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CANADA

DEPARTMENT OF MINES AND TECHNICAL SURVEYS

OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 65-69

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FLOTATION SEPARATION OF COPPER-LEAD-ZINC ON A SAMPLE OF LYNX ORE FROM WESTERN MINES LIMITED, BUTTLE LAKE, VANCOUVER ISLAND, B. C.

CONFIDENTIAL
NOT TO BE QUOTED FROM

by

G. I. MATHIEU

MINERAL PROCESSING DIVISION

01-798682

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by

G. I. Mathieu*

- - -

SUMMARY OF RESULTS

Analysis of the Lynx ore head sample showed 2.66% Cu, 1.48% Pb, 15.4% Zn, 0.08 oz Au and 5.54 oz Ag per ton. Mineralogical examination indicated that a fair proportion of the valuable minerals presented a rather fine structure and were intimately associated with each other.

Attempts were made to recover these minerals by either bulk or selective flotation. The results obtained by both methods can be summarized as follows:

Product	Grade	Recovery
Bulk conc	4.9% Cu	98.1% Cu
	2.6% Pb	97.9% Pb
	26.6% Zn	99.1% Zn
Selective flotation:		
Cu conc	26.0% Cu	71.8% Cu
Pb conc	55.3% Pb	56.9% Pb
Zn conc	56.2% Zn	74.8% Zn

The precious metal recovery was 93.8% Au and 97.0% Ag in the bulk concentrate, but only 63.7% Au and 72.9% Ag in the three selective flotation concentrates. In plant practice, recirculation of the middling products from selective flotation should increase the recovery of both base and precious metals.

*Scientific Officer, Mineral Processing Division, Mines Branch, Department of Mines and Technical Surveys, Ottawa, Canada.

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INTRODUCTION

The sample submitted for investigation originated from the Lynx mine located at the south end of Buttle Lake on Vancouver Island. The Lynx orebody, which is being developed by Western Mines Limited, has been estimated at 800,000 tons averaging 2.3% copper, 1.2% lead, 10.9% zinc, 2.8 oz silver and 0.06 oz gold per ton. The company intends to put this property into production in the near future.

Shipment

A sample of ore weighing 78 lb was received from J. R. Williams and Sons Ltd., on March 12, 1963. A second shipment weighing 218 lb was received from Box 8000, Campbell River, B.C., on March 14th, 1963. The two samples were mixed for the test work.

Purpose of Investigation

The investigation was requested by Mr. H.M. Wright, President, Western Mines Limited, c/o Wright Engineers Limited, 802 Credit Foncier Building, 850 West Hastings Street, Vancouver 1, B.C. At his request, the first step was flotation of a bulk concentrate for delivery to Dr. Arvid Thunaes, Eldorado Mining and Refining Limited. The investigation was then directed toward the conventional method for the treatment of complex copper-lead-zinc ore. This consists of floating successively, a copper-lead concentrate and a zinc concentrate. The copper and lead minerals are then separated by differential flotation.

Sampling and Analysis

The mixed sample was crushed to -10 mesh and a head sample riffled out for chemical analysis.

TABLE 1

Chemical Analysis* of Head Sample

Gold (Au)	-	0.085	oz ton
Silver (Ag)	-	5.54	"
Copper (Cu)	-	2.66	%
Lead (Pb)	-	1.48	"
Zinc (Zn)	-	15.4	"
Iron (Fe)	-	14.1	"
Sulphur (S)	-	23.6	"
Insoluble	-	37.3	"

*From Internal Report MS-AC-63-528.

A spectrographic analysis on a portion of the head sample indicated the presence of the elements listed below in approximate order of decreasing abundance.

TABLE 2

Spectrographic Analysis*of Head Sample

I	- Fe, Zn, Si, Cu, Pb.
II	- Al, Ba, Na, Ca, Cd, Mg.
III	- Ti, Mn, Sr, Ag.
IV	- Ni, V, Mo, Ga, Cr.

*From Internal Report MS-AC-63-528.

Mineralogy of the Ore*

An extensive mineralographic study made by Dr. R.M. Thomson indicated that the Lynx ore consists mainly of pyrite, sphalerite, chalcopyrite, galena, bornite, tennantite, covellite and digemite. The close association of these minerals suggested that a minimum grinding to 80% -200 mesh would be necessary to obtain a satisfactory liberation of the copper, lead and zinc minerals.

OUTLINE OF INVESTIGATION PROCEDURE

The investigation started with a bulk flotation of the copper, lead and zinc minerals.

In the second part of the investigation, the various stages of selective flotation were investigated, in turn, under a series of conditions. These were:

1 - Rougher flotation of a copper-lead concentrate while varying successively grinding, concentration of zinc depressants, alkalinity and aeration conditions.

2 - Cleaning copper-lead concentrate with and without regrinding and alkalis.

3 - Separation of the copper and lead minerals by various techniques of differential flotation.

4 - Rougher flotation of a zinc concentrate using different frothers.

5 - Cleaning zinc concentrate with and without regrinding.

*From a Mineralographic Study, Western Mines Limited, Myra Falls, B.C. By R.M. Thompson, April 1963.

The last part of the investigation consisted of testing an integrated selective flotation flowsheet combining the best practices found previously.

DETAILS OF INVESTIGATION

Part I, Bulk Flotation

Test 1

A 2,000 g sample of ore was ground for 20 min to 77% -200 m and floated using the procedure summarized in Table 3.

TABLE 3

Reagents and Conditions of Bulk Flotation

Operation	Time min	Reagents	lb/ton	pH
Grinding	20	Lime	1.00	
		Xanthate Z-6	0.04	
		Dowfroth 250	0.04	
Bulk flotation	20	Copper sulphate	1.00	9.1
		Xanthate Z-6	0.02	
		Dowfroth 250	0.02	
		After 10 min:		
		Copper sulphate	0.30	8.5
		Xanthate Z-6	0.02	
		Dowfroth 250	0.02	

Bulk flotation gave the following results:

TABLE 4

Results of Bulk Flotation

Product	Weight %	Analysis* % Cu	Pb	Zn	oz/ton Au	Ag	Distribution % Cu	Pb	Zn	Au	Ag
Bulk conc	52.3	4.86	2.64	26.6	0.11	10.15	98.1	97.9	99.1	93.8	97.0
Flot tailing	47.7	0.10	0.06	0.26	0.008	0.34	1.9	2.1	0.9	6.2	3.0
Feed (calcd)	100.0	2.59	1.41	14.0	0.06	5.46	100.0	100.0	100.0	100.0	100.0

*From Internal Report MS-AC-63-1293.

Part II, Selective Flotation

The various stages of selective flotation were investigated separately. This method gave more comparable results, in particular, for the cleaning and separation stages which were carried out on portions from the same feed. It also permitted a reduction in the number of analyses required.

1 - Copper-Lead Rougher Flotation

This series of tests aimed at a high recovery of copper and lead minerals with the zinc content being kept at a minimum.

Tests 2, 3 and 4, Effect of Grinding

Lots of 1,000 g of ore were ground for 20, 30, and 40 min to 77%, 88% and 94% -200 mesh, respectively, and floated under the conditions described in Table 5.

TABLE 5

Reagents and Conditions of Copper-Lead Rougher Flotation

Operation	Time min	Reagents	lb/ton	pH
Grinding		Lime	0.70	8.7
		Zinc sulphate	1.00	
		Sodium cyanide	0.15	
		Xanthate Z-6	0.03	
		Reagent PPG-425*	0.01	
Cu-Pb flotation	8	Xanthate Z-6	0.01	8.6

*Niax Diol Polypropylene Glycol 425.

The results obtained with various grinding periods are shown below.

TABLE 6

Effect of Grinding

Tests	Product	Weight %	Analysis* %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
2 (77% -200 m)	Cu-Pb conc	18.6	11.2	6.8	19.2	86.7	86.5	24.4
	Flot tailing	81.4	0.39	0.24	13.5	13.3	13.5	75.6
	Feed (calcd)	100.0	2.40	1.46	14.7	100.0	100.0	100.0
3 (88% -200 m)	Cu-Pb conc	19.1	11.4	6.7	19.3	89.6	86.8	26.3
	Flot tailing	80.9	0.31	0.24	12.8	10.4	13.2	73.7
	Feed (calcd)	100.0	2.43	1.47	14.1	100.0	100.0	100.0
4 (94% -200 m)	Cu-Pb conc	17.0	13.3	7.1	18.5	91.3	85.3	21.9
	Flot tailing	83.0	0.26	0.25	13.5	8.7	14.7	78.1
	Feed (calcd)	100.0	2.48	1.42	14.3	100.0	100.0	100.0

*From Internal Report MS-AC-64-261.

Tests 5, 6 and 7, Effect of Sodium Cyanide

This series of tests was done on ore ground to 88% -200 mesh. Flotation procedure was as shown in Table 5, except that sodium cyanide was increased to 0.25, 0.50 and 0.80 lb/ton of ore in attempts to depress more zinc without affecting the copper recovery.

TABLE 7

Effect of Sodium Cyanide

Tests	Product	Weight %	Analysis* %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
5 (NaCN-0.25)	Cu-Pb conc	19.9	12.2	7.0	18.3	90.7	88.9	27.0
	Flot tailing	80.1	0.31	0.20	12.3	9.3	11.1	73.0
	Feed (calcd)	100.0	2.68	1.43	13.5	100.0	100.0	100.0
6 (NaCN-0.50)	Cu-Pb conc	19.1	11.9	7.2	17.3	85.9	88.5	24.8
	Flot tailing	80.9	0.46	0.20	12.4	14.1	11.5	75.2
	Feed (calcd)	100.0	2.65	1.41	13.3	100.0	100.0	100.0
7 (NaCN-0.80)	Cu-Pb conc	17.2	11.4	8.1	16.8	72.9	88.9	21.0
	Flot tailing	82.8	0.88	0.21	13.1	27.1	11.1	79.0
	Feed (calcd)	100.0	2.69	1.57	13.7	100.0	100.0	100.0

*From Internal Report MS-AC-63-1178.

Additional zinc sulphate was also tried to depress more sphalerite, but no noticeable effect on the amount of zinc floating with the copper-lead concentrate was observed.

Tests 8, 9 and 10, Effect of Alkalis

These tests followed the basic procedure illustrated in Table 5, except that Test 8 was done without alkali (pH - 7.0), Test 9 was made with 1.0 lb lime/ton (pH - 8.7) and Test 10 was carried out with 1.0 lb soda ash/ton (pH - 7.6).

TABLE 8

Effect of Alkalis

Tests	Product	Weight %	Analysis* %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
8 (No alkali)	Cu-Pb conc	18.3	12.6	7.1	18.3	89.8	89.8	24.6
	Flot tailing	81.7	0.32	0.18	12.6	10.2	10.2	75.4
	Feed (calcd)	100.0	2.57	1.45	13.6	100.0	100.0	100.0
9 (Lime)	Cu-Pb conc	19.7	12.1	7.0	19.5	88.7	89.5	26.7
	Flot tailing	80.3	0.38	0.20	13.1	11.3	10.5	73.3
	Feed (calcd)	100.0	2.69	1.54	14.4	100.0	100.0	100.0
10 (Soda Ash)	Cu-Pb conc	21.1	11.3	6.8	19.2	93.5	91.9	30.3
	Flot tailing	78.9	0.21	0.16	11.8	6.5	8.1	69.7
	Feed (calcd)	100.0	2.55	1.56	13.4	100.0	100.0	100.0

*From Internal Report MS-AC-1434 and 1470.

Tests 11, 12 and 13, Effect of Aeration

These tests were similar to Tests 8, 9 and 10 respectively, except that the ground material was aerated for 20 min prior to flotation.

TABLE 9

Effect of Aeration

Tests	Product	Weight %	Analysis* %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
11 (No alkali)	Cu-Pb conc	16.9	13.8	7.3	18.0	90.4	80.9	21.4
	Flot tailing	83.1	0.30	0.35	13.4	9.6	19.1	78.6
	Feed (calcd)	100.0	2.58	1.52	14.2	100.0	100.0	100.0
12 (Lime)	Cu-Pb conc	18.2	12.0	6.8	19.9	88.7	84.8	25.3
	Flot tailing	81.8	0.34	0.27	13.1	11.3	15.2	74.7
	Feed (calcd)	100.0	2.46	1.46	14.3	100.0	100.0	100.0
13 (Soda Ash)	Cu-Pb conc	17.0	13.3	7.2	20.5	90.1	82.6	24.1
	Flot tailing	83.0	0.30	0.31	13.2	9.9	17.4	75.9
	Feed (calcd)	100.0	2.51	1.48	14.4	100.0	100.0	100.0

*From Internal Report MS-AC-63-1463.

Aeration gave a slight increase in the copper grade, but resulted in lower lead recovery.

2 - Copper-Lead Cleaner Flotation

To provide adequate rougher concentrate for investigation of cleaner flotation techniques, 16,000 g of ore were ground to 87% -200 m and floated in 2,000 g batches by the procedure shown in Table 5 with the omission of lime. Concentrates were combined and split into 500 g fractions.

Tests 14 and 15, Effect of Regrinding

One fraction was reground for 10 min to 95% -325 m; the other, at about 90% -200 m, was treated without further grinding. The procedure for cleaning and the results obtained in these tests are summarized in Tables 10 and 11.

TABLE 10

Reagents and Conditions of Copper-Lead Cleaner Flotation

Operation	Time min	Reagents	lb/ton of ore	pH
Cleaner flotation	5	Zinc sulphate	0.15	6.7
		Sodium cyanide	0.02	
		Xanthate Z-6	0.02	
Recleaner flotation	4	Zinc sulphate	0.10	6.8
		Sodium cyanide	0.01	

TABLE 11

Effect of Regrinding

Tests	Product	Weight %	Analysis* %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
14 (90% -200 m)	Cu-Pb recl conc	57.6	21.4	11.5	10.4	95.1	92.0	36.2
	Cu-Pb recl tailing	13.2	2.1	2.4	26.1	2.2	4.4	20.8
	Cu-Pb cl tailing	29.2	1.2	0.9	24.4	2.7	3.6	43.0
	Feed (calcd)	100.0	13.0	7.2	16.6	100.0	100.0	100.0
15 (95% -325 m)	Cu-Pb recl conc	56.2	21.7	11.2	10.3	92.8	86.3	33.9
	Cu-Pb recl tailing	12.7	4.0	4.7	31.5	3.9	8.2	23.4
	Cu-Pb cl tailing	31.1	1.4	1.3	23.5	3.3	5.5	42.7
	Feed (calcd)	100.0	13.0	7.3	17.1	100.0	100.0	100.0

*From Internal Reports MS-AC-64-725 and 2023.

Tests 16 and 17, Effect of Alkalis

These tests were similar to Test 14, except that lime in ratios of 0.08 and 0.03 lb/ton (Test 16) and soda ash in amounts of 0.30 and 0.10 lb/ton (Test 17) were added to the cleaner and recleaner stages respectively. With the lime, the pH readings were 9.8 and 9.7. In the test with soda ash, these were 9.2 and 9.1.

TABLE 12

Effect of Alkalis

Tests	Product	Weight %	Analysis* %			Distribution %		
			Cu	Pb	Zn	Cu	Pb	Zn
16 (Lime)	Cu-Pb recl conc	62.0	20.1	10.5	11.1	94.0	90.3	39.9
	Cu-Pb recl tailing	8.3	4.1	3.7	25.9	2.6	4.3	12.5
	Cu-Pb cl tailing	29.7	1.5	1.3	27.6	3.4	5.4	47.6
	Feed (calcd)	100.0	13.2	7.2	17.2	100.0	100.0	100.0
17 (Soda ash)	Cu-Pb recl conc	60.7	20.2	10.9	11.9	93.7	89.2	42.6
	Cu-Pb recl tailing	8.8	3.9	3.6	24.8	2.6	4.3	12.9
	Cu-Pb cl tailing	30.5	1.6	1.6	24.7	3.7	6.5	44.5
	Feed (calcd)	100.0	13.1	7.4	16.9	100.0	100.0	100.0

*From Internal Report MS-AC-64-2132.

In an attempt to improve the selectivity in the cleaning stages, a series of tests was carried out along the following lines:

1 - The copper-lead concentrate was agitated with different quantities of sodium sulphide with the purpose of destroying the collecting film and possibly precipitating disturbing ions in the pulp.

2 - The slurry was filtered and thoroughly washed to remove the residual collector.

3 - The filter cake was repulped and then cleaned by flotation using starvation amounts of frother (PPG-425) and collector (Z-6) with liberal addition of zinc sulphate and sodium cyanide.

Unfortunately, this technique failed to reduce the zinc content in the final copper-lead concentrate. This may have been due to reactivation of some free sphalerite as well as sphalerite intimately associated with copper and lead.

3 - Copper-Lead Separation

Two different methods were investigated to separate the copper and lead minerals. The first method consisted of floating the copper while depressing the lead. In the second method, the galena was differentially floated. In both cases, it was found necessary to use a multi-stage separation and several cleaning steps to produce final products of acceptable grade and recovery. Several tests were made with each technique, but only the best of these will be reported. The tests were carried out on fractions from a mixed sample composed of recleaner copper-lead concentrates produced in Tests 14 to 18.

Test 19, Dichromate Method

The procedure and the results of this test are shown in Tables 13 and 14.

TABLE 13

Reagents and Conditions of Copper-Lead Separation with Dichromate

Operation	Time min	Reagents*	lb/ton of ore	pH
Cu separation (twice)	5	Sulphuric acid Potassium dichromate Reagent Z-200	0.5 1.0 0.01	5.2
Cu cleaner flotation (twice)	4	Sulphuric acid Potassium dichromate Reagent Z-200	0.3 0.5 0.005	5.4
Cu recleaner flotation (twice)	3	Sulphuric acid Potassium dichromate	0.2 0.2	5.3

*These reagent additions were made to each of the two separation and cleaning stages.

TABLE 14

Results of Separation with Dichromate

Product	Weight %	Analysis* %			Distribution %		
		Cu	Pb	Zn	Cu	Pb	Zn
Cu recl conc	62.1	28.7	3.9	7.7	83.4	21.1	44.4
Pb conc	24.5	5.9	28.6	17.6	6.8	61.1	40.0
Cu recl tailing	10.4	17.1	13.9	12.9	8.3	12.7	12.4
Cu cl tailing	3.0	10.9	19.6	11.4	1.5	5.1	3.2
Feed (calcd)	100.0	21.4	11.5	10.8	100.0	100.0	100.0

*From Internal Report MS-AC-65-1.

Alternate methods were also investigated to produce a differential flotation of the copper minerals. The main features of these were conditioning either with sulphur dioxide and starch or with a large amount of lime prior to flotation. However, none of these techniques improved the separation.

Test 20, Cyanide Method

The procedure and the results of this test are summarized in Tables 15 and 16.

TABLE 15

Reagents and Conditions of Copper-Lead Separation with Cyanide

Operation	Time min	Reagents*	lb/ton of ore	pH
Pb separation (Four times)	4	Sodium cyanide Reagent PPG-425	1.00 0.001	11.5
Pb cleaner flotation (Twice)	3	Sodium cyanide Reagent PPG-425	0.50 0.001	11.4
Pb recleaner flotation (Once)	2	Sodium cyanide	0.20	11.2

*Reagent additions to each stage.

TABLE 16

Results of Separation with Cyanide

Product	Weight %	Analysis* %			Distribution %		
		Cu	Pb	Zn	Cu	Pb	Zn
Cu conc	72.1	25.3	1.3	9.7	84.6	8.2	69.2
Pb recl conc	16.7	4.9	57.1	10.8	3.8	83.3	17.9
Pb recl tailing	5.2	20.7	12.0	12.8	5.0	5.5	6.6
Pb cl tailing	6.0	23.7	5.8	10.6	6.6	3.0	6.3
Feed (calcd)	100.0	21.6	11.4	10.1	100.0	100.0	100.0

*From Internal Report MS-AC-65-1.

4 - Zinc Rougher Flotation

Tailings from previous copper-lead flotation tests were combined to make a feed for investigating rougher flotation of sphalerite. The only difficulty encountered when floating this mineral was the brittleness and instability of the froth. Different types of frothers, such as Polypropylene Glycol 425, Dowfroth 250 and Butanol High Boiler (BHB), were tried successively in various amounts to improve the quality of the froth and possibly the recovery of zinc.

Tests 21, 22 and 23, Effect of Frothers

The flotation procedure and the best results with each frothing agent tried are shown in Tables 17 and 18.

TABLE 17

Reagents and Conditions of Zinc Rougher Flotation

Operation	Time min	Reagents	lb/ton	pH
Conditioning	5	Lime	0.50	8.9
		Copper sulphate	1.00	
		Aerofloat 211	0.03	
		Sodium Aerofloat	0.03	
Rougher flotation	15	Test 21: PPG-425	0.02	
		Test 22: Dowfroth 250	0.04	
		Test 23: BHB	0.08	

TABLE 18

Effect of Frothers

Test	Product	Weight %	Analysis* % Zn	Distribution % Zn
21 (PPG-425)	Zn conc Flot tailing	24.6 75.4	49.2 0.56	96.6 3.4
	Feed (calcd)	100.0	12.5	100.0
22 (DOW-250)	Zn conc Flot tailing	24.8 75.2	47.7 0.63	96.2 3.8
	Feed (calcd)	100.0	12.3	100.0
23 (BHB)	Zn conc Flot tailing	27.5 72.5	46.0 0.32	98.2 1.8
	Feed (calcd)	100.0	12.9	100.0

*From Internal Report MS-AC-63-1339.

As well as increasing recovery, Butanol High Boiler gave the most stable froth.

5 - Zinc Cleaner Flotation

A few tests were carried out to determine if regrinding prior to zinc cleaner flotation would improve the results of this operation. This proved of little value as shown by the two following representative tests.

Tests 24 and 25, Effect of Regrinding

One half of the concentrate produced in Test 23 was reground to 94% -325 m while the other half was treated without regrinding. Procedure for cleaning and results obtained are shown in Tables 19 and 20.

TABLE 19

Reagents and Conditions of Zinc Cleaner Flotation

Operation	Time min	Reagents*	lb/ton	pH
Cleaner flotation (twice)	8	Lime Copper sulphate Aerofloat 211 Sodium Aerofloat	0.10 0.05 0.005 0.005	10.6

*Reagent additions to each stage.

TABLE 20

Effect of Regrinding

Test	Product	Weight %	Analysis* % Zn	Distribution % Zn
24 (85% -200 m)	Zn cl conc	72.9	59.6	96.2
	Zn cl tailing	27.1	6.3	3.8
	Feed (calcd)	100.0	45.2	100.0
25 (93% -325 m)	Zn cl conc	70.9	61.6	93.2
	Zn cl tailing	29.1	10.9	6.8
	Feed (calcd)	100.0	46.8	100.0

*From Internal Report MS-AC-64-15.

Part III, Integrated Selective Flotation

The purpose of this part of the investigation was to incorporate the features of the various stages of selective flotation in order to simulate a possible flowsheet for the treatment of the Lynx ore.

Test 26, Flowsheet I

The procedure of this test is illustrated by Figure 1 and summarized in Table 21.

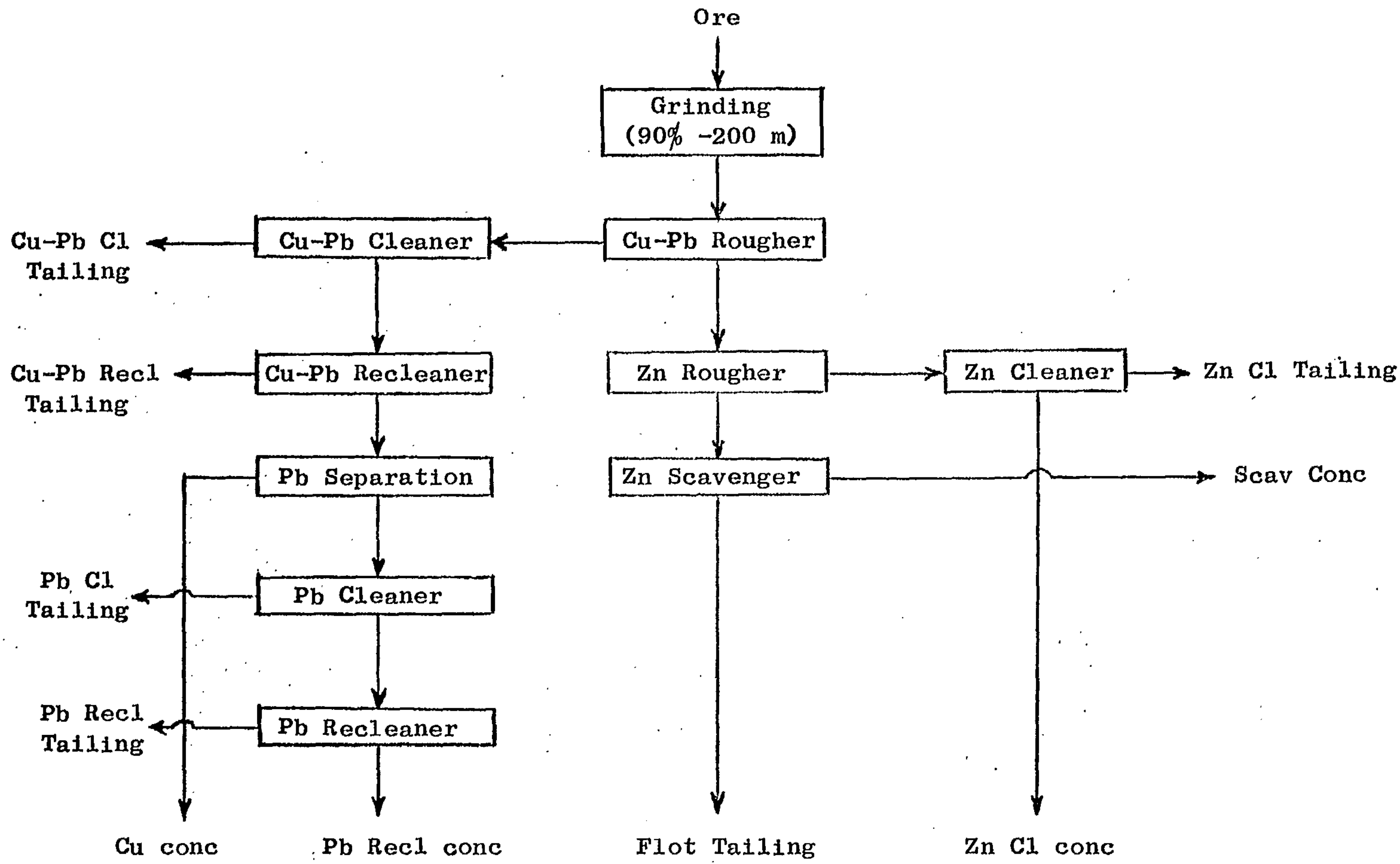

Figure 1 - Flowsheet I

TABLE 21

Reagents and Conditions for Flowsheet I

Operation	Time min	Reagents*	lb/ton	pH
Grinding	30	Zinc sulphate	1.00	
		Sodium cyanide	0.15	
		Xanthate Z-6	0.04	
		Reagent PPG-425	0.01	
Cu-Pb flotation:				
Rougher	8	Xanthate Z-6	0.01	7.1
		Reagent PPG-425	0.005	
Cleaner	5	Zinc sulphate	0.15	7.3
		Sodium cyanide	0.02	
		Xanthate Z-6	0.02	
		Reagent PPG-425	0.001	
Recleaner	4	Zinc sulphate	0.10	7.3
		Sodium cyanide	0.01	
		Xanthate Z-6	0.01	
		Reagent PPG-425	0.001	
Cu-Pb separation:				
Pb rougher	4	Sodium cyanide	1.00	10.9
(Four times)		Reagent PPG-425	0.001	
Pb cleaner	3	Sodium cyanide	0.50	10.7
(Twice)		Reagent PPG-425	0.001	
Pb recleaner	2	Sodium cyanide	0.20	10.6
Zn flotation:				
Conditioning	5	Lime	0.60	9.1
		Copper sulphate	1.00	
		Aerofloat 211	0.03	
		Sodium Aerofloat	0.03	
Rougher	15	Reagent BHB	0.08	
Cleaner	8	Lime	0.10	10.4
(Twice)		Copper sulphate	0.02	
		Aerofloat 211	0.01	
		Sodium Aerofloat	0.01	

*Reagent additions to each stage.

TABLE 22

Test Results for Flowsheet I

Part 1: Weight and Analysis

Product	Weight %	Analysis % Cu	Pb	Zn	oz/ton Au	Ag
Cu conc	7.1	26.0	1.6	9.9	0.40	37.4
Pb recl conc	1.5	5.9	55.3	10.3	0.02	20.2
Zn cl conc	17.6	1.0	1.0	56.2	0.06	5.4
Pb recl tailing	0.5	20.1	15.9	11.3	0.04	17.6
Pb cl tailing	0.5	24.9	8.5	8.5	0.05	18.9
Cu-Pb recl tailing	1.8	3.1	2.9	24.5	0.24	19.5
Cu-Pb cl tailing	4.8	1.5	1.1	23.4	0.09	8.6
Zn cl tailing	11.8	0.5	0.5	4.5	0.03	2.2
Flot tailing	54.4	0.09	0.10	0.50	0.018	0.46
Feed (calcd)	100.0	2.57	1.46	13.21	0.061	5.36

Part 2: Distribution

Product	Distribution % Cu	Pb	Zn	Au	Ag
Cu conc	71.8	7.8	5.4	46.1	49.4
Pb recl conc	3.4	56.9	1.2	0.5	5.7
Zn cl conc	6.9	12.1	74.8	17.1	17.7
Pb recl tailing	3.9	5.4	0.4	0.3	1.6
Pb cl tailing	4.8	2.9	0.3	0.4	1.8
Cu-Pb recl tailing	2.2	3.6	3.3	7.0	6.5
Cu-Pb cl tailing	2.8	3.6	8.5	7.0	7.7
Zn cl tailing	2.3	4.0	4.0	5.7	4.9
Flot tailing	1.9	3.7	2.1	15.9	4.7
Feed (calcd)	100.0	100.0	100.0	100.0	100.0

In this test, the soluble losses were 7.7% of the gold and 3.0% of the silver. To reduce these losses, a sodium cyanide-zinc oxide complex was substituted for sodium cyanide and zinc sulphate. By this technique, the soluble losses were reduced by about 65%, but the copper-lead separation was seriously impaired. Lack of ore prevented further testing on this technique.

SUMMARY AND CONCLUSIONS

The ore received from the Lynx property of Western Mines Limited assayed:

Cu	Pb	Zn	Au	Ag
2.66%	1.48%	15.4%	0.08 oz/ton	5.54 oz/ton

The metallic minerals consisted essentially of pyrite, sphalerite, chalcopyrite, galena, bornite and covellite. The rather close association of these minerals suggested that fine grinding would be necessary to achieve satisfactory liberation.

A bulk flotation test was carried out at the suggestion of Mr. H.M. Wright. The grade and recovery of the concentrate were as follows:

	Cu	Pb	Zn	Au	Ag
Grade	4.9%	2.6%	26.6%	0.11 oz/ton	10.1 oz/ton
Recovery	98.1%	97.9%	99.1%	93.8%	97.0%

The investigation was then directed to a selective flotation technique consisting of copper-lead flotation and separation, followed by zinc flotation. Investigation of this method step by step established the following points:

1 - Selectivity of the copper-lead rougher flotation was slightly increased by finer grinding, but was not affected significantly either by more liberal addition of zinc depressants or by the presence of alkalis with or without aeration.

2 - Reduction of the high zinc content in the copper-lead concentrate was achieved by cleaning stages, but regrinding, addition of alkalis, and treatment with sodium sulphide failed to improve the results of the cleaner flotation.

3 - Best separation of the copper and lead minerals was obtained by depressing the copper with sodium cyanide. Other separation methods using lime, sulphuric acid and potassium dichromate, sulphur dioxide and starch were found to be less effective.

4 - Flotation of sphalerite gave optimum results when using BHB frother, Aerofloat 211 and Sodium Aerofloat. Regrinding of the zinc rougher concentrate prior to the cleaning stages was not found beneficial.

The features from each step of the selective flotation were combined in a test simulating an integrated flowsheet. The results obtained are summarized in the following table:

Product	Cu (%)	Pb (%)	Zn (%)	Au (oz/ton)	Ag (oz/ton)
Cu conc	26.0	1.6	9.9	0.40	37.4
Pb conc	5.9	55.3	10.3	0.02	20.2
Zn conc	1.0	1.0	56.2	0.06	5.4
Recovery (%)	71.8	56.9	74.8	63.7	72.8

Recirculation of the middling products in plant practice should result in appreciable increase in these recoveries.

In the above test, the use of large quantity of sodium cyanide resulted in soluble losses of about 8% of the gold and 3% of the silver. Limited investigation on this problem indicated that the use of a sodium cyanide-zinc oxide complex reduced sharply these losses, but might seriously impair the copper-lead separation.

This investigation did not completely solve the major problem, in the treatment of the Lynx ore, namely, the unduly high zinc content in both copper and lead concentrates. This is probably due to fine intergrowth of spahlerite with other metallic minerals as well as to the presence of secondary copper minerals. The latter contributed copper ions with an activating effect on sphalerite during copper-lead flotation.

The heavy zinc loss attributed to the peculiar characteristics of the Lynx ore might be solved by using a roast-leach technique. Although beyond the objective of this investigation, a preliminary test was carried out in which a copper-lead concentrate was roasted for 3 hours and leached in water containing 50 g of sulphuric acid per liter. About 90% of the zinc was extracted by this technique. Conventional electro-winning would complement the procedure. The residue which contains the lead and the precious metals might be either marketed or treated by conventional methods. In addition to recovering additional zinc, the roast-leach technique would reduce the soluble losses of precious metals, eliminate the multi-stage separation and prevent copper loss in lead concentrate and vice-versa.

For more details about these methods of extraction, the following references are suggested.

1 - G.A. Smithson and J.E. Hanway, "Bench-Scale Development of a Sulfation Process for Complex Ores" and "Pilot Plant Development of a Sulfation Process for Complex Sulphide Ores", A.I.M.E. Transactions, Volume 224 (1962).

2 - H.L. Hazen, R. Ellerman and A.E. Lang, "Why not a Chemical Smelter?", Deco Trefoil, April 1964, and "Cost Studies Build Good Case for a Chemical Smelter", Metal Mining and Processing, April 1964.

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GIM/Jg