acid leaching tests on the coarse and slime fractions to determine the extent of the acid-soluble (i.e. oxide) copper.

These tests were done on 100 gram samples slurried in water to about 25% solids. Sulphuric acid (20% wt/vol) was added gradually to pH 2.0-1.8, and agitation continued for an additional 15 minutes. Residues were recovered by vacuum filtration and washed on the filter several times by displacement. Filtrate and washings were combined for analysis.

TABLE 5
Results of Leaching Tests

Product	Cu %	Min pH	Cu Distribution %		H ₂ SO ₄ (100%) Consumed	
			Extracted (oxide)	Residue (Sulphide)	lb/ton feed	lb/lb Cu extracted
Table tailing - coarse	0.72	1.8	46.8	53.2	80	9.1
" " - slime	1.19	1.8	71.5	28.5	210	16.1
Table feed (-65m ore)	1.16	2.0	40.5	59.5	120	14.4

The high acid consumption, which is due to the presence of about 4% calcite in the ore, makes it uneconomical to recover oxide copper by direct leaching of table tailing or of ore feed. Because of this, and the difficulty of recovering fine sulphide copper by tabling, subsequent test work was based on flotation as the most practical method for recovering the copper in this ore.

Test 6, Sulphide Flotation

A preliminary bulk sulphide flotation test was done on 2000 grams of minus 65 mesh ore (48.0% minus 200 mesh) under the following conditions:

<u>Operation</u>	<u>Reagents,</u>	<u>lb/ton</u>	<u>Time, min</u>	<u>pH</u>
Conditioning	Soda ash	0.2	3	9.0
	Z-6	0.7		
Flotation	Pine oil	0.2	10	

The concentrate, rich in bornite, was cleaned once, without further addition of reagents, by refloating for five minutes. Products were analyzed for copper only.

TABLE 6
Results of Sulphide Flotation

Product	Weight %	Assays* Cu %	Distn, Cu %
Sulph cl conc	1.1	45.87	47.9
" " tailing	1.0	9.08	8.6
Flotn tailing	97.9	0.47	43.5
Head (calcd)	100.0	1.05	100.0

* From Internal Report MS-AC-62-109.

Test 7, Sulphide and Oxide Flotation

Under conditions similar to Test 6, a sulphide flotation concentrate was taken off and cleaned once. From the sulphide flotation tailing a rougher oxide concentrate, dull green in color, was floated under the following conditions:

<u>Operation</u>	<u>Reagents,</u>	<u>lb/ton</u>	<u>Time</u> <u>min</u>	<u>pH</u>
Conditioning	Na ₂ S	1.0	3	8.8
	NaHCO ₃	1.0		
	Aero Promoter 404	0.3		
Flotation	Pine oil	0.1	10	

TABLE 7
Results of Sulphide - Oxide Flotation

Product	Weight %	Assays *			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulph cl conc	1.6	33.60	2.36	9.19	48.1	79.0	54.3
" " tailing	0.8	2.75	0.20	0.69	2.0	3.4	2.0
Oxide conc	1.5	11.53	0.11	1.50	15.5	3.5	8.3
Flotn tailing	96.1	0.40	0.007	0.10	34.4	14.1	35.4
Head (calcd)	100.0	1.12	0.048	0.27	100.0	100.0	100.0

* From Internal Report MS-AC-62-160 and 161.

Because of the relatively low grade of the oxide copper concentrate and its mixed nature (i.e. containing considerable oxide-coated sulphides), acid leaching tests were done to assess the practicability of recovering the oxide copper by that means. To determine approximate ratios of oxide to sulphide copper, and to compare acid consumption, similar tests were made on the flotation tailing and the sulphide cleaner concentrate by the procedure of Test 5.

TABLE 7a
Results of Leaching Tests on Flotation Products

Product	Min pH	Assays* Cu %		Cu Distribution %		H ₂ SO ₄ consumed lb/lb Sol/Cu
		Feed	Residue	Extracted (Oxide)	Residue (Sulphide)	
Oxide conc	2.0	11.53	3.88	66.5	33.5	3.4
Sulph cl tailing	1.9	2.75	1.64	41.0	59.0	14.0
Final flotn "	1.8	0.40	0.18	60.0	40.0	21.8

* From Internal Report MS-AC-62-160.

Test 8, Sulphide and Oxide Flotation

As in the two previous tests, ore at 48% minus 200 mesh was used. However, for the sulphide flotation step, reagents were changed to ensure recovery of any free gold and gold-bearing tarnished sulphides. One cleaning step was used to produce a high grade copper concentrate. For the oxide flotation step a stronger promoter was used, as well as sulphidization, in an attempt to increase recovery of oxidized copper minerals. Flotation conditions and reagents used were as follows:

<u>Operation</u>	<u>Reagents</u>	<u>lb/ton</u>	<u>Time (min)</u>	<u>pH</u>
Sulphide:				
Conditioning	Soda ash	0.5	3	9.0
	Aero Xanthate 301	0.5		
	Aerofloat 208	0.05		
Flotation	Pine oil	0.2	6	
	Dowfroth 250	0.05		
C1 conditioning	Soda ash	0.3	3	9.9
	Sod. silicate	1.0		
	Aero Xanthate 301	0.2		
	Aerofloat 208	0.02		
C1 flotation			6	
Oxide:				
Conditioning	Na ₂ S	1.5	4	9.0
	NaHCO ₃	1.0		
	Aero Promoter 425	0.3		
Flotation	Pine oil	0.1	10	

TABLE 8

Results of Sulphide - Oxide Flotation

Product	Weight %	Assays *			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulph c1 conc	1.4	39.90	3.06	11.16	51.0	77.2	60.0
" " tailing	1.0	7.04	0.34	1.29	6.4	6.1	4.9
Oxide conc	2.2	9.50	0.12	1.19	19.1	4.7	9.5
Flotn tailing	95.4	0.27	0.007	0.07	23.5	12.0	25.6
Head (calcd)	100.0	1.09	0.055	0.26	100.0	100.0	100.0

* From Internal Report MS-AC-62-160.

Because of the greater recovery of both sulphide and oxide copper, and the significant increase in the accompanying gold and silver, the reagents used in this test were adopted as "standard" for all subsequent flotation tests.

Test 9, Tabling of Flotation Tailing

To determine the association of the considerable copper left in the flotation tailing, about 4000 grams of combined tailings from Tests 7 and 8 were treated by tabling and wet magnetic separation according to the flowsheet shown in Figure 1.

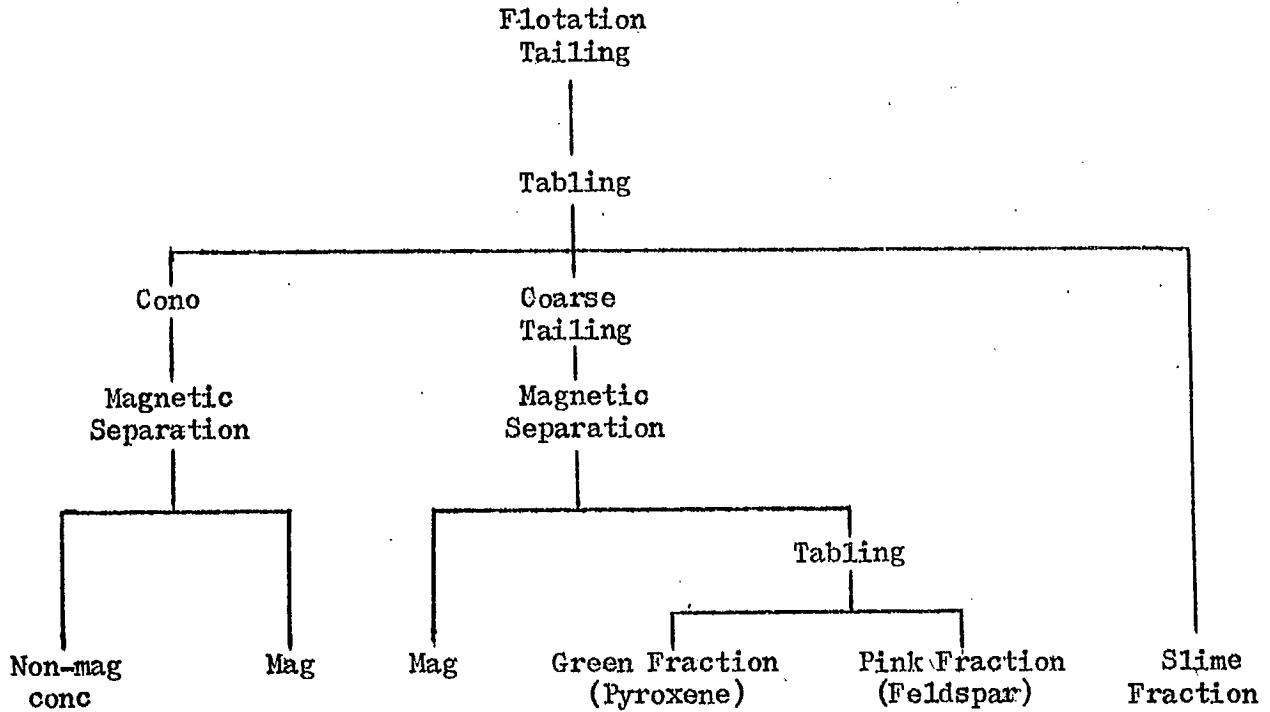


Fig. 1 - Flowsheet for Treatment of Flotation Tailing

TABLE 9
Results of Tabling Test on Flotation Tailing

Product	Weight %	Assays [*] %			Cu Distn %
		Cu	Fe	Insol	
Mag table conc	3.1	0.02	64.2	6.4	0.2
" " tail, coarse	2.0	0.08	27.3	-	0.4
Mag combined	5.1	0.04	49.7	-	0.6
Table conc, non-mag	0.2	2.25	-	-	1.3
" tail, coarse green	33.0	0.23	-	-	22.0
" " " pink	31.8	0.25	-	-	23.1
" " slimes	29.9	0.61	-	-	53.0
Head (calcd)	100.0	0.34	-	-	100.0

* From Internal Report MS-AC-62-218.

Sulphuric acid leaching tests were done on the three major tailing fractions, namely, green (pyroxenite), pink (pegmatite), and slime to determine the proportion of oxide copper in each and the acid consumption.

TABLE 9a
Results of Leaching Tests on Table Tailing Fractions

Product	Min pH	Assays [*] Cu %		Cu extracted %	100% H ₂ SO ₄ consumed lb/lb Cu
		Feed	Residue		
Coarse green (pyroxenite)	2.0	0.23	0.09	60.2	24.8
Coarse pink (pegmatite)	2.0	0.25	0.10	60.0	34.7
Slime	1.8	0.61	0.24	58.6	26.4

* From Internal Report MS-AC-62-218.

The results of the tabling and leaching tests summarized in Tables 9 and 9a show that:

- (1) little additional copper can be recovered by tabling the flotation tailing;
- (2) over 50% of the copper lost in the flotation tailing is in the extreme fines;
- (3) the acid-soluble (oxide) copper constitutes a uniform 60% of each of the major tailing fractions, but cannot be acid-leached economically from any fraction.

Subsequent test work was limited to flotation aimed at improving recovery, both by control of grinding and by staged addition of sodium sulphide in the oxide flotation. Acid leaching tests were made on oxide concentrates and other flotation products.

Test 10, Sulphide and Oxide Flotation

Objectives of this test were to study the effect of finer grinding on copper recovery, and to produce a cleaned oxide copper concentrate of commercial grade. Ore was ground to 73.6% minus 200 mesh. After sulphide flotation, staged additions of reagents were used to promote recovery of oxide copper minerals. Each concentrate was cleaned once. The results of this test are shown in Table 10. Reagents used and flotation conditions for this test (and, with minor variations, for all subsequent tests) were as follows:

<u>Operation</u>	<u>Reagents</u>	<u>lb/ton</u>	<u>Time (min)</u>	<u>pH</u>
Sulphide:				
Conditioning	Soda ash	1.0	3	9.2
	Aero Xanthate 301	0.2		
	Aerofloat 208	0.05		
Flotation	Pine oil	0.2	6	
	Dowfroth 250	0.05		
C1 conditioning	Soda ash	0.3	3	10.1
	Sod. silicate	1.0		
	301	0.1		
	208	0.03		
C1 flotation			6	
Oxide:				
Conditioning (1)	Na ₂ S	1.5	6	9.0
	NaHCO ₃	1.0		
	Aero Promoter 425	0.2		
Flotation			6	

<u>Operation</u>	<u>Reagents,</u>	<u>lb/ton</u>	<u>Time</u> <u>(min)</u>	<u>pH</u>
Oxide: (cont'd)				
Conditioning (2)	Na ₂ S	0.5	3	
	425	0.1		
Flotation			4	
Cl conditioning	Na ₂ S	0.25	3	9.3
	Sod. silicate	0.5		
	NaHCO ₃	0.5		
	425	0.1		
Cl flotation			5	

TABLE 10

Results of Sulphide-Oxide Flotation

Product	Weight %	Assays *			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulph cl conc	1.5	34.13	1.40	9.77	44.9	57.7	62.1
" " tailing	2.4	2.12	0.26	0.59	4.5	17.0	6.0
Oxide cl conc	0.9	24.17	0.28	2.40	19.1	6.9	9.2
" " tailing	1.8	2.78	0.11	0.64	4.4	5.5	4.9
Flot tailing	93.4	0.33	0.005	0.045	27.1	12.9	17.8
Head (calcd)	100.0	1.14	0.036	0.24	100.0	100.0	100.0

* From Internal Report MS-AC-62-932.

The slimes produced by finer grinding seemed detrimental to recovery of sulphide copper. However, the staged addition of reagents increased the recovery of oxide copper concentrate. Despite the acceptable grade of the latter, the difficulty of smelting such fine material might severely limit its market. Therefore, in subsequent tests the oxide copper was recovered only as rougher concentrate suitable for conventional leaching.

Test 11, Flotation, Leaching and Cementation

Using ore ground to 75% minus 200 mesh, the flotation conditions of Test 10 were repeated to produce a rougher sulphide and a rougher oxide concentrate. The latter was leached with sulphuric acid for thirty minutes at room temperature and pH 1.8. To the leach slurry was added the rougher sulphide concentrate containing some oxide copper mineral. The pregnant copper solution was separated by decantation and filtering. The washed leach residue was subjected to a cleaner sulphide flotation under the same conditions as in Test 10. From the pregnant solution, at room temperature and pH 2.7, copper was precipitated by addition of 12.5 grams of iron powder (65% minus 200 mesh), equivalent to about 2.7 times the theoretical requirement. Without separating the excess iron, the cement copper was recovered by settling and filtration.

TABLE 11
Results of Flotation, Leaching and Cementation

Product	Weight %	Assays *			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulphid conc	1.3	37.52	1.72	10.96	48.8	74.8	61.1
" " tailing	2.6	1.27	0.07	0.91	3.3	6.1	10.2
Cement copper	0.7	34.34	-	-	24.0	-	-
Flotation tailing	95.4	0.25	0.006	0.07	23.9	19.1	28.7
Head (calcd)	100.0	1.00	0.03	0.23	100.0	100.0	100.0

* From Internal Report MS-AC-62-995.

Acid consumed in leaching was 3.9 pounds of 100% H₂SO₄ per pound of copper extracted. Precipitation efficiency was 99%.

Addition of the sulphide rougher concentrate to the oxide leach slurry served two purposes:

- (1) to neutralize excess acid and so to reduce iron powder consumption in the copper precipitation step;
- (2) to clean oxide-coated sulphides and facilitate their flotation, as shown by the relatively low copper-gold values in the sulphide cleaner tailing.

Test 12, Flotation of Deslimed Ore

For this test, a 2000 g sample of minus 10 mesh ore was water-washed to remove natural slimes. The coarse fraction was ground to 65.8% minus 200 mesh for flotation under conditions similar to Test 10. The sulphide concentrate was cleaned once. Leaching tests were done on the slimes, oxide concentrate and flotation tailing.

TABLE 12
Results of Flotation Test on Deslimed Ore

Product	Weight %	Assays*			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Slimes	8.7	1.31	0.02	0.37	10.1	4.6	12.3
Sulph cl conc	1.2	41.76	2.03	11.37	44.5	64.6	52.0
" " tailing	0.7	2.80	0.45	0.96	1.7	8.4	2.5
Oxide conc	1.5	20.59	0.27	2.31	27.4	10.7	13.2
Flot tailing	87.9	0.21	0.005	0.06	16.3	11.7	20.0
Head (calcd)	100.0	1.13	0.038	0.26	100.0	100.0	100.0

* From Internal Reports MS-AC-62-212 and 248.

TABLE 12a
Leaching Tests on Flotation Products

Product	Min pH	Cu * %		Cu extracted %	Acid consumed (100% H ₂ SO ₄) lb/lb Cu
		Feed	Residue		
Slimes	1.7	1.31	0.37	74.9	11.0
Oxide conc	1.7	20.59	3.28	84.0	2.5
Flot tailing	1.8	0.21	0.06	72.6	34.2

* From Internal Report MS-AC-62-248.

On the basis of deslimed feed to flotation, recoveries of copper, gold and silver were 81.8%, 87.7% and 77% respectively. However, losses in the slime fraction reduced overall recoveries to 73.6%, 83.7% and 67.7%. The prohibitive cost of leaching these slimes, and the relatively high residual copper (mostly sulphide) prompted a flotation test on the slimes.

Test 13, Bulk Flotation from Slimes

For this test, about 500 grams of slimes were washed from 6300 grams of minus 10 mesh ore. After being filtered and dried, the slimes were repulped in a 500-gram Denver Sub-A flotation cell. Three concentrates (A, B and C) were recovered by staged addition of reagents according to the following procedure:

Operation	Concentrate		
	A	B	C
Reagents, lb/ton feed			
Aerosol 18	0.05	--	--
Sodium silicate	0.5	--	--
Sodium sulphide	0.8	0.4	0.2
Aeropromoter 425	0.2	0.1	0.05
Pine oil	0.02	--	--
Dowfroth 250	0.02	--	--
Conditioning time, min	6	3	3
Flotation time, min	4	3	3
pH	7.9		

Each of the three concentrates and the flotation tailing were leached separately to determine acid consumption and extractability of the copper.

TABLE 13

Results of Flotation from Slimes and Leaching of Products

Product	Weight %	Cu [*] %	Distribution %				Acid consumed (100% H ₂ SO ₄) 1b/1b Cu
			Oxide	Sulph	Total	Orig ore basis	
Conc A	9.0	6.6	29.7	12.3	42.0	4.2	12.3
Conc B	11.1	2.4	16.3	2.5	18.8	1.9	16.7
Conc C	6.9	1.7	7.0	1.3	8.3	0.8	20.0
Conc A+B+C	27.0	3.6	53.0	16.1	69.1	6.9	14.6
Flot tailing	73.0	0.3	23.3	7.6	30.9	3.2	40.6
Feed (calcd)	100.0	1.4	76.3	23.7	100.0	10.1	22.2

* From Internal Report MS-AC-62-304.

Despite recovery of a bulk concentrate containing 69.1% of the slime copper (equivalent to 6.9% additional recovery on original ore basis), its low grade and the high acid consumption in extracting the oxide fraction (76.5%) discouraged further work on slime treatment. However, the success in floating both sulphide and oxide copper from deslimed ore in Test 12 prompted investigation of stage grinding, without desliming, as a means of increasing recovery by flotation.

Tests 14 and 15, Flotation after Staged Grinding

For these tests, two 2000 gram lots of minus .10 mesh ore were stage-ground in the laboratory ball mill to pass 65 mesh and 100 mesh screens respectively. The minus 200 mesh size fractions of the flotation feeds were 59.1% and 68.8% respectively. Flotation procedure was similar to that of Test 10 except that the bulk oxide concentrate was not cleaned. After each test, the cleaner sulphide tailing and the bulk oxide concentrate were leached separately to determine acid requirement and amount of extractable oxide copper. Results are summarized in Tables 14 and 15.

TABLE 14

Flotation Test on Ore Stage-Ground to -65 mesh (Test 14)

Product	Weight %	Assays *			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulph cl conc	1.35	38.17	2.62	11.42	44.6	66.2	54.6
" " tailing	0.91	4.21	0.48	1.72	3.3	8.2	5.5
Oxide conc	2.09	17.38	0.15	2.19	31.4	5.9	16.2
Flot tailing	95.65	0.25	0.011	0.07	20.7	19.7	23.7
Feed (calcd)	100.00	1.16	0.05	0.28	100.0	100.0	100.0

* From Internal Report MS-AC-62-703.

TABLE 15

Flotation Test on Ore Stage-Ground to -100 mesh (Test 15)

Product	Weight %	Assays *			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulph cl conc	1.39	38.28	3.14	11.92	47.2	75.5	61.1
" " tailing	2.49	2.33	0.075	0.85	5.2	3.3	7.8
Oxide conc	2.96	11.20	0.10	1.27	29.4	5.1	13.9
Flot tailing	93.16	0.22	0.01	0.05	18.2	16.1	17.2
Feed (calcd)	100.00	1.13	0.058	0.27	100.0	100.0	100.0

* From Internal Report MS-AC-62-703.

In Test 15, combined rougher recoveries of 81.8%, 83.9% and 82.8% for copper, gold and silver respectively were achieved. These compare very favourably with those obtained in Test 12 by flotation of the deslimed fraction only, under near-ideal conditions. Thus, by staged grinding to minus 100 mesh, the problems of slime treatment are avoided.

To determine acid consumption and the amount of extractable oxide copper, the sulphide cleaner tailing and oxide concentrate from both tests were leached at room temperature by gradual addition of sulphuric acid to a slurry at about 12% solids with results as shown in Table 15a.

TABLE 15a

Leaching Test on Products of Tests 14 and 15

Test No.	Product	Min pH	Assays [*] Cu %		Cu extracted %	Acid consumed (100% H ₂ SO ₄) lb/lb Cu
			Feed	Residue		
14	Sulph cl tailing	1.7	4.21	2.55	39.5	11.9
	Oxide conc	1.7	17.38	2.39	86.3	2.6
15	Sulph cl tailing	1.7	2.33	0.66	71.9	8.5
	Oxide conc	1.7	11.20	1.50	86.7	2.6

^{*} From Internal Reports MS-AC-62-638, 644 and 703.

These results indicate the high percentage of soluble copper in the oxide concentrate and the acid economy in its leaching. The copper in the leach residue, largely in sulphide form, is of the same order as that in the sulphide cleaner tailing and would be suitable for recovery in the sulphide rougher flotation step. Although leaching of the sulphide cleaner tailing is not practical from the standpoint of acid consumption, the sulphide copper in the residue would be oxide-free and, therefore, amenable to sulphide flotation.

Test 16, Flotation of Cement Copper

Because of the conventional nature of this flotation, one test only was done to determine if a satisfactory grade of cement copper could be produced. A "synthetic" leach slurry was prepared by repulping 75 grams of oxide flotation tailing in two litres of copper solution from Test 14. To minimize consumption of iron powder, the pH of the slurry was increased from 1.7 to 3.1 by addition of lime. Ten grams of iron powder (64.8% minus 200 mesh) were added slowly. After the slurry was agitated vigorously for 30 minutes, it was transferred to a Fagergren 500-gram cell for flotation

under the following conditions:

<u>Operation</u>	<u>Reagents</u>	<u>Time (min)</u>	<u>pH</u>
Conditioning	Aero Xanthate 301 - 0.1 g	3	5.0
	Aerofloat 208 - 0.05 g		
Flotation	Dowfroth 250 - 0.05 g	10	

Filtrates and washings from the flotation concentrate and tailing were combined and analyzed for copper.

TABLE 16
Results of Copper Cementation and Flotation

Product	Volume or Weight	Cu		Fe	
		Analysis*	Grams	Analysis	Grams
Added:					
Copper solution	2000 mls	1.61 g/l	3.22	-	-
Oxide flot tailing	75 g	0.25 %	0.19	3.48 %	2.6
Iron powder	10 g	-	-	-	10.0
Totals			3.41		12.6
Recovered:					
Flot conc	4.5 g	56.34 %	2.54	5.13 %	0.2
" tailing	85.3 g	0.96 %	0.82	10.51 %	9.0
Filtrate and washings	3300 mls	0.004 g/l	0.01	1.0 g/l	3.3
Totals			3.37		12.5

* From Internal Report MS-AC-62-649.

Copper precipitation from the solution was 99.6% complete, but only 79% was recovered in the flotation concentrate. This low recovery was due to the low density of the flotation pulp, the small amount of copper involved and the presence of considerable interfering fine gangue in the tailing used to make the "synthetic" slurry. The copper content of the concentrate, 56.34%, is higher than that usually obtained by flotation of cement copper in plant practice.

Test 17, Flotation and Leaching

Purposes of this final test were -

- (1) to produce sufficient sulphide concentrate for precious metals analysis;
- (2) to determine the effect of the addition of Aero Xanthate 301 reagent (for scavenging sulphides) in the oxide flotation;
- (3) within the limitations of the batch test, to follow a procedure suitable for mill practice. This is illustrated by the flowsheet in Figure 2.

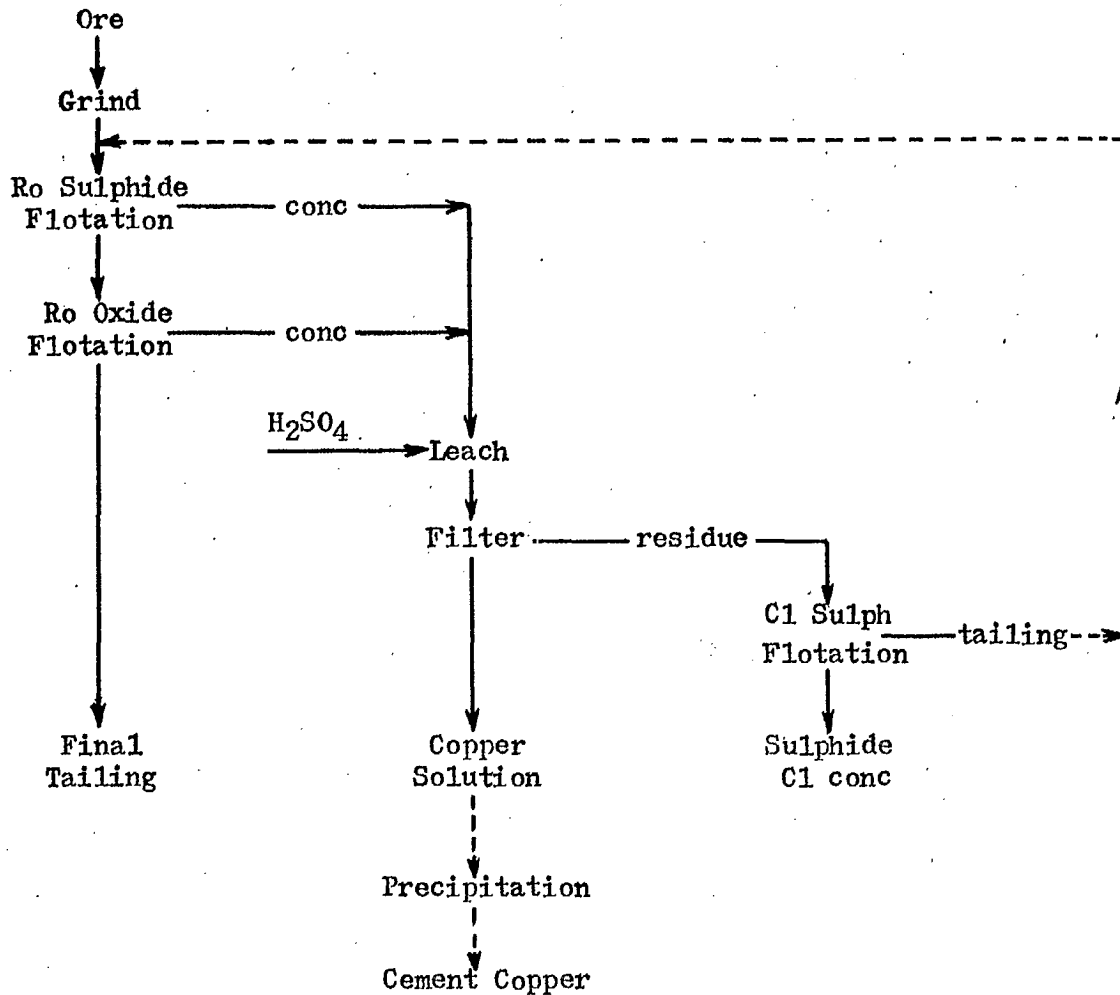


Figure 2. Flowsheet for Test 17

Two 2000-gram lots of minus 10 mesh ore were ground, without staging or classification, to 72.4% minus 200 mesh. By flotation of the first lot under the conditions of Test 10, rougher sulphide and rougher oxide concentrates were recovered; the second lot was similarly treated, except that Aero Xanthate 301 (0.1 lb/ton) was added with the Aero Promoter 425 in the first stage of the oxide flotation.

Combined oxide concentrates were leached for thirty minutes at room temperature by addition of sulphuric acid to pH 1.8. Then, the combined sulphide concentrates were added to the slurry, followed by acid to reduce the pH value from the resulting 4.4 to 1.9. Copper solution was recovered by filtration. Filtrate and washings were combined for analysis. Copper was not precipitated.

The leach residue was repulped for sulphide cleaner flotation under conditions similar to those followed in Test 10. A cleaner concentrate and a cleaner tailing were obtained.

Final (oxide) flotation tailings were sampled separately for analysis.

TABLE 17

Results of Sulphide-Oxide Flotation and Concentrate Leaching

Product	Weight % or Volume	Assays [*]			Distribution %		
		Cu %	Au oz/ton	Ag oz/ton	Cu	Au	Ag
Sulphid conc ^{**}	1.1	41.22	2.52	10.0	41.5	60.8	43.0
" " tailing	5.0	2.80	0.18	1.8	12.7	19.7	35.2
Copper solution	(7.1 1)	0.17	-	-	28.2	-	-
Final tailing A	47.1	0.23	0.011	0.06	9.9	11.4	10.9
" " B	46.8	0.18	0.008	0.06	7.7	8.1	10.9
Head (calcd)	100.0	1.09	0.046	0.26	100.0	100.0	100.0

* From Internal Report MS-AC-62-1225.

** Palladium content of the sulphide cleaner concentrate was 0.84 oz/ton, representing a recovery of 57.6% calculated by difference from a head sample analysis of 0.016 oz/ton. Platinum in the concentrate was less than 0.04 oz/ton. Rhodium was not detected. Tailings were not analyzed for these precious metals.

The use of Aero Xanthate 301 along with Aero Promoter 425 in the oxide flotation appeared to be beneficial as indicated by the significantly lower copper and gold content in tailing B, equivalent to increased recoveries of 2.2% and 3.3% respectively.

DISCUSSION OF RESULTS

The fine occurrence of both the sulphide and the oxide copper minerals made gravity concentration unsatisfactory. By a combination of tabling and magnetic separation (to remove about 4% magnetite), the best recoveries of copper, gold and silver were only 31.5%, 66.0% and 30.4% respectively.

Wet, high-intensity magnetic separation (Test 4) was ineffective, producing a copper concentrate of only 1.65% grade with 25% recovery.

Only by flotation was satisfactory concentration of copper achieved. Conventional sulphide flotation produced cleaner concentrates at copper grades of 34-45%, with rougher recovery of about 54% of the total copper or 90% of the sulphide copper.

By a subsequent oxide flotation step with staged sulphidization, additional copper recovery of about 28% (70% of the oxide copper) was obtained in a rougher concentrate of 10-17% grade. Although this was upgraded to 24% in one cleaning step (Test 10), the difficulty of smelting such fine material, even when pelletized or briquetted, might severely limit its market.

Since sulphuric acid leaching is common metallurgical practice for treating oxide copper ores, this technique was investigated thoroughly. Because of the 4% calcite content of the ore sample, acid-leaching of the whole ore, or even the sulphide flotation tailing, was impractical due to acid consumption of 110-120 pounds per ton of ore (equivalent to nearly 15 pounds of 100% H_2SO_4 per pound of copper dissolved). However, by leaching the oxide flotation concentrate (as in Tests 14 and 15), the acid consumption was only about 2.6 pounds for each pound of copper extracted. Theoretical requirement is about 1.6 pounds.

Much of the bornite, in grains down to 20 micron size, is finely disseminated in the pegmatite. For maximum recovery it was necessary to grind to at least 65% minus 200 mesh. However, the resulting slimes, particularly from the calcite and oxidized copper minerals, adversely effected both flotation steps and increased the acid consumption in subsequent leaching.

Preliminary desliming (Test 12) facilitated both flotation steps and produced the highest grade concentrates, but at the expense of 10% loss of copper in the slime fraction (1.31% Cu). In Test 13 an attempt was made to produce a useful concentrate from the slimes by flotation. Although 60%

recovery (6%, original ore basis) was obtained, the copper grade was only 4.3% in a highly calciferous gangue which consumed about 14 pounds of acid per pound of copper extracted.

Best recovery came from careful size classification achieved in laboratory tests by staged grinding to minus 100 mesh as shown by the following results of similar flotation tests on ore ground under different conditions:

Test	Procedure	% -200 mesh	Rougher Flotation Recovery, %				
			Cu			Au	Ag
			Sulph	Oxide	Total		
12	Deslime, grind	65.8	46	27	73	83	68
14	Stage grind to -65m	59.1	48	31	79	80	76
15	" " " -100m	68.8	52	29	81	84	83

Sulphuric acid leaching of the oxide concentrate to pH 1.8 gave practically complete extraction of the oxide copper. This was precipitated with iron powder and recovered by two methods:

- (a) Precipitation in the leach slurry and flotation of the cement copper, as in Test 16. Precipitation was 99.6% complete; recovery by flotation in a concentrate containing 56% copper was only 79% due to limitations of the small scale test. In plant practice 90% recovery is common. Tailing from this step would be returned to the sulphide flotation circuit for recovery of residual sulphide copper and unfloated cement copper.
- (b) Separation of copper solution, precipitation and recovery of the cement copper by settling and decantation, as in Test 11. Recovery was 99% in a concentrate containing 34% copper. The lower grade was due to contamination with iron powder which was not separated. In plant practice, sponge iron or shredded de-tinned cans are used, and separated magnetically from the cement copper. The leach residue would be returned to the sulphide flotation circuit for recovery of residual copper sulphides.

Method (b) is favoured by the good settling and filtering characteristics of the leach residue and by the relatively small amount of solids involved (about 2% of the original ore feed).

In the final test, which simulated a possible plant flowsheet but without staged grinding or classification, recoveries were: copper, 82.4%; gold, 80.5%; silver, 78.2%; and palladium, 57.6%.

CONCLUSIONS

No satisfactory recovery of copper, gold and silver could be made by tabling.

Wet, high-intensity magnetic separation was ineffective.

Best results were obtained by two-stage flotation which produced a rich sulphide copper concentrate containing gold, silver and palladium, and a low-grade oxide copper concentrate. Although the latter was upgraded by a cleaner flotation step, the fineness of such an oxide concentrate would probably limit its acceptability for smelting. However, by sulphuric acid leaching of the bulk oxide concentrate and precipitation of the copper with iron, a marketable cement copper concentrate was produced. Overall recoveries achieved were: copper, 82.4%; gold, 80.5%; silver, 78.2% and palladium, 57.6%.

This investigation indicates that the best procedure for maximum recovery of saleable concentrates from the Friday Creek ore is as follows:

- (1) Grinding to 65-70% minus 200 mesh with minimum formation of slimes;
- (2) Two-stage flotation -
 - a. Sulphide (rougher and cleaner),
 - b. Oxide, by staged addition of reagents;
- (3) Leaching the oxide concentrate with sulphuric acid, separating the copper solution and precipitating the copper by cementation with iron;
- (4) Returning oxide leach residues to sulphide flotation for recovery of residual sulphides.

Two useful references describing plant flowsheets for treatment of similar ores by this procedure are:

- Deco Trefoil (Spring-Summer, 1962), p. 17-20, Copper Leaching;
Deco Trefoil (Jan.-Feb., 1963), p. 15-16, Leach - Precipitation - Flotation.

ACKNOWLEDGEMENTS

The writer acknowledges, with thanks, the contribution to this investigation by the staff of the Mineral Sciences Division, including R. C. McAdam, R. W. Buckmaster, H. Lauder, H. H. Bart, F. Piche, G. H. Faye, P. E. Moloughney, L. Lutes, C. Derry and D. Cumming for chemical analyses, Miss E. M. Kranck for spectrographic analyses and W. Petruk for mineralogical examination.