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DEPARTMENT OF ENERGY, MINES AND RESOURCES

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MINES BRANCH INVESTIGATION REPORT IR 72-5

BENEFICIATION OF A BASE METAL ORE FROM CHAPUT MINE, LUMBY, B. C.

by

D. RAICEVIC AND R. W. BRUCE

MINERAL PROCESSING DIVISION

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D. Raicevic* and R.W. Bruce**

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SUMMARY OF RESULTS

The sample of ore received for this investigation contained:

Ag oz./ton	Cu %	Pb %	Zn %
20.96	0.22	1.53	1.53

All of the silver and most of the copper were in the form of argentiferous tetrahedrite and only a small amount of the copper was present as chalcopyrite.

Most of the lead was present as galena containing inclusions of tetrahedrite, sphalerite and chalcopyrite.

The major portion of the zinc was as sphalerite but a small amount of the zinc was present as tetrahedrite.

Two procedures were developed for obtaining separate copper-silver, lead, and zinc concentrates; one consisted of flotation and tabling as major steps, while jigging and flotation were the major steps in the second procedure.

The results of the separate concentrates obtained from each procedure were as follows:

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	Flotation-Tabling			Jigging-Flotation		
	Ag oz/ton	Cu %	Pb %	Ag oz/ton	Cu %	Pb %
Copper-silver conc.						
Assay	1677.0	16.92	16.47	1847.5	19.18	14.09
Distribution	67.6	60.4	8.2	66.6	61.2	6.3
Lead concentrates						
Assay	230	1.8	62	220.8	1.84	61.60
Distribution	24.1	22.3	75.2	22.8	16.7	78.4

Zinc concentrates

The zinc concentrates obtained from each procedure assayed about 55% Zn with a 75% zinc recovery.

Although it appeared that the jigging-flotation procedure could have some economic advantage such as a smaller flotation plant and a lower operating cost over the flotation-tabling procedure, a proper evaluation of these two procedures could be done only on a pilot-plant scale.

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INTRODUCTION

Location of Property

The Chaput Mine property is located near the town of Lumby in Osoyoos District, about 20 miles east of the north end of Okanagan Lake in the southeastern part of British Columbia. The property was operated by the F.K. Exploration Ltd. of Vancouver but recently was acquired by Alberta Gypsum Ltd., of Calgary, Alberta.

Ore Shipment, Sampling and Analysis

A 240-lb sample of about minus one-inch ore was received on May 12, 1971. After picking about two dozen fragments of sulphides and non-sulphides for the mineralogical examination, the ore was mixed and riffled into two halves. One half was stored in the uncrushed (original) form to minimize oxidation and the other half was crushed to minus 10 mesh and rifled into approximately 2000-gram representative samples. One of these samples was chosen at random as a head sample for the chemical analysis reported in Table 1.

TABLE 1

Chemical Analysis of Head Sample

oz/ton		per cent			
Au	Ag	Cu	Pb	Zn	Insol
0.0075	20.96	0.22	1.53	1.53	64.20

Purpose of Investigation

The objective of this laboratory investigation was to develop a procedure that will produce a separate copper-silver, a separate lead, and a separate zinc concentrate with maximum recoveries of metals in the corresponding concentrates. Each concentrate should be suitable for smelting, i.e., the copper concentrate should contain at least 15% Cu and the lead and zinc concentrates should contain over 50% lead and 50% zinc respectively.

MINERALOGICAL EXAMINATION OF THE ORE*

Detailed mineralogical examination of the ore and the beneficiation products of this investigation were done by Dr. D. C. Harris and Mr. D. R. Owens, of Mineral Sciences Division, Department of Energy, Mines and Resources. Only a pertinent summary of their investigations regarding the metallic minerals will be included in this report.

The mineralogical studies showed that this ore consists of siliceous and carbonaceous gangue in which are disseminated small masses of metallic minerals.

Copper-Silver Minerals

Practically all of the silver and most of the copper in the ore are present in the form of coarse-grained argentiferous tetrahedrite. Its average chemical formula, determined by electron microprobe analysis, was as follows: $(Cu, Ag, Fe, Zn)_3 (Sb, As) S_3$

The average chemical composition of this tetrahedrite determined by the same analysis is given in Table 2.

TABLE 2

Results of Microprobe Analysis of Tentrahedrite, Chaput Ore

Per Cent

Cu	Ag	Fe	Zn	Sb	As	S
30.56	10.80	3.84	3.42	23.64	3.45	23.84

The results in Table 2 represent constituents chemically bonded in the crystal structure of tetrahedrite and not those as inclusions of the other minerals in tetrahedrite.

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*From Mines Branch Investigation Report IR-71-52 and IR 71-57.

These results showed that the copper, silver and antimony contents in the crystal of tetrahedrite were fairly high but that the arsenic content was low. The analysis also showed that this tetrahedrite contained some zinc and iron in its crystal structure but no lead. A fair amount of tetrahedrite occurs as inclusions in galena (PbS) as shown in Figure 1 and a small amount as inclusions in chalcopyrite (CuFeS) as shown in Figure 2.

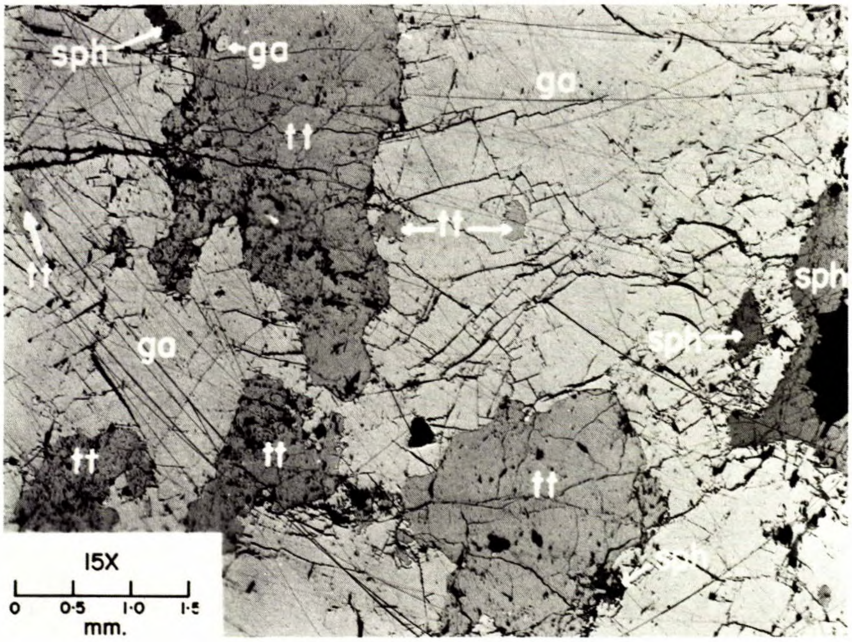

Figure 1. Photomicrograph of a polished section showing part of one of the hand specimens of galena (greyish white). The galena contains quite large inclusions of argentiferous tetrahedrite (medium grey) and a few smaller ones of sphalerite (dark grey). The galena is heavily fractured, and many of the fractures are filled with anglesite (black).

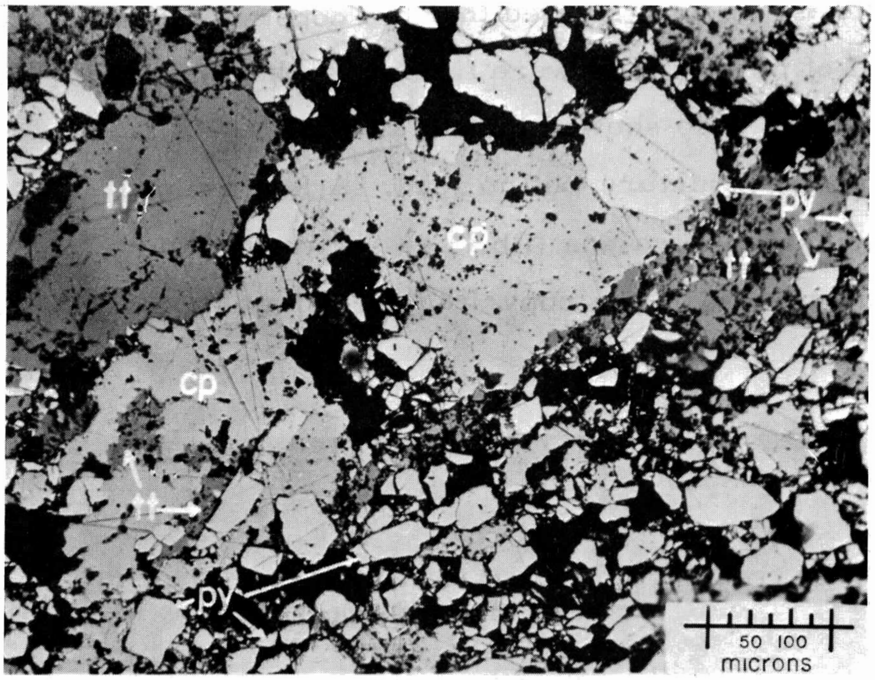

Figure 2. Photomicrograph (in oil immersion) of a polished section showing associated chalcopyrite (cp), argentiferous tetrahedrite (tt) and pyrite (white). The black areas are mainly polishing pits.

Only few inclusions of tetrahedrite were observed in sphalerite (Zn S).

The inclusions of the other minerals in tetrahedrite consist of galena, sphalerite pyrite (FeS_2), chalcopyrite and gangue minerals.

Only a small amount of copper in the ore is presant as chalcopyrite and only traces were present as covellite (Cu S).

Lead Minerals

The lead in this ore is mainly present as coarse-grained galena but a small amount is present in the form of bournonite (Pb Cu SbS_3) and anglesite (Pb SO_4). Small amounts of the galena also occur as the inclusions in tetrahedrite and sphalerite and, to a lesser degree, in pyrite and gangue minerals. The minerals forming inclusions in galena are tetrahedrite, sphalerite, anglesite, pyrite, arsenopyrite (Fe As S), chalcopyrite and gangue minerals. The inclusions of the first three minerals mentioned are shown in Figure 1.

Zinc Minerals

The only significant zinc-bearing mineral in the ore is sphalerite (Zn S). Although tetrahedrite contains some zinc as shown in Table 2, the amount of zinc contained in tetrahedrite is small. Only a small portion of sphalerite contains inclusions of galena, pyrite, arsenopyrite, tetrahedrite, chalcopyrite and gangue minerals. Some sphalerite also occurs as inclusions in tetrahedrite and in pyrite.

The electron microprobe showed that this sphalerite, like many others, is a cadmium-bearing sphalerite and contains 0.8% cadmium.

Gangue Minerals

The gangue minerals present in this ore are quartz, calcite, mica, chlorite,feldspar,and epidote.

Other Minerals

Besides the minerals mentioned, this ore contained small amounts of goethite ($Fe_2 O_3.H_2O$), rutile (TiO_2) and native gold.

OUTLINE OF INVESTIGATION

The presence of sphalerite together with other sulphide minerals always presents a problem in selective flotation. The zinc contained in the other sulphide concentrates is usually not paid for by a smelter and is therefore lost.

The first part of this investigation(Preliminary Testwork) was concerned with minimizing the zinc losses to the copper-lead rougher concentrate by using conventional zinc depressants.

After this was established,a procedure consisting of flotation and tabling was developed for obtaining a separate copper-silver, a separate lead,and a separate zinc concentrate with a minimum of zinc in the first two concentrates. The separation of the copper and lead minerals from the copper-silver-lead rougher concentrate was done by the dichromate method.

Flotation of the sphalerite was done in the conventional way by using lime, copper sulphate and frother.

As an alternative to the flotation and tabling procedure, a procedure consisting of jigging the ore prior to flotation followed by flotation of the ground pre-concentrate was also investigated. Although the jigging would add an additional cost which is usually low, the jigging-flotation procedure would mean (a) lower grinding cost because only the pre-concentrate would require fine grinding instead of the entire ore as would be the case with the flotation-tabling procedure and (b) a smaller flotation plant and, thus, lower capital and operating costs.

The preliminary test work and the two procedures developed for producing the three separate concentrates will be described separately.

DETAILS OF INVESTIGATION

Preliminary Test Work

In processing this type of ore some zinc will be lost to the copper and lead concentrates. To minimize this loss, zinc sulphate and sodium cyanide were used as sphalerite depressants during the rougher flotation of the copper and lead minerals. The copper-lead rougher concentrate was floated by the additions of lime, sodium isopropyl xanthate as collector, and Dowfroth 250 as a frother. To achieve more selectivity, both the collector and the frother were added in two stages.

Three flotation tests were done to investigate the relationship between the zinc loss to the copper-lead rougher concentrate and the amounts of the depressants used. For comparison purposes, no zinc depressants were added to the first test of this series. In the second test, the zinc sulphate alone was used as the zinc depressant, and in the third test both zinc depressants were added.

Detailed flotation conditions of this rougher float are recorded in Table 3 and the results of this series are given in Table 4.

TABLE 3

Conditions for Copper-Lead Rougher Flotation

Operation	Reagents - lb/ton flot. feed						
	CaO	$ZnSO_4$*	NaCN*	Xanth.	Dow fr.250	Time Min.	pH
Cond. No. 1	0.3	2.0	0.05	0.005	0.005	5	8.9-9.0
Cu-Pb ro flot. (1st-stage)	-	-	-	-		2	
Cond. No. 2	-	-	-	0.01	0.01	5	8.8-8.9
Cu-Pb ro flot. (2nd-stage)	-	-	-	-		3	

*When used (see Table 4)

TABLE 4

Effect of $ZnSO_4$ and NaCN on Zinc Losses to the Copper-Lead Rougher Concentrate

Zn Depressant lb/ton of feed		Product	% Weight	Assay*				Distribution %			
$ZnSO_4$	NaCN			Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Nil	Nil	Test CH-1									
		Cu-Pb ro conc	9.20	181.00	1.85	11.20	7.60	94.0	82.6	72.3	49.4
		" " " tail	90.80	1.16	0.034	0.44	0.79	6.0	17.4	27.7	50.6
		Head (calcd)	100.00	17.70	0.21	1.46	1.41	100.0	100.0	100.0	100.0
2	Nil	Test CH-3									
		Cu-Pb ro conc	6.65	240.0	2.80	20.00	6.04	94.5	87.0	90.4	29.0
		" " " tail	93.35	1.01	0.029	0.15	1.05	5.5	13.0	9.6	71.0
		Head (calcd)	100.00	16.87	0.21	1.48	1.38	100.0	100.0	100.0	100.0
2	0.05	Test CH-4									
		Cu-Pb ro conc	3.17	532.0	5.18	42.0	4.70	93.9	79.5	88.0	10.2
		" " " tail	96.83	1.15	0.048	0.19	1.37	6.1	20.5	12.0	89.8
		Head (calcd)	100.00	18.03	0.23	1.51	1.47	100.0	100.0	100.0	100.0

*Assay in this and all subsequent tables is expressed in per cent, except Ag which is expressed in oz/ton.

These results showed that, although the addition of zinc sulphate alone reduced the zinc loss to the copper-lead rougher concentrate from 49.4% to 29.0%, the addition of 0.05 lb/ton of sodium cyanide in combination with the zinc sulphate reduced the zinc loss to 10.2%. The addition of the above amount of cyanide reduced the lead recovery slightly but the copper recovery was reduced to a larger degree. To minimize the copper losses and still reduce the zinc losses, both zinc depressants were used throughout the investigation but the amount of cyanide was reduced to about 0.04 lb NaCN per ton of flotation feed. The amount of zinc sulphate was maintained at 2 lb per ton of the flotation feed.

Flotation and Tabling Procedure

Results of the preliminary test work showed that the rougher flotation produced a rougher concentrate with good copper, silver and lead recoveries and a low zinc loss. This concentrate comprised less than 4% of the ore by weight. As the main objective of this investigation was to produce separate copper, lead and zinc concentrates, various methods for achieving this objective were tried. One of the successful procedures, consisting of flotation and tabling, will be described in detail.

This procedure consisted of grinding the ore to minus 100 mesh, floating the copper, silver, and lead minerals in a copper-silver-lead rougher concentrate by applying the same flotation conditions as in the "Preliminary Test Work," using zinc sulphate and 0.04 pounds of cyanide per ton of ore for sphalerite depression.

The separation of the copper-silver minerals from galena, to obtain a separate copper-silver and a separate lead concentrate, was done by flotation applying the dichromate method. Dichromate depresses galena but does not affect flotation of chalcopyrite and tetrahedrite. The latter two float and form a copper-silver concentrate and the separation tailing, containing most of the galena, becomes the lead concentrate.

The separation was accomplished by a three-stage addition of potassium dichromate. In the first separation stage, dichromate was added to the copper-silver-lead rougher concentrate, and a first copper-silver concentrate and a first lead concentrate (separator tail-1) were obtained. The first copper-silver concentrate then was used as a flotation feed for the second-stage separation where more potassium dichromate was added and a second copper-silver concentrate and a second lead concentrate (separator tail-2) were obtained. This separation was repeated a third time and a final copper-silver concentrate and a third lead concentrate (separator tail-3) were obtained. The three lead concentrates (the three separator tailings) were combined to form a lead concentrate. The copper-silver concentrate had a sufficient copper grade for smelting operation.

As the lead grade of the combined lead concentrate was about 34% Pb this concentrate was tabled on a sand-deck shaker table and a sand-deck lead concentrate of about 75% lead grade was obtained. The sand-deck tabling was then re-tabled on a slime-deck table and a slime-deck lead concentrate of about 55% lead grade was obtained.

The lead analyses of the slime-deck tailing showed that a fair amount of lead was still left in this tailing. A microscopic examination of this tailing showed that most of the galena present was free, i.e. liberated from other minerals and in sizes finer than 200 mesh. As a result it was decided to use this tailing as a flotation scavenger feed to recover the fine galena. The depressing effect of the dichromates should have been eliminated or considerably diminished after two-stage tabling. Small amounts of collector and frother were added to the scavenger float and a scavenger lead concentrate containing about 35% lead was obtained.

The three lead concentrates produced (two table concentrates and the scavenger concentrate) were combined to form a final lead concentrate. The scavenger tailing was assayed separately.

Flotation of sphalerite to obtain a final zinc concentrate with a zinc grade of over 50% was done in the conventional manner by applying a two-stage flotation procedure using lime, copper sulphate and frother for the rougher flotation, and lime in the cleaner flotation.

Detailed flotation conditions of this procedure are recorded in Table 5, flowsheet in Figure 3 and the results in Table 6.

TABLE 5

Flotation Conditions

Operation	% Solids	Reagents - lb/ton of Flotation Feed								
		CaO	$ZnSO_4$	NaCN	Xanth*	Froth**	$CuSO_4$	Dichr.	pH	Time Min
Conditioner 1		0.3	2.0	0.04	0.008	0.008			8.8	5
Cu-Pb ro float	25									5
Conditioner 2								0.2	6.7	5
Separator float 1	10									3
Conditioner 3								0.2	6.3	5
Separator float 2	5									3
Conditioner 4								0.2	5.9	5
Separator float 3	3									3
Conditioner 5		0.2			0.002	0.002	0.036		10.1	8
Zn ro float	20									4
Conditioner 6		0.04							11.0	5
Zn cl float	8									3

* Sodium Isopropyl xanthate
** Dowfroth 250

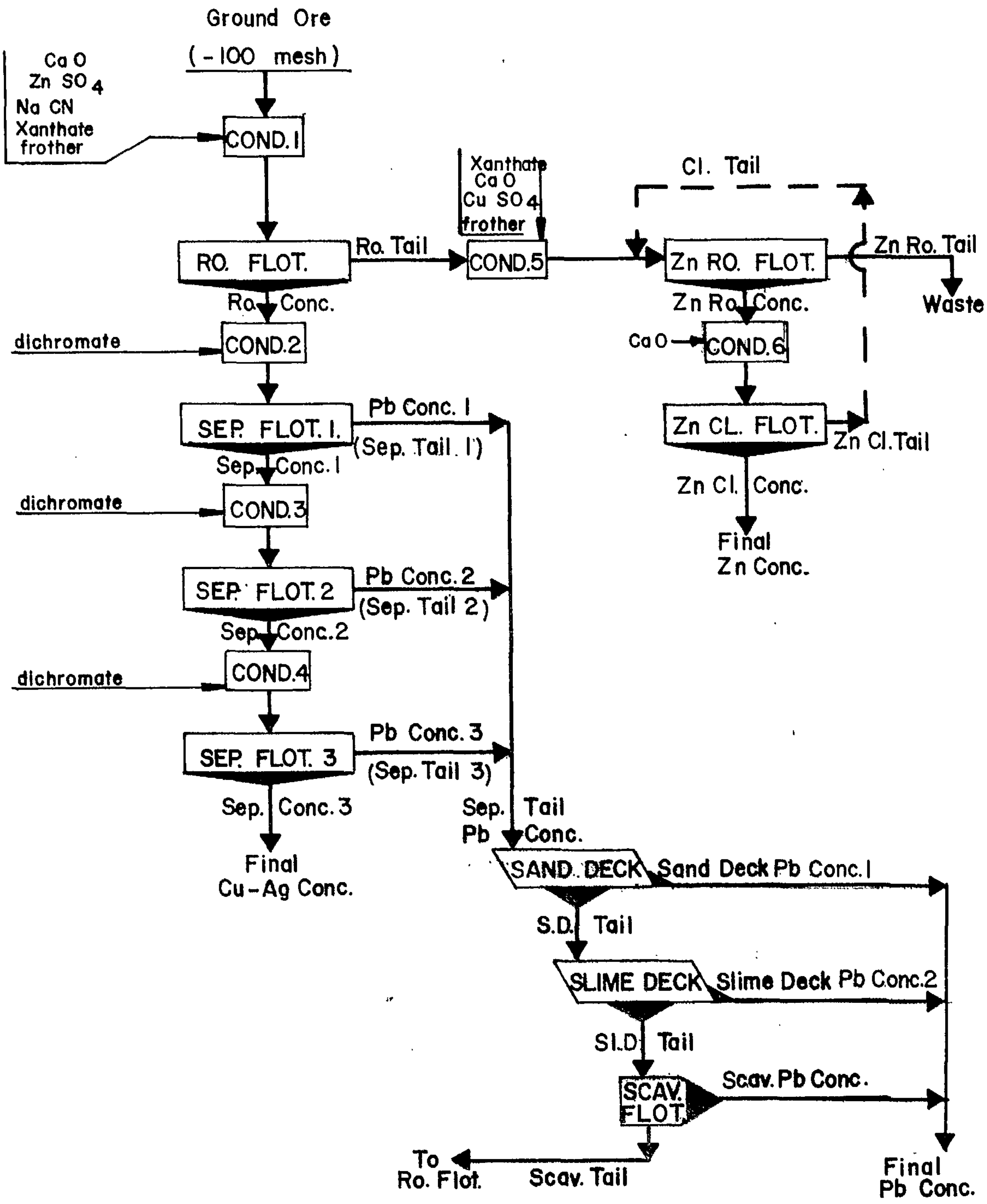

FIGURE 3 FLOTATION AND TABLING PROCEDURE

TABLE 6

Results of Flotation - Tabling Procedure

Product	% Weight	Assay				Distribution %			
		Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Test CH-9									
Copper-silver conc	0.72	1677.0	16.92	16.47	11.37	67.6	60.4	8.2	5.5
Lead concentrate	1.51	257.0	2.45	67.20	3.30	22.0	18.3	70.2	3.3
Lead scav. tailing	1.66	45.2	0.53	8.00	2.47	4.3	4.0	9.1	2.7
Zinc cl concentrate	1.75	11.0	0.26	0.71	57.87	1.1	2.0	0.7	67.0
Zinc cl tailing	2.00	14.37	0.28	1.32	11.57	1.5	2.0	2.1	15.3
Zinc ro tailing	92.36	0.68	0.03	0.15	0.10	3.5	13.3	9.7	6.2
Feed (calcd)	100.00	18.00	0.20	1.44	1.45	100.0	100.0	100.0	100.0

These results showed that the copper-silver concentrate contained a high amount of silver and had an acceptable copper grade for smelting with 67.6% silver recovery and 60.4% copper recovery. The lead and the zinc losses to this concentrate were 8.2% of the lead and 5.5% of the zinc in the ore.

The final lead concentrate produced after tabling and scavenger flotation had a high lead grade (67.20% Pb) containing 70.2% of lead in the ore but the scavenger tailing contained 9.1% of the lead. About one half of this lead would be recovered while this tailing was recirculating in the plant operation, therefore the overall lead recovery in the lead concentrate would be about 75 per cent, but the lead grade of the concentrate would be lowered to about 62% lead.

The lead concentrate (including the recycling of the lead scavenger tailing in a plant operation) would contain about 20% of the copper in the ore; see Table 6.

The zinc loss to the copper and lead concentrates combined was 8.8% of the zinc in the ore.

The zinc cleaner concentrate produced contained 67% of the zinc in the ore with a 57.87% zinc grade. The zinc cleaner tailing contained 15.3% of the zinc. About one half of the zinc present in the zinc cleaner tailing would be recovered while this tailing was recirculated in a plant operation, therefore it can be expected that an overall zinc recovery would be about 75% but the zinc grade of the concentrate would be lowered to about 55% zinc.

Jigging and Flotation Procedure

To investigate a possibility of reducing the operating cost, it was decided to pre-concentrate the ore prior to flotation, reject a portion of the gangue material,and use the pre-concentrate as a flotation feed.

In many cases jigging is an effective and inexpensive method of achieving this objective particularly if the mineralogical association of the valuable minerals and the gangue material is favourable. The pre-concentration and flotation steps of this procedure will be described separately.

Jigging of 10-Mesh Crushed Ore

The ore, crushed to minus 10 mesh, was jigged in the usual manner using a 1 M Denver laboratory jig. Two tests were done with a slight variation in the amount of ragging while the other conditions were constant.

The jigging results are recorded in Table 7.

TABLE 7

Jigging Results

Amount of Ragging	Product	% Weight	Assays				Distribution %			
			Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Medium	Test CH-11									
	Jig conc	15.5	89.00	0.96	8.09	6.10	72.5	66.0	76.1	64.7
	" bed	8.2	5.10	0.07	0.51	0.40	2.2	2.7	3.2	2.2
	" tailing	76.3	6.31	0.09	0.44	0.64	25.3	31.3	20.7	33.1
	Feed (calcd)	100.0	19.03	0.22	1.64	1.47	100.0	100.0	100.0	100.0
Low	Test CH-12									
	Jig conc	21.0	64.55	0.75	5.72	4.84	73.2	71.2	75.5	71.7
	" bed	15.5	4.66	0.06	0.67	0.22	3.9	4.0	6.5	2.3
	" tailing	63.5	6.67	0.09	0.45	0.58	22.9	24.8	18.0	26.0
	Feed (calcd)	100.0	18.52	0.22	1.59	1.42	100.0	100.0	100.0	100.0

Jigging is not very efficient at recovering fine material, therefore considerable fine valuable minerals remained in the jig tailing. To determine the metal distribution in the various size fractions, the tailings from these tests were separated into four size fractions, each fraction was assayed separately, and the metal distributions were calculated.

The size and metal distributions in the tailings are recorded in Table 8.

TABLE 8

Size and Metal Distribution in Jig Tailings

Jig Tailing		Assays				Distribution % *			
Mesh	% Wt. in Ore	Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Test CH-11									
Jig tail, +48	47.0	0.60	0.02	0.05	0.10	1.5	4.5	1.4	3.2
" -48+100	9.3	6.65	0.09	0.34	1.21	3.3	3.6	2.0	7.6
Jig tail, +100	56.3	1.60	0.03	0.10	0.28	4.8	8.1	3.4	10.8
" -100+325	15.7	21.50	0.28	1.56	1.87	17.7	20.0	15.0	20.1
" -325	4.3	12.23	0.16	0.87	0.76	2.8	3.2	2.3	2.2
Jig tail, -100	20.0	19.52	0.25	1.41	1.63	20.5	23.2	17.3	22.3
Jig tail, total	76.3	6.31	0.09	0.44	0.64	25.3	31.3	20.7	33.1
Test CH-12									
Jig tail, +48	36.4	0.55	0.02	0.04	0.06	1.1	3.2	0.9	1.5
" -48+100	7.8	3.45	0.05	0.20	0.52	1.5	1.8	0.9	2.9
Jig tail, +100 mesh	44.2	1.06	0.03	0.07	0.14	2.6	5.0	1.8	4.4
-100+325	8.9	24.00	0.27	1.20	2.29	11.5	10.8	6.8	14.3
" -325	10.4	15.73	0.19	1.43	1.00	8.8	9.0	9.4	7.3
Jig tail, -100	19.3	19.43	0.23	1.33	1.60	20.3	19.8	16.2	21.6
Jig tail, total	63.5	6.67	0.09	0.45	0.58	22.9	24.8	18.0	26.0

*In the ore

These results showed that the highest metal content was in the minus 100-mesh fractions of the jig tailings while the plus 100-mesh fractions had low metal content. This was particularly true in Test CH-12 in which the plus 10-mesh fraction of the jig tailing comprized 44.2% weight of the ore and contained 2.6% of silver, 5.0% of copper, 1.8% of lead and 4.4% of zinc. As a result, it was decided to add the minus 100-mesh portion of the jig tailing to the jig concentrate and discard the plus 100-mesh portion of the jig tailing.

The jig beds of the laboratory jigging, being part of the feed, must be always taken into calculation for the metallurgical balance. In an operating plant this would not be the case because most of the values present in the bed would be recovered in the jig concentrate but most of the gangue material present in the jig beds would become part of the jig tailing. As a result the jig beds of these laboratory tests were added to the jig concentrates and the minus 100-mesh portion of the jig tailing so that these three products formed the pre-concentrates.

Results of this pre-concentration procedure are recorded in Table 9.

TABLE 9

Pre-concentration Results

Product	% Weight	Assay				Distribution %			
		Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Test CH-11									
Pre-concentrate*	43.7	41.50	0.47	3.62	3.00	95.2	91.9	96.9	89.2
Jig tail, + 100 mesh	56.3	1.60	0.030	0.10	0.28	4.8	8.1	3.1	10.8
Feed (calcd)	100.0	19.03	0.22	1.64	1.47	100.0	100.0	100.0	100.0
Test CH-12									
Pre-concentrate*	55.8	32.25	0.38	2.79	2.43	97.5	95.0	98.2	95.6
Jig tail, + 100 mesh	44.2	1.06	0.025	0.07	0.14	2.5	5.0	1.8	4.4
Feed (calcd)	100.0	18.52	0.22	1.59	1.42	100.0	100.0	100.0	100.0

*Jig concentrate, jig bed and minus 100-mesh portion of jig tailing.

These results showed that jigging in combination with the separation of jig tailing produced a pre-concentrate with high recoveries of the metal valuables and also produced a rejectable material (plus 100-mesh portion of the jig tailing) comprising between 44 and 56% of the ore by weight with low losses of the valuable metals.

Flotation of Pre-concentrate

Based on the pre-concentration results of Test CH-12, Table 9, a test designated as Test CH-10 was done using about 50 lbs of ore as jig feed, to obtain a larger amount of pre-concentrate. The pre-concentrate of Test CH-10 was composed of jig concentrate, jig bed, both ground to minus 100 mesh and the minus 100 -mesh portion of jig tailing; the plus 100-mesh portion of jig tailing was discarded.

The pre-concentration results of Test CH-10 are recorded in Table 10.

TABLE 10

Results of Pre-concentration Test CH-10

Product	% Weight	Assays				Distribution - %			
		Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Test CH-10									
Pre-concentrate	54.0	31.90	0.35	2.61	2.52	95.8	93.0	97.2	93.6
Jig tail, +10 mesh	46.0	1.66	0.04	0.09	0.21	4.2	7.0	2.8	6.4
Feed (calcd)	100.0	18.01	0.204	1.46	1.50	100.0	100.0	100.0	100.0

It was stated that the jig beds in these laboratory tests were included in the pre-concentrates. In a continuous plant operation this would not be the case; the weight of the jig bed would remain fairly constant and the jig feed would all go to the concentrate or tailing. In view of this, the per cent weight of the pre-concentrate would be smaller than that recorded in Table 9 and 10 and it would comprise about 50 per cent of the ore by weight and the rejected material the other 50 per cent.

In this procedure the pre-concentrate of Test CH-10, ground to minus 100 mesh, was used as a flotation feed. The flotation conditions of the rougher float, separation floats and the zinc flotation were the same as in the Flotation-Tabling Procedure recorded in Table 5. The flowsheet and detailed results of this procedure are given in Figure 4 and Table 11.

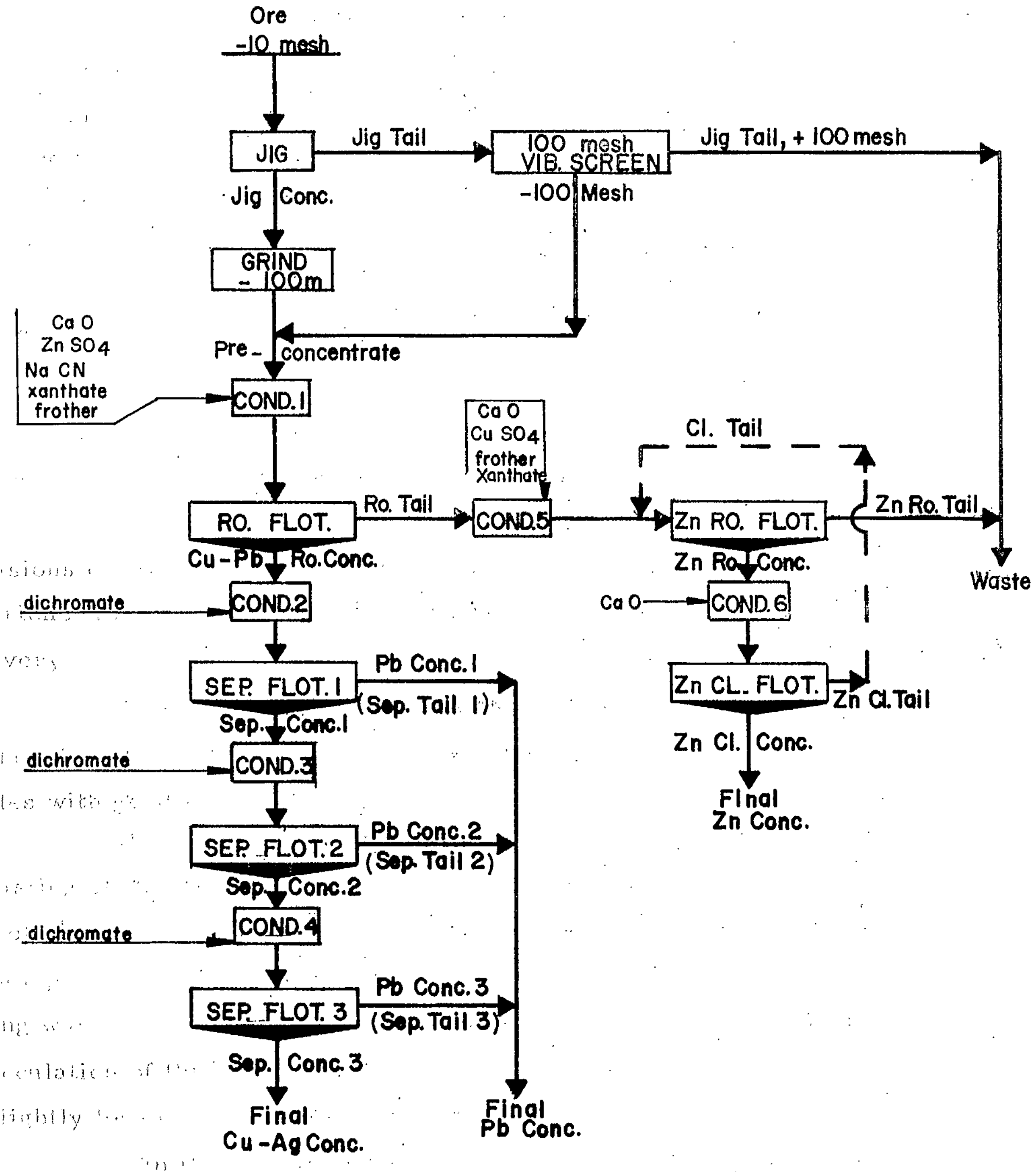

FIGURE 4 JIGGING AND FLOTATION PROCEDURE

TABLE 11

Results of Jigging and Flotation Procedure

Product	% Weight	Assay				Distribution %			
		Ag	Cu	Pb	Zn	Ag	Cu	Pb	Zn
Test CH-10									
Copper-silver conc	0.65	1847.5	19.18	14.09	12.15	66.6	61.2	6.3	5.3
Lead conc	1.85	220.8	1.84	61.60	2.32	22.8	16.7	78.4	2.8
Zinc cl conc	1.75	11.0	0.26	0.71	57.87	1.1	2.4	0.8	67.4
Zinc cl tailing	2.00	14.37	0.28	1.32	11.57	1.7	2.4	1.8	15.4
Zinc ro tailing	47.75	1.36	0.04	0.30	0.09	3.6	10.3	9.9	2.7
Pre-concentrate	54.00	31.90	0.35	2.61	2.52	95.8	93.0	97.2	93.6
Jig tail, +100 mesh	46.00	1.66	0.04	0.09	0.21	4.2	7.0	2.8	6.4
Feed (calcd)	100.00	18.01	0.204	1.46	1.50	100.0	100.0	100.0	100.0

These results showed that the copper-silver concentrate produced by this procedure assayed 19.18% copper and 1847.5 oz silver/ton of concentrate and contained 61.2% of the copper and 66.6% of the silver in the ore.

The lead concentrate (separation tailing)produced by flotation alone (without tabling and without the scavenger flotation) assayed 61.60% lead and contained 78.4% of the lead, 22.8% of the silver,and 16.5% of the copper in the ore.

The zinc loss to both concentrates was only 8.1% of the zinc in the ore.

The zinc cleaner concentrate produced contained 68% of the zinc in the ore and assayed 57.87% zinc. About 15.4% of the zinc in the ore was in the zinc cleaner tails. Because about one half of the zinc values in the zinc cleaner tails would be recovered in the zinc cleaner concentrate while the cleaner tails were recirculated in a plant operation, the overall zinc recovery from this ore sample would be between 75 and 76 per cent in a concentrate that would assay about 55 per cent.

DISCUSSION

The mineralogical examination of the ore and products obtained from this investigation showed that the argentiferous tetrahedrite contained all of the silver, most of the copper, and some zinc in its crystal structure as well as some antimony, iron, arsenic and sulphur (Table 2). Fine inclusions of galena and sphalerite were also observed in the tetrahedrite. As a result, some zinc and lead will be present in the copper-silver concentrate.

Due to small inclusions of tetrahedrite, chalcopyrite, sphalerite, bournonite, and arsenopyrite in galena, the lead concentrate will contain some silver, copper, antimony and arsenic and, to a lesser degree, some zinc regardless of the grade of the lead concentrate.

Only a small portion of the sphalerite contained a few inclusions of other minerals, therefore, no particular difficulty was experienced in obtaining a high-grade zinc concentrate and a good zinc recovery.

Treatment of this ore, by the two procedures developed, produced separate copper-silver, lead, and zinc concentrates of marketable grades with good recoveries.

The copper-silver concentrate produced by the first procedure consisting of flotation and tabling, would contain about 70% of the silver and about 62% of the copper in the ore, if about 50% of the silver and 50% of the copper present in the lead scavenger tailing could be recovered while this tailing was being recirculated in a plant operation (see Table 6). Due to the recirculation of this tailing, the grade of copper-silver concentrate would be slightly lower than that recorded in Table 6.

Similar silver and copper recoveries but higher-grade copper-silver concentrate were obtained by the second procedure (jigging and flotation, see Table 11).

The lead content in the copper-silver concentrate from either procedure was higher than expected. The mineralogical examination of the final copper-silver concentrate showed that a considerable amount of galena present in this concentrate was as free grains. A possible explanation for the presence of this free galena was the low per cent solids and heavy froth in the second and, particularly, in the third separation float.

The lead and silver recoveries in the lead concentrate were about the same for both procedures, ranging from 75 to 78% for lead and from 22 to 24% for silver. In plant operation, the grade of lead concentrate will be about 62% lead and 220 oz silver per ton.

The zinc recoveries from both procedures were about 75% of the zinc in the ore after the usual assumption for recirculation of the zinc cleaner tailings. The zinc cleaner concentrates would assay 55% zinc.

Based on these results, the two procedures developed produced separate copper-silver, lead, and zinc concentrates with good recoveries and in grades acceptable to smelters.

It seems that the jigging-flotation procedure could have some economic advantages over the flotation-tabling procedure.

Because jigging would reject about one half of the ore by weight at a 10-mesh grind, only one half of the ore would require the fine grinding; whereas for the flotation-tabling procedure the entire ore was ground to minus 100 mesh. Jigging would contribute to lower grinding cost because most of the hard siliceous material would be rejected.

By using the pre-concentrate as a flotation feed, i.e., floating about only one half of the ore, the size of the copper-lead rougher circuit and zinc rougher circuit as well as the consumption of the flotation reagents for these two circuits would be reduced by about one half. This would also reduce the capital cost for the flotation plant as well as the flotation operating costs.

The jigging-flotation procedure would not require tabling as in the flotation-tabling procedure but it would require a jigging operation. This could be slightly costlier than tabling, because the tabling operation in the flotation-tabling procedure would be rather small. The small lead scavenger circuit of the flotation-tabling procedure would be also eliminated by the jigging-flotation procedure.

Based on these facts, it seems that the jigging-flotation procedure would have a lower operating cost for treating this ore than the flotation-tabling procedure.

A proper metallurgical evaluation of these two procedures can be done only on a pilot-plant scale.

SUMMARY

Separate copper-silver, lead, and zinc concentrates were produced from this ore by two procedures developed for this purpose.

The first procedure consisted of grinding the ore to minus 100 mesh and obtaining a flotation rougher concentrate containing most of the copper, silver and lead minerals followed by separation of copper-silver minerals from the lead minerals by a flotation dichromate method, tabling the separator tailings (lead concentrates) and then applying a lead scavenger flotation on table tailing, (Figure 3).

The zinc concentrate was obtained by the conventional copper sulphate method. The flotation-tabling procedure recovered about 92% of the silver, 62% of the copper, 75% of the lead and about 75% of the zinc in separate marketable concentrates.

The second procedure consisted of pre-concentrating the 10-mesh-crushed ore by jigging prior to flotation and rejecting about one half of the ore (jig tail) with low metal losses. The pre-concentrate, ground to 100 mesh, was used as a flotation feed from which a final copper-silver, lead and zinc concentrates were obtained by the dichromate method i.e. no tabling and scavenger flotation were required (Figure 4). This procedure recovered 89.5% of the silver, 62% of the copper, 78% of the lead and about 75% of the zinc in the ore. All concentrates were of marketable grades suitable for smelting.

Despite the additional jigging cost, it appears that the jigging-flotation procedure could have some economic advantage because only one half of the ore is treated by flotation and no tabling and scavenger flotation is required. However, a pilot-plant investigation on this ore would be necessary to properly evaluate the two procedures proposed for processing this ore.

ACKNOWLEDGEMENTS

The authors wish to acknowledge the assistance of the staff of the Mineral Science Division, namely Dr. D. C. Harris and D. R. Owens for the mineralogical investigations and of the members of the Analytical Chemistry section for the chemical analyses.

The acknowledgements are also extended to the technicians of the Mineral Dressing Laboratory, namely J. C. Banks and M. Raicevic for performing jigging, tabling and other auxiliary operations in this investigation.

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