IR 1717

O T T A W A October 10th, 1944.

R E P O R T

of the

ORE DRESSING AND METALLURGICAL LABORATORIES.

Investigation No. 1717.

Concentration of a Tin Ore from the Regal Silver Mines Limited, Albert Canyon, British Columbia.

(Copy No.___.)

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Shipment:

A shipment of 66 bags of ore, total weight 6,000 pounds, was received on August 26th, 1944, from the Snowflake property of Regal Silver Mines Limited. This shipment was consigned to these Laboratories by A. S. MacCulloch, 501-02 Vancouver Block, Vancouver, British Columbia.

Previously, in June 1929, and May 1943, other ore shipments had been received from this property. These were covered by Reports of Investigation Nos. 337 and 1404, respectively.

Location of the Property:

The Snowflake property of the Regal Silver Mines Limited, is situated in Albert Canyon, Revelstoke mining division, British Columbia.

Purpose of Investigation:

To obtain sufficient tin concentrate to carry out experimental work by fusion and leaching for recovery of tin.

Sampling and Analysis:

After crushing, cutting and grinding by standard methods, a representative sample of the shipment was obtained which assays as follows:

Gold	-	0.005	oz./ton.
Silver	-	24.66	"
Copper	-	3.46	per cent.
Tin	-	3.40	"
Lead	-	3.50	"
Zinc	-	7.82	"
Iron	-	15.44	"
Sulphur	-	21.37	"

Characteristics of the Ore:

There was no microscopic examination of polished sections from the present shipment. As given in Report of Investigation No. 1404, "metallization is strong in the polished sections and is represented by stannite, pyrite, sphalerite, chalcopyrite, and galena. For the most part, these minerals occur in complex admixture."

From the chemical analysis, the approximate quantities of the different metallic minerals are calculated as follows:

		Per cent
Stannite	-	12.4
Pyrite	-	24.7
Sphalerite	-	11.7
Chalcopyrite	-	0.6
Galena	-	4.1

Results of Investigative Work:

The work performed consisted of small-scale flotation tests to check former investigations, followed by a mill run on the main bulk of the ore shipments to obtain tin concentrate.

(Continued on next page)

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(Results of Investigative Work, cont'd) -

In the small-scale work, a tin concentrate was obtained assaying 21.8 per cent tin with a recovery of 82.2 per cent of the tin in the concentrate and middling products.

In the mill run, tin concentrates were secured assaying 18.5 and 19.7 per cent tin and having a combined recovery of 51.9 per cent of the tin. In addition, 39.5 per cent of the tin remained in the lead concentrate, which assayed 12.6 per cent tin. The larger percentage of tin still remaining in the lead concentrate was due to the necessity of using large amounts of flotation reagents in order to assure a low tailing but, at the same time, this excess of reagents prevented a clean-cut separation of the stannite and galena. Three hundred and six pounds of tin concentrate and 354 pounds of lead concentrate were obtained. The tin concentrates were shipped to Vancouver.

DETAILS OF TESTS:

This investigational work is divided into two parts. Part I gives the results of the small-scale test work, and Part II the ensuing mill run.

PART I. - Small-Scale Tests.

In these tests portions of the ore at minus 14 mesh were ground in a ball mill to pass 85 per cent minus 200 mesh. The grind was made with two parts of ore to one of water. The pulp was than transferred to a Denver flotation cell and a bulk flotation concentrate obtained. This concentrate was cleaned twice in a smaller cell. The cleaner concentrate was then agitated in cyanide solution and a lead concentrate floated. The tailing from this operation was the tin concentrate. In the different tests, flotation reagents were added to the grind and cells in the amounts as noted. From previous test work on other shipments it was shown that a grind of from 85 to 90 per cent minus 200 was necessary in order to free the lead and tin

(Details of Tests, cont'd) -

minerals.

Test No. 1.

In this test the ore was floated in a 2,000-gram flotation cell and the resulting concentrate cleaned in a 1,000-gram machine. The cleaner concentrate was agitated in cyanide in a 500-gram machine prior to flotation.

Reagents Added -

To Grind:		Lb./ton ore
Soda ash	-	6.0
NaCN	-	0.3

To Flotation:

Pine oil	-	0.15
Cresylic acid	-	0.13
Amyl xanthate	-	0.30

To Agitation of Cleaned Concentrate:

NaCN - 1.0 lb./ton ore.

The pH of the bulk flotation was 8.4.

The cleaned flotation concentrate was agitated in cyanide for 5 minutes prior to selective flotation.

Results of Selective Flotation:

Product	Weight, per cent	ASSAYS: Ag, oz./ton	Cu, Per Cent	Pb, Per Cent	Sn, Per Cent	Zn, Per Cent	Ratio of concentration
Feed	100.00	24.18*	3.56*	3.53*	3.13*	7.68*	
Lead conc.	9.78	105.9	15.50	25.35	12.52	9.49	10.2:1.
Tin conc.	12.09	74.8	14.10	4.12	13.36	13.54	8.3:1.
Middling	17.51	18.3	1.94	1.65	1.07	16.12	
Tailing	60.62	2.6	0.01	0.44	0.18	3.79	
		DISTRIBUTION, Per Cent: Ag	Cu	Pb	Sn	Zn	
Feed	100.00	100.0	100.0	100.0	100.0	100.0	
Lead conc.	9.78	33.7	42.5	70.2	22.0	5.0	10.2:1.
Tin conc.	12.09	32.6	47.8	14.1	56.9	11.9	8.3:1.
Middling	17.51	25.7	9.5	8.2	16.2	39.0	
Tailing	60.62	8.0	0.2	7.5	4.9	44.1	

* Calculated.

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(Details of Tests, cont'd) -

Test No. 2.

In this test the cleaned concentrate was agitated in a 1,000-gram cell prior to selective flotation. It was thought, from the results of Test No. 1, that the 500-gram cell used for agitation was too small to permit the cyanide solution to depress the stannite, on account of the large amount of accumulated reagents.

Reagents Added -

To Grind:

		Lb./ton ore
Soda ash	-	0.7
Cyanide	-	0.3

To Flotation:

Pine oil	-	0.15
Cresylic acid	-	0.13
Amyl xanthate	-	0.35

To Agitation of Cleaned Concentrate:

NaCN - 1.0 lb./ton ore.

The pH of the bulk flotation was 8.6 and the cleaner flotation concentrate was agitated for 5 minutes prior to flotation of the lead concentrate.

Product	Weight, per cent	ASSAYS Ag, oz./ton	Per Cent Cu	Pb	Sn	Zn	Ratio of concentration
Feed	100.00	24.47•	3.67•	3.63•	3.24•	7.73•	
Lead conc.	6.87	120.0	11.74	41.84	10.38	5.66	14.5:1.
Tin conc.	8.26	96.6	24.30	0.15	22.30	11.11	12.1:1.
Middling	19.52	32.2	3.66	2.27	2.68	15.46	
Tailing	65.35	3.0	0.01	0.46	0.24	5.22	

	DISTRIBUTION Per Cent Ag	Cu	Pb	Sn	Zn	
Feed	100.00	100.0	100.0	100.0	100.0	100.0	
Lead conc.	6.87	42.8	21.9	79.2	39.0	12.1	14.5:1.
Tin conc.	8.26	37.4	54.6	0.3	51.5	21.3	12.1:1.
Middling	19.52	13.3	19.4	12.2	6.0	36.7	
Tailing	65.35	6.5	4.1	8.3	3.5	29.9	

• Calculated.

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(Details of Tests, cont'd) -

Test No. 3.

In an endeavour to obtain a more selective separation between the different minerals, the amount of cyanide added, to both the ball mill and the final flotation, was increased. The time of agitation prior to the selective flotation was also lengthened. Conditions otherwise were similar to previous tests.

Reagents Added -

To Ball Mill: Lb./ton ore

Soda ash	-	7.0
NaCN	-	0.4

To Flotation:

Pine oil	-	0.15
Cresylic acid	-	0.13
Amyl xanthate	-	0.35

To Agitation of Cleaner Concentrate:

NaCN - 1.5 lb./ton ore.

The pH of the bulk flotation was 8.6. The cleaned flotation concentrate was agitated for 8 minutes prior to flotation of the lead concentrate.

Product	Weight per cent	ASSAYS					Ratio of concentration
		oz./ton	T Per Cent Cu	Pb	Sn	Zn	
Feed	100.00	24.66•	3.43•	3.82•	3.32•	7.59•	
Lead conc.	5.73	118.9	6.86	49.73	7.31	4.19	17.5:1.
Tin conc.	10.47	98.7	23.03	1.42	21.83	6.05	9.5:1.
Middling	17.11	31.4	3.10	2.43	2.60	15.48	
Tailing	66.69	3.2	0.14	0.61	0.26	6.10	
		DISTRIBUTION, Per Cent					
		Ag	Cu	Pb	Sn	Zn	
Feed	100.00	100.0	100.0	100.0	100.0	100.0	
Lead conc.	5.73	27.6	11.5	74.6	12.6	3.1	17.5:1.
Tin conc.	10.47	41.9	70.3	3.9	68.8	8.3	9.5:1.
Middling	17.11	21.8	15.5	10.9	13.4	35.0	
Tailing	66.69	8.7	2.7	10.6	5.2	53.6	

• Calculated.

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(Details of Tests, cont'd) -

Test No. 3, cont'd. -

From the above results of this test it is apparent that an increase in the amount of cyanide added to the ball mill succeeded in depressing the spalerite, 53.6 per cent of this mineral reporting in the tailing. The addition of 1.5 pounds of cyanide to the selective flotation of the galena and stannite minerals produced a fairly sharp separation.

This test concluded the small-scale work on the shipment, and the ensuing mill run was based on the results obtained.

PART II. - Mill Run.

The flow-sheet used for this concentration was as follows:

The ore at minus 3/8-inch size was fed to a 30" x 48" ball mill at a feed rate of 350 pounds per hour; 800 pounds of steel balls had been added to the mill. The mill discharge, which had a density of 70 per cent solids, was conveyed by means of a bucket elevator to a Dorr classifier. The classifier return went back to the ball mill and the classifier overflow was pumped to a conditioning tank. The conditioned pulp was concentrated by flotation in ten Denver No. 7 cells as follows: A rougher flotation concentrate was first obtained, using 4 cells. The rougher tailing was then further concentrated by three additional scavenger cells. The scavenger tailing was discarded. The combined rougher and scavenger concentrates were then cleaned in two cleaner cells and the concentrates from this operation were again cleaned in a final cell. This cleaned concentrate was then pumped to a conditioning tank and agitated in cyanide solution prior to selective flotation. The pulp was

(Details of Tests, cont'd) -

then floated in two cells and the cleaned lead concentrate again floated in a final cell, where the finished lead concentrate was obtained. The tailing from these selective flotations was the tin concentrate.

The following reagents were added during the run:

		Lb./ton ore
To the Ball Mill:		
Soda ash	-	7.0
KCN	-	0.4
To No. 1 Conditioner:		
Amyl xanthate	-	0.15
To Rougher Cells:		
Pine oil	-	0.10
Cresylic acid	-	0.09
Amyl xanthate	-	0.12
To Scavenger Cells:		
Pine oil	-	0.05
Cresylic acid	-	0.04
Amyl xanthate	-	0.06
To No. 2 Conditioner:		
NaCN	-	0.7
To Cleaned Concentrate Pump:		
NaCN	-	0.4
To Selective Flotation Pump:		
NaCN	-	0.4

The ball mill discharge was 70 per cent solids; the classifier overflow and flotation feed was 29 per cent solids; and the flotation tailing was 18 per cent solids.

The pH of the flotation feed was 9.0. The fineness of grinding was 85.3 per cent minus 200 mesh.

A screen test on the classifier overflow resulted as follows:

(Continued on next page)

(Details of Tests, cont'd) -

Mesh		Weight, per cent
+100	-	0.1
-100+150	-	4.4
-150+200	-	10.2
-200	-	85.3

The mill run results were as follows:

Product	Weight, per cent	ASSAYS Per Cent Ag, oz./ton	Cu	Pb	Sn	Zn	Ratio of concentration
Feed	100.00	24.66	3.46	3.50	3.40	7.82	
Lead conc.	9.62	113.43	13.17	28.76	12.60	6.16	10.4:1.
#1 tin conc.	4.58	88.16	19.62	1.49	18.60	12.39	21.8:1.
#2 tin conc.	3.75	102.43	20.92	5.77	19.71	10.25	26.7:1.
Tailing	82.05	5.96	0.40	0.67	0.32	8.73	

	DISTRIBUTION, Per Cent Ag	Cu	Pb	Sn	Zn	
Feed	100.00	100.0	100.0	100.0	100.0	100.0	
Lead conc.	9.62	46.1	38.7	76.8	39.5	6.8	10.4:1.
#1 Tin conc.	4.58	17.1	27.4	1.9	27.8	6.5	21.8:1.
#2 tin conc.	3.75	16.2	23.9	6.0	24.1	4.4	26.7:1.
Tailing	82.05	20.6	10.0	15.3	8.6	82.3	

The bulk flotation concentrate, weighing 660 pounds, was agitated in cyanide and floated. From this procedure a No. 1 tin concentrate weighing 168 pounds was obtained. The companion lead concentrate was again agitated in cyanide and floated. From this second separation a tin concentrate weighing 138 ponnds resulted, leaving a lead concentrate weighing 354 pounds.

In the above calculation of weights and assays, the weights per cent and subsequent distributions per cent were figured from the assay of the flotation tailings and concentrates in conjunction with the recorded head assay and weights of the recovered concentrates. Owing to the comparatively large amount of pulp remaining in the ball mill, classifier,

(Details of Tests, cont'd) -

flotation cells, etc., on the conclusion of the run, a compilation which would include the amount of feed to the ball mill would not be correct.

A panorama of the flow-sheet used in this mill run would be as follows:

BALL MILL
↓
Return to Ball Mill. ← CLASSIFIER → Overflow to No. 1 Conditioner.
↓
NO. 1 CONDITIONER
↓
Concentrates to No. 2 Conditioner. ← FLOTATION CELLS → Tailing to Waste.
↓
NO. 2 CONDITIONER
↓
Concentrate = Lead Concentrate. ← FLOTATION CELLS → Tailing = Tin Concentrate.

Summary

SUMMARY AND CONCLUSIONS:

In the small-scale test work on the shipment, a lead concentrate was obtained, assaying 118.9 ounces silver per ton, 6.86 per cent copper, 49.7 per cent lead, 7.31 per cent tin and 4.19 per cent zinc, giving a recovery of 74.6 per cent of the lead at a 17.5:1 ratio of concentration. A tin concentrate was also obtained, assaying 98.7 ounces silver per ton, 23.0 per cent copper, 1.42 per cent lead, 21.8 per cent tin and 6.0 per cent zinc, with a recovery of 68.8 per cent of the tin at a 9.5:1 ratio of concentration. If the lead and tin in the middling product were added, these recoveries would be 85.5 per cent of the lead and 82.2 per cent of the tin. The grind was 85 per cent minus 200 mesh.

In the mill run, a lead concentrate, weight 354 pounds,

(Summary and Conclusions, cont'd) -

assaying 113.4 ounces silver per ton, 13.2 per cent copper, 28.7 per cent lead, 12.6 per cent tin and 6.2 per cent zinc was obtained at a 10.4:1 ratio of concentration, with a 76.8 per cent recovery of the lead. Two tin concentrates were also obtained; No. 1, assaying 88.1 ounces silver per ton, 19.6 per cent copper, 1.15 per cent lead, 18.6 per cent tin and 12.4 per cent zinc, with a ratio of concentration of 21.8:1; and No. 2 concentrate, assaying 102.4 ounces silver per ton, 20.9 per cent copper, 5.8 per cent lead, 19.7 per cent tin and 10.2 per cent zinc at a 26.7:1 ratio of concentration. The combined recoveries of these two tin concentrates was 51.9 per cent of the tin. Tin Concentrate No. 1 weighed 168 pounds and Tin Concentrate No. 2, 138 pounds. These weights and recoveries represented the results obtained from the amount of ore fed to the ball mill minus the amount of ore remaining in the mill circuit after the run. The grind was 85.3 per cent minus 200.

The results of the mill run show that some 39.5 per cent of the tin still remains in the lead concentrate after two stages of agitation in cyanide solution and subsequent flotation. This condition appeared to be due to the large amount of reagents necessarily contained in the primary bulk concentrate, these reagents coating the stannite and adhering galena minerals and preventing the depression of the stannite by agitation in cyanide. In order to correct this condition, this bulk concentrate would have to have these excess reagents removed by filtration plus hot water washing; steaming, or some other method prior to selective flotation.

The present mill run shows that while the flow-sheet as given on Page 10 is applicable to this type of ore, it is

(Summary and Conclusions, cont'd) -

apparent that some additional equipment will be necessary in order to obtain the desired separation of the lead and tin minerals.

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