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REPORT

of the

ORE DRESSING AND METALLURGICAL LABORATORIES

Experimental work on Whitewater lead-zinc tailings from Retallack, British Columbia

+ by C. S. Parsons +

Shipment: A shipment of 65 pounds of lead-zinc tailings was received December 9, 1926 from Mr. M. S. Davys, Kaslo, B.C. This sample was from the Whitewater tailing dumps at Retallack, B.C.

Characteristics and analysis: The sample received was a tailing from the early operations of a gravity concentration mill on the Whitewater mine ore. The Whitewater tailing dump is of considerable tonnage, lying in the bed of Kaslo Creek, extending for about one half mile below the mine. The chief values are zinc and silver, but galena up to one per cent is present, and also small values in gold. Part of the galena is oxidized making its recovery by flotation difficult.

Analysis of the sample:

Lead	1.46 %
Zinc	6.65 %
Silver	7.31 oz/ton

Purpose of tests: The sample was submitted for the purpose of having flotation tests made to obtain information as to the relative merits of selective flotation as compared to bulk flotation followed by tabling to separate the lead and zinc in the bulk concentrate.

Experimental tests: The various tests will be described separately as each one was made to obtain certain information to compare the different methods of treatment.

Test No. 1 - Selective Flotation. The crude tailings were crushed dry to 14 mesh. A 1000 gram sample was taken and ground in a small ball mill for 15 minutes. The following reagents were used.

Soda carbonate	10 lbs/ton	Added at ball mill
Cyanide	0.25 "	15 mins. time contact
Zinc sulphate	2.0 "	15 " " "
Phospho cresylic acid	0.15 "	For lead flotation
Copper sulphate	1.5 "	For zinc flotation
Xanthate	0.3 "	" " "

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Results:

Product	Weight %	Assays Pb %	Zn %	Au oz	Ag oz	Per cent of values Pb	Zn	Au	Ag
Lead conc.	3.5	17.30	24.11		127.5	40.8	12.2		59.8
Zinc "	15.4	1.56	36.01	0.01	12.5	16.0	79.4		25.5
Tailing	81.1	0.80	0.72	tr	1.36	43.2	8.4		14.7

Tests Nos. 2 & 3 - Two tests were made by floating a bulk concentrate containing both the lead and zinc and endeavouring to make a selective separation on the bulk concentrate. The two tests were unsuccessful and the results too poor to report.

Test No. 4 - This is a selective flotation test using cyanide alone and adding it to the ball mill. The zinc concentrate was recleaned.

Reagents:

Soda carbonate	10 lbs/ton	added to ball mill
Cyanide	0.25 "	" " " "
Phospho cresylic acid	0.15 "	For lead flotation
Copper sulphate	1.5 "	3 minutes contact before
Xanthate	0.3 "	zinc flotation

Results:

Lead conc.	3.9	14.78	14.42	0.14	112.38	40.3	8.0	60.9
Zinc "	9.0	1.30	57.25	0.02	12.78	8.2	73.0	15.9
Zinc midd.	4.6	3.0	20.52	tr	17.50	10.4	13.3	11.2
Tailing	82.5	0.72	0.49	tr	1.06	41.1	5.7	12.0

Test No. 5 - The purpose of this test was to determine if a flotation concentrate could be taken off the cells containing the bulk of the lead but only a small proportion of the zinc, after which the zinc would be floated in the usual manner by the addition of copper sulphate. In this operation the copper sulphate used to promote the flotation of zinc would not be added until after the first concentrate had been floated. The advantage of such an arrangement would be that a smaller quantity of material would have to be tabled for the lead and zinc separation - it being necessary to table the first float only. The second float would, after cleaning, be a final zinc concentrate. In this test the zinc concentrate was not recleaned. In practice this concentrate should be recleaned and the middling from the cleaner cell be sent back to the head of the second flotation. Reagents used:

Soda carbonate	10 lb/ton	added to ball mill
Acid Creosote (D.T & C.Co)	0.35 "	" " " "

floated first concentrate with these reagents and then added:

Copper sulphate	1.5 lb/ton	3 minutes contact
Xanthate	0.4 "	added to cell

Results:

Lead conc.	9.7	7.04	23.70	0.02	48.18	45.9	32.4	64.0
Zinc "	11.6	2.06	37.76	tr	14.64	16.1	62.1	23.5
Tailing	78.7	0.72	0.30	tr	1.16	38.0	5.5	12.5

Remarks: It is difficult to see from the results of this test how it

would pay to table the second concentrate which only contains 2.06% lead. The extra loss of silver and zinc in slimes resulting from tabling would also be considerable.

Test No. 6 - On this test a bulk flotation of the lead and zinc was made, the concentrate being recleaned. The object was to obtain results by which this method could be compared with the previous method of treatment. Reagents used:

Soda carbonate	12 lbs/ton	Added to ball mill
Acid coal tar creosote	0.35 "	" " " "
Copper sulphate	1.5 "	Time contact 3 minutes
Xanthate	0.4 "	Added to cells
Pine oil #5	0.02 "	" " "

Results:

Product	Weight %	Assays				Per cent of values			
		Pb %	Zn %	Au oz	Ag oz	Pb	Zn	Au	Ag
Flot. conc.	11.0	5.03	50.52	0.18	50.04	37.3	81.8		70.4
Flot. midd.	5.3	4.43	17.26	0.12	24.16	15.7	13.4		16.3
Tailing	83.7	0.83	0.39	tr	1.24	47.0	4.8		13.3

Test No. 7 - The poor recovery of lead obtained in the previous tests indicated that it was oxidized. This test was run using sodium sulphide to sulphidize the oxidized lead. The ore at -14 mesh was ground for 15 minutes in a ball mill with the following reagents:

Soda carbonate	12 lbs/ton
Acid coal tar creosote	0.35 "

Then 2 lbs/ton sodium sulphide, calculated as Na_2S was added, the pulp density being kept as thick as possible. It required 8 minutes to consume the sodium sulphide, the pulp being tested with lead acetate until no free sulphide was found. Free sodium sulphide is detrimental to flotation when present in the pulp. A selective flotation was then made, advantage being taken of the selective action of sodium sulphide. A lead product was floated which was not in this case recleaned. Recleaning however would be necessary in practice. A zinc concentrate was then floated by the addition of

Copper sulphate	2 lbs/ton	3 minutes time contact
Xanthate	0.4 "	added to cells
Pine oil	approx 0.02 "	to froth

Results:

Lead conc.	3.0	13.38	16.65	0.12	64.94	47.7	12.4	35.8	48.3
Zinc "	10.4	2.57	53.14	0.10	23.08	19.0	81.7		35.6
Zinc midd.	3.5	2.21	4.53	0.01	9.07	5.6	2.4		4.8
Tailing	81.1	0.48	0.29	tr	0.94	27.7	3.5		11.3

Test No. 8 - In this test the ore was ground much finer before floating, the object being to determine whether finer grinding increased the

recovery of the lead. The same reagents were used, and the same procedure followed, as in Test No. 7

Results:

Products.	Weight %	Assays Pb %	Zn %	Au oz	Ag oz	Per cent of values Pb	Zn	Au	Ag
Lead conc.	11.9	8.35	16.68	0.10	44.34	68.8	29.4		72.0
Zinc "	7.7	1.51	54.6	0.04	15.64	8.1	63.2		16.7
Zinc midd.	3.3	1.91	7.11	0.04	8.20	4.5	3.5		3.7
Tailing	77.1	0.35	0.34	trace	0.72	18.6	3.9		7.6

Test No. 9 - This test is a duplicate of Test No. 7

Results:

Lead conc.	6.5	11.57	10.25	0.08	42.36	49.6	9.6		36.6
Zinc "	10.7	2.62	54.18	0.04	31.26	18.7	84.0		44.9
Zinc midd.	2.8	3.22	6.90	0.03	18.79	6.1	2.8		7.2
Tailing	80.0	0.48	0.31	0.002	1.00	25.6	3.6		11.3

Test No. 10 - This test is a duplicate of Test No. 8

Results:

Lead conc.	6.2	11.87	10.25	0.08	42.48	49.7	9.3		37.8
Zinc "	10.7	2.67	55.33	0.04	30.52	19.2	86.1		46.8
Zinc midd.	3.1	1.96	2.69	0.03	8.17	4.1	1.2		3.6
Tailing	80.0	0.50	0.39	0.002	1.02	27.0	3.4		11.8

Test No. 11 - The ore for this test was ground for 20 minutes in the ball mill. The object of the test was to determine the effect of using sodium sulphide when making a bulk concentrate which in practice would be tabled to separate the lead and zinc. Reagents used:

Soda carbonate	12 lbs/ton	Added to ball mill
Acid coal tar creosote	0.35 "	" " " "
Na_2S (sulphidizing period)	2.5 "	(approx. 8 lb. commercial salt) Time contact 8 mins.
Copper sulphate	2.0 "	Time contact 2 mins.
Xanthate	0.4 "	Added to cells
Pine oil	0.02 "	" " "

made bulk concentrate and recleaned.

Results:

Flot. conc.	12.0	6.30	51.77	0.26	40.54	55.8	88.5		72.2
Flot. midd.	5.5	4.48	7.11	0.22	15.32	18.2	5.6		12.7
Flot. tailing	82.5	0.43	0.50	0.025	1.23	26.0	5.9		15.1

(note - this test probably salted from high grade gold ore)

Summary: Tests 1 to 4 are selective flotation tests. Test No. 5 represents an attempt to gather the lead into a small weight of concentrate by taking advantage of the natural property of lead to float more readily than zinc. This product, even under ideal conditions would run high in zinc and would have to be tabled for the lead. The advantage to be gained by such practice is that the main bulk of the zinc is floated afterwards and does not have to be tabled. Test No.6 is a straight bulk flotation test, the object being to determine the grade of concentrate and recovery which should be readily obtained in

practice. Tests 7 to 10 are selective flotation tests in which the oxidized lead was sulphidized by the use of sodium sulphide. In Test 10 the ores was ground finer than in the other three tests. Test No. 11 is a bulk flotation test in which the oxidized lead was sulphidized and then floated in bulk with the zinc.

Conclusions: It can readily be seen that selective flotation is not feasible on these tailings. The principle reason that they do not respond, is the oxidized condition of the lead. Test No. 6 represents ordinary bulk flotation practice and the results can be used as a basis of comparison when considering the results of other tests. This test shows that a 50% zinc concentrate can easily be produced by one recleaning. The recovery of the zinc is high - 81.8% in the concentrate and 13.4% in the middling, which in practice is continuously returned to the feed to the cells. This means that a 90% recovery of the zinc can be obtained in practice. The silver recovery is high showing 70.5% in the bulk concentrate and 16.4% in middling. It is not safe to assume over a 50% recovery of the silver in the middling, so that 78% total recovery is about the maximum. The lead recovery is low, 37.3% in the concentrate and 15.7% in the middling. The total recovery can be assumed at 45% in a bulk concentrate assaying about 5% lead, 50% zinc with 50 ozs of silver. This bulk concentrate would have to be tabled to separate the lead, and the table losses cannot be determined by small scale tests. As this is the practice at present used on this material the operators can form a close estimate of what they will be.

By comparing the results of tests in which sodium sulphide was used for a sulphidizer, it will be observed that in every test the recoveries of the three metals are higher. Tests 7 and 9 are duplicate tests. A selective separation was attempted in these tests by the use of sodium sulphide sulphidizing of the oxidized lead. These tests show rather promising possibilities. There is a slight variation in the results of the two tests. More of the silver reports in the lead product in Test No. 7 than in Test No. 9. Test No. 9 is the better figure to use in any calculations made to determine the commercial possibilities of this separation.

Tests Nos. 8 and 10 are duplicate tests and are similar to tests nos. 7 and 9 with the exception that the feed was ground much finer. Test No. 10 represents the better practice of the two on account of the flotation of the lead product in Test 8 being carried too far which forced considerable zinc into it. The results of Test No. 10 are practically the same as those of Test No. 9, which shows that finer grinding is not necessary.

The operator can use the results obtained in Tests Nos. 7, 9, and10 to calculate whether the production and marketing of a lead product by this method will increase the value of his ore sufficiently to compensate for the cost of the additional equipment required. It should be kept in mind that the lead in this concentrate is really recovered, while the lead recovered in the bulk lead and zinc concentrate of tests 6 and 11 will be subjected to further loss when tabling to produce a lead concentrate. The additional equipment required would consist of a surge tank to give a time contact of 10 minutes for the addition of sodium sulphide. Extra cell capacity for flotation of the lead rougher concentrate, and a cleaner cell for recleaning the lead concentrate.. Ample capacity should be allowed when figuring on size of lead cells as the material floats slowly.

Test No. 11 represents the use of sodium sulphide sulphidization for the production of a bulk lead zinc concentrate. While the writer has no cost figures available to estimate the returns from such a product sent direct to the smelter, this practice looks the most promising. The additional equipment required would be a surge tank for a 10 minute contact of sodium sulphide, and additional cell capacity for recleaning of the concentrate. If Callow cells are used for the cleaners, they should be short, not over 8 ft. in length, and preferably of the flat bottom type (this applies to cleaner cells only) For the production of the low tailing obtained in these tests the present cell capacity for the production of the rougher concentrate should at least be doubled.

It will be observed that all the tests were made on the crude sample. The writer understands that the practice used at present is to jig the crude material and collect a concentrate through the

hutches of the jig, which is then reground and floated. It would appear to the writer that there must be considerable loss of lead by this practice because a large part of the lead in the crude material is present as slime.

In conclusion, the writer wishes to state that it is impossible to make any definite recommendations without a careful study of all the costs of milling and the freight and treatment charges on the products.