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BETHLEHEM COPPER CORPORATION LTD.,
ASHCROFT, B.C.

DIFFERENTIAL FLOTATION OF LEAD AND ZINC FROM A BASE METAL-BARITE SAMPLE (LOT III) SUBMITTED BY KAMAD SILVER COMPANY LTD.

MAY 17 TO JULY 16, 1971

BY E. A. LOWE, CHIEF METALLURGIST,
BETHLEHEM COPPER CORPORATION LTD.,
JULY 29, 1971.

Remaining work on lead-zinc metallurgy is of lesser significance, consisting of some optional refinements to the basic differential flotation process. The recovery of a separate copper concentrate by further treatment of the copper-lead-silver bulk concentrate is an alternative which might be dictated by marketing agreements.

Particle size in the flotation pulps appears to be an important factor, particularly in relation to the loss of zinc due to entrainment in the lead concentrate. There was little attention to grinding in the current investigation, and the possibility exists that further work on grinding and regrinding could lead to improved results in both rougher and cleaner flotation stages. A grinding-recovery test series could be a secondary object of future studies.

II. OBJECT

To develop a process for differential flotation of lead, zinc, copper, silver and barite values from a Kamad base metal-barite sample (Lot III).

III. SAMPLE

The Lot III sample consisted of 1,500 pounds minus 8-inch broken rock delivered May 17, 1971, by Messrs. D. Berry and F. Shukin of Kamad Silver Company Ltd. It was reported to be a composite sample of shaft, drift and sub-drift muck from the sponsor's Homestake property.

The following head assays were determined in our mine laboratory:

ITEM		ASSAY, % OR OZ/TON
Pb		2.40
Zn		3.76
Cu		0.48
Fe		4.39
Ag	oz/ton	4.88
Au	oz/ton	0.015

An approximate assay of 7% Ba was returned by the sponsor's analytical laboratory. This value was accepted for use in our laboratory work.

IV. PROCEDURE

In preparation for flotation the sample was dry-crushed to minus 8 mesh by jaw, cone and roll crushers. Remaining reduction to liberation size was by wet grinding and regrinding in small ball mills.

The procedure was developed empirically over the course of the two-month testing period. In its terminal form it consisted of primary grinding followed by the various lead and zinc recovery operations.

Lead recovery involved conditioning and roughing, regrind of the rougher concentrate, and two stages of cleaning to the final bulk copper-lead-silver product. The cleaner tailings were combined and scavenged to recover a middlings product referred to as the lead cleaner scavenger concentrate. The scavenger tailing was returned to the rougher tailing pulp for zinc recovery.

Zinc recovery involved operations somewhat similar to the lead procedure except for distinct differences in reagent and pulp conditions. Long zinc conditioning was followed by rougher flotation, regrind of the rougher concentrate and two stages of cleaning to the final zinc concentrate. The zinc cleaner tailings were not returned for retreatment but were combined and held as a single end product for convenience and economy of assaying.

Subsequent operations for recovery of pyrite and barite involved direct conditioning and rougher flotation steps.

Details of all grinding and flotation operations are presented in following tabulations. The laboratory flow sheet is shown in schematic outline.

1. GRINDING

Grinding was carried out in 8-inch steel mills carrying 10 Kg ball charges, turning at 76 rpm. Bethlehem fresh water was used in grinding and flotation. Regrinding was carried out in the same units.

ITEM	MAGNITUDE
Number of mills ground	18
Grams per mill	2,500
Pulp density, % solids	65
Time, minutes	25
Reagent, NaCN, lb/ton	0.05

GRIND SIZE BY Zn Ro TAILING SCREEN ANALYSIS	% WT. RETAINED
+ 65	0.9
+ 100	3.0
+ 150	8.7
+ 200	13.6
+ 325	14.9
- 325 mesh	58.9
TOTAL	100.0

2. FLOTATION

Flotation was carried out in three individual 15 Kg tests. The three sets of product weights and assays were combined on a weighted basis to give representative results for a single 45 Kg test.

All rougher flotation operations were carried out in a Galigher 15" cell with the pulp density initially set at 35% solids for commencement of lead flotation. All cleaning operations were conducted in 2,000-gram cells. Regrinding data is included in the flotation tabulation.

1. SUMMARY

Laboratory flotation studies were carried out on a base metal-barite sample submitted in May, 1971, by Kamad Silver Company Ltd. The sample was mineralized in galena, sphalerite, chalcopyrite, pyrite and barite and registered a head assay of 2.40% Pb, 3.76% Zn, 0.48% Cu, 4.88 oz. Ag/ton and approximately 7% Ba. The sponsor was interested in a recovery process for separating the mineral values into concentrate products of marketable grade and composition.

Test work on the base metal values succeeded in development of a differential flotation process yielding two sulfide concentrates bearing good levels of metal recovery. Considerable attention was directed to treatment of the sulfide tailing for barite recovery but the laboratory failed to turn up a method meeting both grade and recovery requirements.

The base metal process yields, firstly, a bulk lead concentrate carrying copper and silver as accessory values and, secondly, a zinc concentrate of substantially clean sphalerite.

The lead concentrate returned a bulk grade of 48.44% Pb, 9.55% Cu, 9.57% Zn and 87.62 oz. Ag/ton, at recoveries of 85.2% for lead, 84.2% for copper and 78.1% for silver. The zinc content is an entrained impurity amounting to a loss of 10.7% in zinc recovery.

The zinc concentrate graded 56.34% Zn at 80.3% open circuit recovery. Entrainment of the associated sulfides imparted grades of 0.50% Cu, 2.02% Pb, and 7.05 oz. Ag/ton to this concentrate, corresponding to recovery values of 4.0% for copper, 4.5% for lead and 8.0% for silver.

For the unfinished work on barite recovery the procedure involved a direct continuation of flotation treatment on the zinc rougher tailing pulp. Remaining sulfides were first removed in a scavenger type flotation step yielding a pyrite rougher concentrate carrying residual lead, zinc and copper values. A low grade barite rougher concentrate of specific gravity, 3.7, was then floated from the sulfide-free pulp. Various attempts at flotation cleaning aimed at producing a concentrate in the specific gravity range 4.3 - 4.5, have been unsuccessful in the laboratory work to date.

A clean barite concentrate reading specific gravity, 4.4, was possible by shaking table treatment of the flotation rougher concentrate. However, the method is slow and barite recovery is poor.

Recovery of barite at a premium grade remains the major unresolved problem in treatment of this particular lean grading base metal-barite sample. An all-flotation solution to the problem is believed to be preferable and possible. The need for more extensive flotation work to investigate reagents and pulp conditions is indicated. This work should be the object of continued or future laboratory studies.

Table 1 a

FLOTATION, REGRIND AND REAGENT CONDITIONS

	LEAD OPERATIONS						ZINC OPERATIONS				
STAGE	CONDITION	ROUGHER	ROUGHER CONCENTRATE REGRIND	CLEANER NO. 1	CLEANER NO. 2	CLEANER TAILING SCAVENGER	CONDITION	ROUGHER	ROUGHER CONCENTRATE REGRIND	CLEANER NO. 1	CLEANER NO. 2
REAGENTS *											
NaCN	(1) .05		.033							(1) .005	(1) .005
$ZnSO_4 \cdot H_2O$	(2) 1.0			(1) .50	(1) .50						
MIBC	(3) .04			.005	.003	.003				.003	.001
S-3501	(4) .0067	.013		.004	.004	.003		.04		.013	.009
$CuSO_4 \cdot 5H_2O$							(2) .30		.10		
Lime							(1) 1.1		.13		
Minewater								to pH		to pH	to pH
Time, Minutes	5	9	25	9	8	5	20	10	25	9	8
Temperature, C		18		17	17		22	22		18	
pH		7.9		7.5	7.4	7.5	10.9	11.0		11.1	11.0
Machine	3 x 15"	3 x 15"	3 x 8"	3 x 2,000	3 x 2,000	3 x 2,000	3 x 15"	3 x 15"	3 x 8"	3 x 2,000	3 x 2,000

* NOTES:

1. Bracketed numbers indicate order of reagent addition
2. Reagent quantities are in units of pounds per ton original feed
3. Reagent legend:

NaCN,	Sodium cyanide
$ZnSO_4 \cdot H_2O$	Zinc sulfate monohydrate
MIBC	Frother methyl isobutyl carbinol
S-3501	Cyanamid dithiophosphoric collector S-3501
$CuSO_4 \cdot 5H_2O$	Copper sulfate pentahydrate

Table 1 b

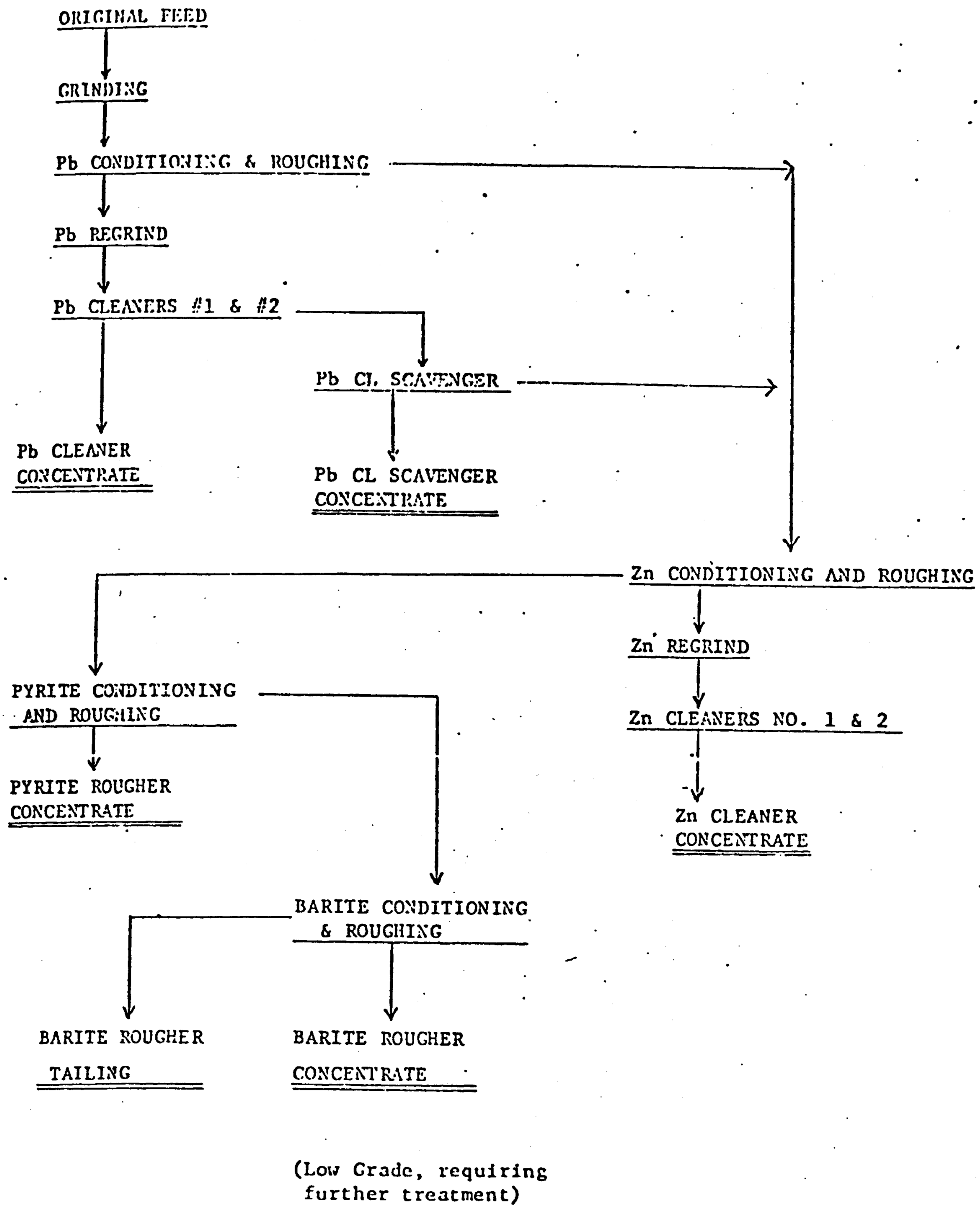
FLOTATION, REGRIND AND REAGENT CONDITIONS

STAGE		PYRITE OPERATIONS		BARITE OPERATIONS	
		CONDITION	ROUGHER	CONDITION	ROUGHER
Reagents *					
H_2SO_4	(1)	1.5		(2) 1.5	
KAX		.02	.02		
A-65		.003	.023		
$Na_2SiO_3 \cdot 9H_2O$				(1) 4.0	
A-825					3.0
Time, Minutes		3	13	10	14
pH		7.7	8.4	9.1	9.1
Machine		3 x 15"	3 x 15"	3 x 15"	3 x 15"

* Reagent Legend:

H_2SO_4	Sulfuric acid
KAX	Potassium amyl xanthate
A-65	Cyanamid frother aerofroth 65
$Na_2SiO_3 \cdot 9H_2O$	Sodium metasilicate
A-825	Cyanamid sulfonate collector aero 825

3. SCHEMATIC OUTLINE OF LABORATORY PROCEDURE



V. RESULTS

Table 2 shows flotation metallurgy for the consolidated 45 Kg test. Head assays calculated from the test products were found to be in good agreement with the assayed head values. All products were assayed for lead, zinc and copper, but silver and gold assays were run on only the head and the lead and zinc concentrates. Silver and gold recovery values are indicated levels based on the assayed head values.

Lead flotation resulted in a final copper-lead-silver concentrate grading 48.44% Pb, with an entrained zinc content of 9.57% Zn. The recovery levels were quite favourable, registering 85.2% for lead, 84.2% for copper and 78% for silver.

Complete elimination of the zinc calculates to a theoretical grade of 60% Pb. In the laboratory testing our highest concentrates ran near 51% Pb; in practice this level may be close to the maximum attainable. A portion of the sphalerite appears to be highly activated, resisting normal measures to reject it from the lead concentrates. The possibility exists that this entrained zinc is in the form of locked middlings and that its rejection may be enhanced by more careful attention to regrinding. Our test procedure did not encounter difficulty with lead sliming, hence there appears to be some latitude for finer grinding of the rougher concentrates, and further studies in this area should be carried out to establish optimum zinc rejection.

No work was carried out on separation of copper from the laboratory lead concentrates. The copper content appears to be in the form of chalcopyrite and its separation may be possible by a conventional flotation technique. Complete elimination of copper from the bulk lead concentrate calculates to a theoretical 73% Pb product. However, the optimum practical metallurgy for such a separation would have to be determined by test work.

Zinc flotation resulted in a final concentrate grading 56.34% Zn at 80.3% recovery. The zinc was found to be readily flotable and the mineralization was observed to vary from white sphalerite through to brownish-black marmatite, hence the laboratory grade appears to be near the maximum attainable. The major losses in zinc recovery were due to entrainment of active mineral in the lead products, 95% of the total zinc being accounted for by the two lead concentrates and the zinc concentrate. It is apparent from the test results that any further steps to enhance lead metallurgy by rejection of entrained zinc will very likely give rise to a simultaneous improvement in zinc recovery. Less than 5% zinc recovery is accounted for by the combined zinc rougher and cleaner tailing products, so no further contributions to zinc recovery may be expected from these sources.

TABLE 3

BARITE RECOVERY BY TABLING

PRODUCT	% WT.	SPECIFIC GRAVITY	INDICATED % Ba	RECOVERY
Table Concentrate	14.18	4.44	+ 96	21
Table Tailing	85.82	3.62	62	79
Feed: Ba Rougher Concentrate	100.00	3.72	67	100

The positive metallurgical results found for the base metal constituents unfortunately did not extend to barite metallurgy. The procedure of scavenger flotation to recover a pyrite rougher concentrate created a clean non-sulfide tailing pulp from which attempts were made at recovering a clean barite product.

In the terminal tests a barite rougher concentrate was produced registering 22.0% weight at specific gravity 3.72. This corresponds to a 67% barite grade and converts to a theoretical 15% weight of pure barite at specific gravity, 4.5. Based on the head assay of 7% Ba, or 12% barite provided by the sponsor, it is evident that a high recovery is obtained in the rougher concentrate.

In the work to date the problem has been that the rougher concentrate has not responded in subsequent cleaning stages to yield a premium grade end product. The best cleaner products have been in the area of specific gravity, 3.9 - 4.0, far short of the target range S.G. 4.3 - 4.5. Nevertheless, a flotation solution appears to offer the best hope for eventual success and test work in this direction should be continued or resumed at a future date.

With no strong indications of a workable flotation method at hand, recourse was made to gravity treatment to obtain a sample of clean barite for the sponsor's reference purposes. Table 3 presents results of shaking table treatment of the flotation rougher concentrate. The table concentrate recovered 21% of the barite at a premium grade of specific gravity, 4.44, indicating over 96% barite purity. The table tailing measured S.G. 3.62, indicating a grade of 62% barite, accounting for a loss of 79% of the contained barite. The table middlings were totally recirculated.

The tabling process was found to be quite slow and tedious and a large part of the barite loss was observed to be due to a rafting effect caused by residual collector coating on particles and entrainment of air during feeding, resulting in short circuiting of much barite to the tailing side of the table. Although a clean barite product can be obtained by this method, it does not carry much hope for good capacity or high recovery.

E.A. [illegible]

Subsequent to the report submitted July 29, 1971, presenting favourable results of laboratory work on flotation recovery of separate lead-copper and zinc concentrates from Kamad base metal-barite sample Lot III, some additional work was carried out on the unresolved problem of barite recovery.

This new work tested a specific barite reagent, trade-named Stepanflote 24, recommended by Stepan Chemical Company. The testing returned highly favourable results for both grade and recovery of barite.

Metallurgical data and results are presented in tabulations following. The flotation procedure involved first a clean-out of sulfides in a bulk sulfide stage, recovering copper, lead, zinc and iron sulfides into a single product. The barite was then floated directly from the sulfide tailing pulp in a rougher step, and finished by four stages of cleaning.

Because of limitations in analytical methods for barite an exact percentage recovery figure has not been determined, but the available data indicates a level in excess of 90% barite recovery. The barite concentrate reported a weight recovery of 15.29%, or 305.8 pounds per ton of original feed, with a measured purity of specific gravity 4.40.

Recourse was made to the bulk sulfide flotation short-cut, since the earlier work had adequately resolved the problems in differential flotation of the sulfides. With this terminal work on barite recovery the indications are that an all-flotation procedure is technically feasible for the Kamad Lot III base metal-barite sample.

E. A. [illegible] P. ENG.

A. BARITE FLOTATION METALLURGY (TEST K-98)

PRODUCT	WEIGHT, %	PURITY
Barite Cleaner Conc. #4	15.29	S.G. 4.40 Feb. 18, 1972 $BaSO_4$ 98.2 Ba 57.8
Barite Cleaner Tail. #1-#4	6.06	
Barite Rougher Tailing	55.51	
Bulk Sulfide Concentrate	23.14	
BULK	100.00	

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ASHCROFT, B. C.

DIFFERENTIAL FLOTATION OF LEAD AND ZINC FROM A BASE METAL-BARITE SAMPLE (LOT III) SUBMITTED BY KAMAD SILVER COMPANY LTD.

MAY 17 TO JULY 16, 1971

BARITE FLOTATION SUPPLEMENTARY REPORT

NOVEMBER 12, 1971

BY

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BARITE FLOTATION DATA (TEST K-98)

ITEM	OPERATION								
	GRIND	BULK SULFIDE CONDITIONING	BULK SULFIDE FLOTATION	BARITE CONDITIONING	BARITE ROUGHER	BARITE CLEANER #1	BARITE CLEANER #2	BARITE CLEANER #3	BARITE CLEANER #4
Time, Minutes	25' per 2500 grams	15	15	3	12	6	5	5	5
Reagents, Pounds per Ton									
Copper sulfate pentahydrate		.30							
Potassium amyl xanthate			.067						
Frother A-65			.032						
Frother MIBC									
Stepanflote 24				.67	1.33				.012
Sodium metasilicate						.40	.40	.40	.40
pH			8.7		8.7	9.4	9.7	9.6	9.6
Temperature, F			72		72				
Machine	6 x 8"	15"	15"	15"	15"				