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MINES BRANCH INVESTIGATION REPORT IR 61-142

INVESTIGATION OF A GOLD-ANTIMONY ORE FROM THE ACE PROPERTY OF BRALORNE PIONEER MINES LIMITED, BRALORNE, B. C.

by

T. F. BERRY

MINERAL PROCESSING DIVISION

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INVESTIGATION OF A GOLD-ANTIMONY ORE FROM THE ACE PROPERTY OF BRALORNE PIONEER MINES LIMITED, BRALORNE, B.C.

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SUMMARY OF RESULTS

The ore under investigation was not amenable to treatment either by straight amalgamation or straight cyanidation.

In Test 15, 90.20% of the gold was recovered in a bulked sulphide flotation concentrate. The removal of a separate stibnite concentrate by differential flotation assaying 30.50% Sb and representing 71.4% of the Sb resulted in a loss of 13.5% of the gold leaving a roaster feed containing 76.7% of the total gold in the ore.

In Test 20, the roasting of the concentrate produced in Test 15 and the subsequent cyanidation of the calcine resulted in an extraction of 87.5% of the gold in the calcine which represented only 67.1% of the gold in the ore.

In several tests salt was added to the roaster feed in an attempt to overcome the detrimental effect of Sb on the cyanidation of gold.

No advantage accrued from the addition of salt to roaster feeds from which the Sb had been largely removed by flotation.

In the treatment of flotation concentrates containing a large percentage of Sb, the addition of salt prior to roasting gave a greatly increased gold extraction from the resulting calcines, over that obtained from calcines produced without the addition of salt.

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

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INTRODUCTION

In a letter dated April 4, 1960, Mr. J.P. Weeks, Chief Geologist of Bralorne Pioneer Mines Limited, 355 Burrard St., Vancouver B.C., asked the Mines Branch in Ottawa to carry out an investigation on a gold ore containing appreciable percentages of arsenopyrite and stibnite.

Location of Property

The property from which the sample was taken is a gold-antimony prospect under option to the company, located in the Bridge River valley of British Columbia.

Shipment

A shipment comprising 7 sacks of lump ore from the "Ace Property" of Bralorne Pioneer Mines Limited weighing an aggregate of 475 lb was received by the Extraction Metallurgy Division on December 16, 1960.

Nature of Investigation Requested

Since the extraction problems presented by this ore were more of an ore dressing nature in that flotation, calcination and cyanidation were involved, the shipment was turned over to the Mineral Processing Division on March 16, 1961.

While the primary purpose of the investigation concerned the extraction of the gold from the ore, the company asked the Mines Branch to investigate the possibility of producing a stibnite concentrate containing a minimum of the gold.

Sampling and Analysis*

On receipt of the shipment the Extraction Metallurgy Division selected several representative specimens of ore for a mineralogical investigation. The remainder of the shipment was crushed to -10 M and samples were riffled out for a semi-quantitative spectrographic and a chemical analysis.

The spectrographic analysis showed no elements of economic importance other than gold and antimony.

The results of the chemical analysis on the ore may be seen in the following table.

TABLE 1

Results of Chemical Analysis

Element	
Gold (Au)	0.165 oz/ton
Silver (Ag	0.095 oz/ton
Arsenic (As)	1.68 %
Antimony (Sb)	0.51 %
Iron (Soluble Fe)	7.10 %
Sulphur (Total S)	3.93 %

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*Chemical Analysis by L. Lutes and R.W. Buckmaster, Analytical Chemistry Sub-division, Mineral Sciences Division.

MINERALOGICAL INVESTIGATION

Since a separate Mines Branch Investigation Report* has been issued outlining the mineralogical work done on specimens of the Bralorne ore it will be sufficient as far as this present work is concerned to include only the summary of the results of that report.

Summary of Results

"The rock comprising a gold ore sample from Bralorne Pioneer Mines Ltd., consists of a light coloured fine grained groundmass of quartz, dolomite and sericite cut by veins of quartz and dolomite. Fine grained pyrite and arsenopyrite are disseminated through parts of the groundmass while coarser masses of stibnite occur in the veins. Although the ore contains 0.17 oz Au/ton none was observed microscopically. It is concluded therefore that the gold occurs in sub-microscopic form and is fairly evenly distributed throughout the rock with only minor concentration in pyrite".

DETAILS OF INVESTIGATION

Amalgamation of the Ore

Test 1

A 1000 g sample of the ore was ground to 68.7% -325 M and amalgamated with lime and mercury for 1 hr.

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*Mines Branch Investigation Report IR 61-15 by M.R. Hughson and S.K. Kaiman, Extraction Metallurgy Division, Mines Branch, Ottawa, February 25, 1961.

The amalgam-mercury mixture was recovered and assayed for gold along with the amalgamation residue.

No gold was recovered by amalgamation.

Cyanidation of the Ore

Tests 2, 3, & 4

The results obtained when the ground ore was cyanided are shown in the following self-explanatory table. In each of these tests the pulp at a dilution of 2:1 was cyanided at a solution strength of 0.5 lb NaCN and 0.5 lb CaO/ton.

TABLE 2

Results of Straight Cyanidation of Ore

Test No.	Grind % -325M	Agitation time hr	Reagents Added lb/ton Soln		Consumption lb/ton Ore		RP[*]	Residue oz/ton	Extraction[**] %
			NaCN	CaO	NaCN	CaO		Au	Au
2	38.9	72	0.5	0.5	2.38	9.12	828	0.160	3.04
3	60.3	"	"	"	3.54	8.64	856	0.160	"
4	74.0	"	"	"	4.28	8.76	920	0.160	"

[*] RP - Reducing power - ml N/10 $KMnO_4$/1

[**] Extraction calculated by difference on a head assay of 0.165 oz Au/ton.

Determination of Gold Association in the Ore

Test 5

A 500 g sample of the cyanide residue from Test 4 was treated with 10% HCl until all effervescence had ceased. The pulp was then filtered and thoroughly washed to eliminate the acid and was cyanided for 24 hr at a solution strength of 0.5 lb NaCN and 0.5 lb CaO/ton. This cyanide residue was assayed for gold.

A 3 assay ton sample of the acid leached cyanide residue was accurately weighed and was treated with hot aqua regia until no more dissolution was evident and only an insoluble gangue remained. The aqua regia residue was assayed for gold.

The assays of the residues from the first and second acid treatments were corrected for the weight loss occasioned by the HCl leach.

The results obtained may be seen in the following table.

TABLE 3

Results of Gold Association Test

Product	Assay oz/ton Au	Residue % Au	Extraction % Au
Feed	0.165	(100.0)	
First cyanide residue	0.160	97.0	3.0
Second " " (HCl leach)	0.148	89.7	7.3
Aqua regia "	0.00125	0.8	88.9

Calculated Results	oz/ton Au	Distribution % Au
Exposed gold (cyanide soluble)	0.005	3.0
Gold enclosed in carbonates*	0.012	7.3
" " " sulphides	0.147	89.1
" " " insoluble gangue	0.00125	0.6

*Gold enclosed in hydrochloric acid soluble pyrrhotite was probably negligible.

Flotation

The mineralogical investigation which was done indicated that the sulphides in the ore were very finely disseminated and that this would necessitate fine grinding to affect liberation of the gold.

Three preliminary flotation tests were done in each of which the ore was ground to approximately 75% -325 M. In each test a different reagent combination was used. About seven concentrates were floated and these were assayed for Au, Sb, As and S. The results of these tests were very similar and thus only one of them, Test 6, will be shown in this report. The results of Test 6 which are shown in Table 4-b were plotted in the figure shown on page 10 and indicate the relationship which exists between the gold and the sulphur, arsenic and antimony.

<u>Test 6</u>

A 2000 g sample of the ore was ground to approximately 95.0% -200 M (75.0% -325 M) with 1.0 lb Na_2CO_3 ton added to the grind. The pH of the resulting pulp was 8.7. Eight separate concentrates were floated using staged additions of $CuSO_4$, Aero Xanthate 301 and pine oil. The pulp was conditioned after each reagent addition.

TABLE 4-a

<u>Flotation Scheme Test 6</u>

Concentrate No.	1	2	3	4	5	6	7	8	Total
<u>Reagents lb/ton Feed</u>									
$CuSO_4$	0.3	0.1	0.05	0.05	0.05	0.05	0.05	-	0.65
Aero xanthate 301	0.1	0.05	"	"	"	"	"	-	0.40
Pine oil	0.06	0.02	-	0.02	0.02	-	0.02	-	0.14
<u>Operation Time Min.</u>									
Conditioning	7	2	2	2	2	2	2	-	19
Flotation	2	2	3	3	4	4	4	5	27
pH (after)	-	8.5	8.5	8.4	-	8.4	-	8.3	-

TABLE 4-b

Results of Test 6

Product	Weight %	Assays oz/ton Au	Assays % Sb	As	S	Distribution % Au	Sb	As	S
Flotation concentrate No. 1	1.3	0.52	20.73	4.03	28.91	3.9	51.9	4.6	9.1
" " 2	3.5	0.92	3.98	7.22	30.55	18.8	26.8	22.2	25.8
" " 3	5.7	1.04	1.00	7.26	24.88	34.6	11.0	36.3	34.3
" " 4	3.2	0.69	0.40	4.55	13.19	12.9	2.5	12.8	10.2
" " 5	2.7	0.49	0.26	2.96	8.40	7.7	1.3	7.0	5.5
" " 6	1.9	0.37	0.19	2.20	6.04	4.1	0.7	3.7	2.8
" " 7	4.0	0.20	0.15	1.10	3.13	4.7	1.2	3.9	3.0
" " 8	3.0	0.16	0.05	0.90	2.60	2.8	0.3	2.4	1.9
Flotation tailing	74.7	0.024	0.03	0.11	0.41	10.5	4.3	7.1	7.4
Head (calcd)	100.0	0.171	0.52	1.14	4.14	100.0	100.0	100.0	100.0

The flotation tailing from Test 6 was infrasized and the fractions were assayed for gold. For comparison purposes a fresh sample of ore was given an identical grind to that used in Test 6 and a sample of this pulp was infrasized with the fractions being assayed for gold.

The results of this phase of Test 6 may be seen in Table 4-c.

TABLE 4-c

Results of Infrasizer Test on Feed and Flotation Tailing from Test 6

Size Fractions	Weight % Ground Pulp	Weight % Flotation Tailing	Assays oz/ton Au Ground Pulp	Assays oz/ton Au Flotation Tailing	Distribution % Au Ground Pulp	Distribution % Au Flotation Tailing
+200 M	4.8	9.7	0.09	0.030	2.6	11.1
-200 M + 56 u	4.8	4.8	0.19	0.025	5.5	4.6
- 56 u + 40 "	15.6	15.0	0.14	0.015	13.3	8.6
- 40 " + 28 "	15.6	13.7	0.17	0.015	16.1	7.8
- 28 " + 20 "	13.3	11.6	0.21	0.010	17.0	4.4
- 20 " + 14 "	8.7	8.0	0.235	(0.0125	12.4	(6.6
- 14 " + 10 "	6.8	5.8	0.24		9.9	
- 10 "	30.4	31.4	0.125	0.0475	23.2	56.9
Head (calcd)	100.0	100.0	0.164	0.026	100.0	100.0

In Figure 1 in which the results of Test 6 are plotted, it may be seen that there exists a close relationship between the gold arsenic and sulphur. The same close relationship between the gold and the antimony is not as apparent.

Test 7

Using the same grind and flotation scheme as in Test 6 a rougher flotation concentrate was obtained and cleaned twice. The object was to determine the grade of concentrate and the recovery of gold and antimony which might be obtained. No attempt was made to float a separate stibnite concentrate. The results of this test may be seen in Table 5.

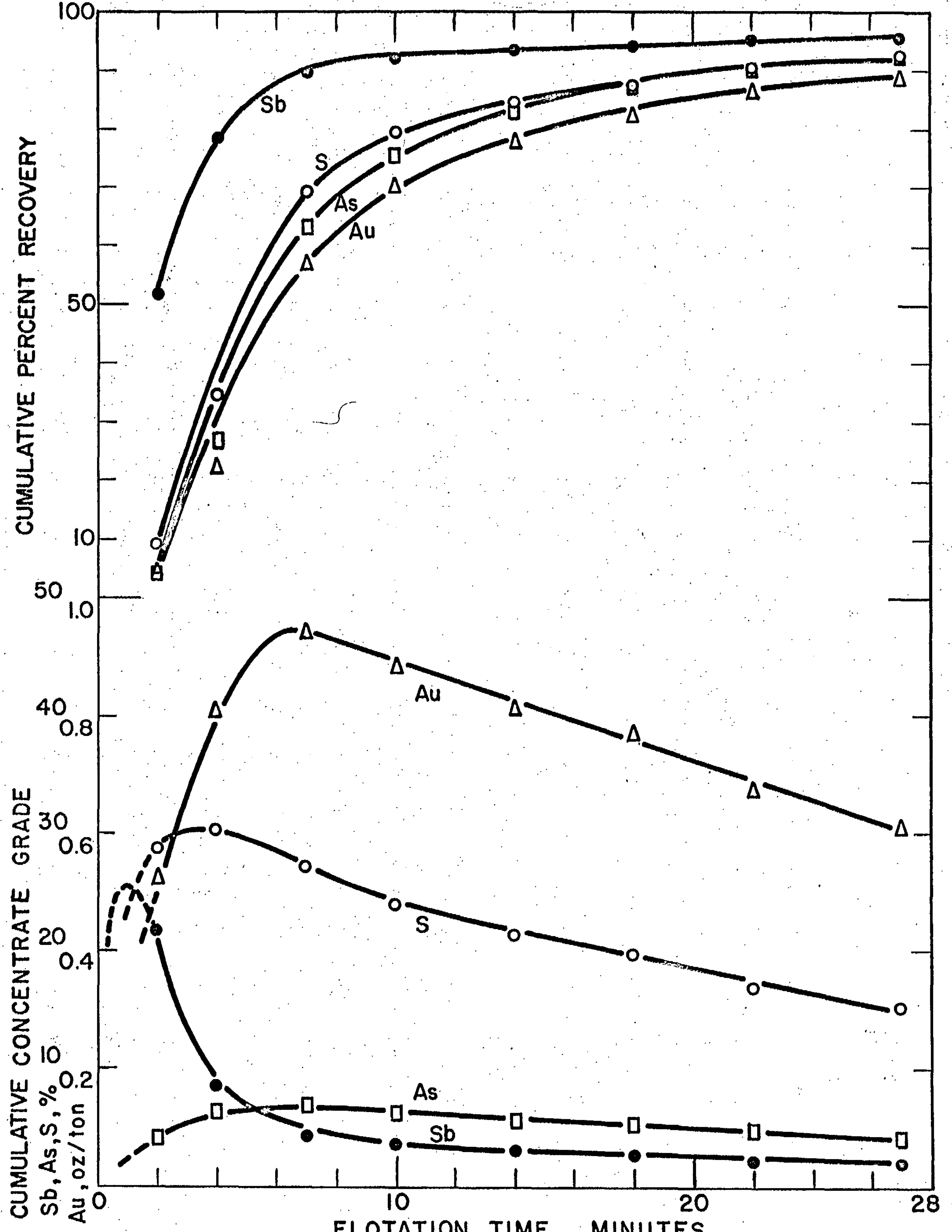

Figure 1 - Relationship between Gold and Arsenic Antimony and Sulphur

TABLE 5

Results of Test 7

Product	Weight %	Assays oz/ton	Assays %			Distribution %			
		Au	Sb	As	S	Au	Sb	As	S
Sulphide recl concentrate	9.9	1.04	5.35	7.87	28.50	60.5	94.5	67.3	73.2
Sulphide recl tailing	3.8	0.61	0.20	3.90	10.16	13.6	1.4	12.8	10.0
Sulphide cl tailing	12.0	0.215	0.07	1.24	3.34	15.1	1.5	12.8	10.4
Flotation tailing	74.3	0.025	0.02	0.11	0.33	10.8	2.6	7.1	6.4
Head (calcd)	100.0	0.171	0.56	1.16	3.85	100.0	100.0	100.0	100.0

Test 8

In this test 2,- 2000 g samples were floated using the same flotation conditions as prevailed in Test 6 with the exception that in stage 8, 0.1 lb $CuSO_4$, 0.1 lb Aero Xanthate 301 and 0.02 lb pine oil were added. The rougher concentrates were combined and an attempt was made to remove a stibnite concentrate. The results of this test were quite unsatisfactory and will not be outlined in this report. The test is important however because the flotation tailing was cyanided in an attempt to recover additional gold. The flotation tailing assayed 0.04 oz Au/ton. Five samples of the flotation tailing weighing 500 g each were riffled out and cyanided. The following results were obtained.

TABLE 6

Results of Cyanidation of Flotation Tailing from Test 8

Test No.	Dilution Ratio	Agitation Time hr	Reagents Added lb/ton Soln		Consumption lb/ton Ore		RP*	Residue oz/ton Au	Extraction** % Au
			NaCN	CaO	NaCN	CaO			
8-A	2:1	24	0.5	0.5	0.60	7.32	144	0.032	18.0
8-B	2:1	48	"	"	0.76	7.40	108	0.028	28.2
8-C	4:1	24	"	"	0.56	7.20	60	0.032	18.0
8-D	"	48	"	"	0.96	8.48	64	0.028	28.2
8-E	"	"	1.0	1.0	1.00	11.52	52	0.030	23.1

* RP - Reducing power, ml N/10 $KMnO_4$/1.
**Extraction calculated by difference.

Test 9

Four 2000 g samples of ore were ground separately to approximately 95.0% -200 M and flotation proceeded using the same reagents as in Test 8. However a stibnite concentrate was floated first from each 2000 g sample prior to the flotation of a bulk sulphide concentrate. The flotation time for the stibnite was 3 min. The four stibnite concentrates were combined and cleaned twice. The four bulk sulphide concentrates were combined and cleaned three times. The results of this test may be seen in Table 7.

TABLE 7

Results of Test 9

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Stibnite cl conc	1.7	0.925	26.22	6.65	33.80	9.3	78.9	10.0	14.5
" " tailing	1.5	0.765	1.14	5.07	21.48	6.8	3.0	6.7	8.1
Final sulphide conc	7.7	1.06	1.00	8.02	26.43	48.1	13.6	54.9	51.4
3rd cl tailing	1.3	0.60	0.23	4.08	10.76	4.6	0.5	4.7	3.5
2nd " "	2.1	0.47	0.10	2.83	8.34	5.8	0.4	5.3	4.4
1st " "	8.3	0.24	0.05	1.47	4.36	11.7	0.8	10.8	9.2
Flotation "	77.4	0.03	0.02	0.11	0.45	13.7	2.8	7.6	8.9
Head (calcd)	100.0	0.17	0.56	1.13	3.96	100.0	100.0	100.0	100.0

Test 10

In this test a new flotation scheme was used. The following reagents were added to the grind;

Na_2CO_3 - 2.0 lb/ton ore
$PbNO_3$ - 0.8 " " "
Pot. amyl xanthate - 0.1 " " "
Pot. ethyl xanthate - 0.4 " " "
Pine oil/cresylic acid -50/50 - 0.1 " " "

The ore was ground to approximately 95.0% -200 M and floated for 4.5 min. This concentrate was cleaned once for 3 min. A second or scavenger concentrate was floated for 6 min. No additional reagents were added during the flotation of this concentrate. The results may be seen in the following table.

TABLE 8

Results of Test 10

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Bulk sulphide cl concentrate	10.5	1.05	4.77	8.05	29.50	66.8	92.2	69.9	80.2
Bulk sulphide cl tailing	3.7	0.47	0.12	3.19	5.71	10.5	0.8	9.8	5.5
Scavenger concentrate	4.6	0.18	0.12	1.29	3.74	5.0	1.0	4.9	4.4
Flotation tailing	81.2	0.036	0.04	0.23	2.92	17.7	6.0	15.4	9.9
Head (calcd)	100.0	0.165	0.54	1.21	3.86	100.0	100.0	100.0	100.0

Test 11

This test differed from Test 10 in that the flotation time of the bulk sulphide concentrate was increased to 7 min. This concentrate was cleaned twice. The flotation time of the scavenger concentrate was increased to 15 min with the following reagents being added to a conditioning stage:

$CuSO_4$ - 0.3 lb/ton ore

Pot. amyl xanthate - 0.1 " " "

Pine oil/cresylic acid - 50/50 - 0.05 " " "

The results were as follows:

TABLE 9

Results of Test 11

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Bulk sulphide recl concentrate	9.9	1.13	4.65	8.51	31.42	66.1	94.0	67.8	80.1
Bulk sulphide recl tailing	2.4	0.66	0.12	4.56	8.76	9.3	0.6	8.8	5.4
Bulk sulphide cl tailing	3.4	0.22	0.12	1.33	2.75	4.4	0.8	3.6	2.4
Scavenger concentrate	8.5	0.20	0.86	1.55	3.09	10.2	1.5	10.6	6.7
Flotation tailing	75.8	0.0225	0.02	0.15	0.27	10.0	3.1	9.2	5.4
Head (calcd)	100.0	0.17	0.49	1.24	3.88	100.0	100.0	100.0	100.0

The amount of gold recovered in this scavenger concentrate indicated that a much longer bulk sulphide flotation time would be advantageous.

Test 12

This test was identical to Test 11 except that the flotation time for the bulk sulphide concentrate was increased to 20 min. The results were as follows:

TABLE 10

Results of Test 12

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Bulk sulphide recl concentrate	12.1	0.905	3.70	6.91	27.62	65.3	93.0	69.1	86.8
Bulk sulphide recl tailing	1.8	0.77	0.28	5.00	4.28	8.3	1.0	7.4	2.0
Bulk sulphide cl tailing	6.6	0.275	0.12	1.86	1.92	10.8	1.6	10.2	3.2
Scavenger concentrate	6.9	0.17	0.09	1.17	2.21	7.0	1.3	6.7	4.0
Flotation tailing	72.6	0.02	0.02	0.11	0.21	8.6	3.1	6.6	4.0
Head (calcd)	100.0	0.17	0.48	1.21	3.85	100.0	100.0	100.0	100.0

Test 13

This was a duplicate of Test 12 with two exceptions. The grind was considerably coarser being approximately 80.0% -200 M and the bulk sulphide concentrate was not cleaned. The results may be summarized as follows:

TABLE 11

Results of Test 13

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Bulk sulphide conc	22.9	0.58	2.03	4.33	15.00	80.0	95.7	83.3	90.4
Scavenger conc	7.5	0.22	0.09	1.44	2.53	9.9	1.4	9.1	5.0
Flotation tailing	69.6	0.024	0.02	0.13	0.25	10.1	2.9	7.6	4.6
Head (calcd)	100.0	0.17	0.48	1.19	3.80	100.0	100.0	100.0	100.0

This grind may have been a bit too coarse when one compares the overall recovery of gold of 89.9% with that obtained in Test 12 of 91.4%.

Test 14

Locked cycle bulk sulphide flotation

It was necessary to obtain a large amount of a bulk sulphide concentrate to provide sufficient material for the stibnite separation phase of the investigation and to determine the amenability of the arsenopyrite-pyrite concentrate to roasting and cyanidation. The flotation test was done in locked cycle to determine whether prolonged recycling of the middling fractions (cleaner tailing and scavenger concentrate) would result in appreciable gold losses in the flotation tailing.

A 20,000 g sample of -10 M ore was ground in lots of 2000 g to approximately 85.0% -200 M. The following flotation scheme was used.

Added to grind

Na_2CO_3	- 2.0 lb/ton ore
$PbNO_3$	- 0.8 " " "
Aero Xanthate 350	- 0.1 " " "
" " 303	- 0.4 " " "
Pine oil/Cresylic acid - 50/50	- 0.1 " " "

pH of ground pulp - 9.2

Bulk sulphide flotation time - 20 min

Cleaner stage " " - 12 min no reagents added

A scavenger concentrate was floated after conditioning the pulp for 5 min with the following reagents

$CuSO_4$ - 0.3 lb/ton ore

Aero Xanthate 350 - 0.1 " " "

Pine oil - 0.04 " " "

Scavenger flotation time - 15 min

The sulphide cleaner tailing and the scavenger concentrate were returned to the head of each succeeding rougher flotation cycle. Table 12 shows the results which were obtained.

TABLE 12

Results of Locked Cycle Bulk Sulphide Flotation Test 14

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Bulk sulphide cl concentrate No. 1	1.4	0.860	3.16	6.69	22.66	7.2	8.8	7.7	8.4
" " 2	1.9	0.800	3.16	6.16	19.04	9.1	11.8	9.6	9.6
" " 3	1.8	0.790	2.44	6.08	18.30	8.5	8.1	9.0	8.7
" " 4	1.8	0.820	3.16	6.39	18.83	8.8	11.3	9.5	9.0
" " 5	1.9	0.780	2.21	6.00	17.58	9.0	8.3	9.4	8.8
" " 6	1.9	0.780	2.86	6.00	19.84	8.8	10.8	9.4	10.0
" " 7	1.8	0.780	3.22	6.00	20.56	8.8	12.2	9.4	10.3
" " 8	1.9	0.800	2.98	5.93	19.41	9.1	11.2	9.3	9.7
" " 9	1.8	0.825	1.97	5.93	19.61	8.9	7.0	8.8	9.3
" " 10	1.9	0.805	2.21	6.00	19.29	9.1	8.3	9.4	9.7
Bulk sulphide cl tailing	0.4	0.1000	0.03	0.65	1.13	0.2	0.02	0.2	0.1
Scavenger concentrate	0.9	0.2000	0.03	1.44	2.90	1.1	0.05	1.1	0.7
Flotn tailing No. 1	7.8	0.0260	0.02	0.15	0.28	1.2	0.3	1.0	0.6
" " " 2	7.9	0.0225	0.015	0.11	0.27	1.1	0.2	0.7	0.5
" " " 3	8.1	0.0250	0.015	0.11	0.27	1.2	0.2	0.7	0.5
" " " 4	8.2	0.0240	0.015	0.11	0.27	1.2	0.2	0.7	0.5
" " " 5	8.2	0.0225	0.015	0.11	0.27	1.1	0.2	0.7	0.6
" " " 6	7.9	0.0225	0.015	0.11	0.27	1.1	0.2	0.7	0.6
" " " 7	8.0	0.0225	0.015	0.10	0.27	1.1	0.2	0.6	0.6
" " " 8	8.2	0.0225	0.012	0.10	0.27	1.1	0.2	0.7	0.6
" " " 9	8.1	0.0240	0.012	0.10	0.27	1.2	0.2	0.7	0.6
" " " 10	8.2	0.0225	0.012	0.10	0.37	1.1	0.2	0.7	0.6
Head (calcd)	100.0	0.167	0.50	1.21	3.79	100.0	100.0	100.0	100.0

The results of Test 14 which are shown in Table 12 are summarized for simplification in Table 13. The sulphide cleaner tailing and the scavenger concentrate obtained from the 10th cycle of the locked cycle test were combined.

TABLE 13

Summary of Locked Cycle Bulk Sulphide Flotation Test 14

Product	Weight %	Assays oz/ton	Assays %			Distribution %			
		Au	Sb	As	S	Au	Sb	As	S
Bulk sulphide cl concentrate	18.1	0.803	2.71	6.10	19.43	87.3	97.5	91.4	93.4
Scavenger concentrate + cl tailing from 10th cycle	1.3	0.170	0.03	1.20	2.36	1.3	0.1	1.3	0.8
Flotation tailing	80.6	0.0235	0.015	0.11	0.27	11.4	2.4	7.3	5.8
Head (calcd)	100.0	0.167	0.50	1.21	3.76	100.0	100.0	100.0	100.0

Test 14 (cont'd)

Differential Flotation

The ten cleaner concentrates produced in the bulk sulphide flotation were thoroughly mixed and split into four nearly equal fractions for ease of handling. Each fraction was conditioned for 2 hr at 50% solids with the following reagents.

NaCN - 5 lb/ton concentrate
NaOH - 2.5 " "

Flotation of the stibnite was carried out for 10 min at an average pH of 9.5 with no other reagents being added.

The stibnite concentrate floated from the four fractions was combined and conditioned a further 0.5 hr with 0.5 lb NaOH/ton of stibnite concentrate. The concentrate was cleaned for 7 min at a pH of 9.4 and the cleaner concentrate recleaned for 4 min. No additional reagents were added during the two cleaner stages.

The solution from the rougher flotation was retained for a gold assay to determine whether the long conditioning with cyanide would result in an appreciable gold loss. Table 14 shows the results which were obtained.

TABLE 14

Results of Differential Flotation Test 14

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Stibnite recl conc	1.8	0.26	30.03	1.67	20.60	0.6	22.7	0.5	1.5
" " tail	1.4	0.43	22.09	3.04	17.80	0.8	13.0	0.7	1.0
" cl "	2.3	0.575	18.86	4.10	18.90	1.7	18.3	1.6	1.7
Flotation "	94.5	0.80	1.16	6.08	25.53	96.9	46.0	97.2	95.8
Head (calcd)	100.0	0.78	2.38	5.91	25.16	100.0	100.0	100.0	100.0

After the 2 hr conditioning stage the solution assayed 0.00184 oz Au/ton. This represented 0.08% of the original gold in the bulk sulphide concentrate.

In Table 15 which follows, the results of Test 14 have been calculated in one metallurgical balance. As in Table 13 the sulphide cleaner tailing and the scavenger concentrate, obtained from the 10th cycle of the locked cycle test, were combined. The bulk sulphide concentrate from which the stibnite was recovered is referred to as an arsenopyrite-pyrite concentrate.

TABLE 15

Summary of Results of Test 14

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Stibnite recl conc	0.3	0.26	30.03	1.67	20.60	0.5	20.4	0.4	1.3
" " tail	0.3	0.43	22.09	3.04	17.80	0.8	15.0	0.8	1.1
" cl "	0.4	0.575	18.86	4.10	18.90	1.4	17.0	1.4	1.6
Arsenopyrite-pyrite concentrate	17.1	0.80	1.16	6.08	25.53	84.3	44.8	88.5	90.9
Cl tailing + scavenger concentrate from 10th cycle	1.3	0.17	0.03	1.20	2.36	1.4	0.1	1.3	0.6
Flotation tailing	80.6	0.0235	0.015	0.11	0.27	11.6	2.7	7.6	4.5
Head (calcd)	100.0	1.62	0.44	1.17	4.80	100.0	100.0	100.0	100.0

Test 15

Locked cycle bulk sulphide flotation

A 10,000 g sample of -10 M ore was ground in lots of 2000 g each to 95.2% -200 M. The following reagents were added to the grind:

Na_2CO_3 - 2.0 lb/ton ore
$PbNO_3$ - 0.8 " "
Potassium amylxanthate - 0.2 " "
" ethyl " - 0.3 " "
Pine oil/Cresylic acid 50/50 - 0.1 " "
pH of ground pulp - 9.4
Bulk sulphide flotation time - 20 min

Each concentrate was cleaned once for 12 min with no reagents added. The cleaner tailings were returned to each succeeding cycle.

The rougher tailing was then conditioned for 5 min with the following reagents;

$CuSO_4$	-	0.3 lb/ton ore
Potassium amyl xanthate	-	0.1 " "
Pine oil	-	0.04 " "

and a scavenger concentrate was floated for 15 min and also returned to the next cycle.

Test 15 (cont'd)

Differential Flotation

The five cleaner concentrates were combined and ground for 20 min with the following reagents:

NaCN	-	5.0 lb/ton concentrate
NaOH	-	2.5 " "

The ground pulp was conditioned for 2 hr and stibnite flotation proceeded for 8 min with no additional reagents being added. The pH of the pulp was 11.5. The stibnite concentrate was conditioned for 0.5 hr at a pH of 11.3 obtained by adding 0.5 lb NaOH/ton of original concentrate.

The first stibnite cleaner flotation time was 4 min, the second was 3 min and a final cleaning stage took 3 min. No additional reagents were added during these cleaning stages. The following table summarizes the results which were obtained in this test. As in Table 15 the bulk sulphide concentrate from which the stibnite was removed is referred to as an arsenopyrite-pyrite concentrate.

TABLE 16

<u>Results of Test 15</u>

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Final stibnite concentrate	1.2	0.335	30.50	2.30	30.94	2.4	71.4	2.3	9.5
3rd stibnite cl tailing	0.4	0.570	8.29	4.67	34.83	1.3	6.5	1.6	3.6
2nd " "	0.5	0.840	2.56	6.43	31.32	2.5	2.5	2.7	4.0
1st " "	1.6	0.775	1.71	5.64	28.28	7.3	5.3	7.5	11.6
Arsenopyrite-pyrite concentrate	17.9	0.730	0.31	5.21	14.52	76.7	10.8	78.1	66.5
Sulphide cl tailing (5th cycle)	2.3	0.115	0.09	0.84	1.51	1.6	0.4	1.6	0.9
Scavenger concentrate (5th cycle)	2.3	0.100	0.09	0.65	1.16	1.4	0.4	1.2	0.7
Flotn tailing No. 1	13.4	0.0175	0.024	0.091	0.18	1.4	0.6	1.0	0.6
" " " 2	16.5	0.0175	0.018	0.084	0.18	1.6	0.6	1.2	0.7
" " " 3	13.6	0.0140	0.018	0.080	0.17	1.1	0.5	0.9	0.6
" " " 4	15.2	0.0160	0.018	0.077	0.17	1.4	0.5	1.0	0.7
" " " 5	15.1	0.0150	0.018	0.069	0.17	1.3	0.5	0.9	0.6
Head (calcd)	100.0	0.170	0.51	1.20	3.91	100.0	100.0	100.0	100.0

Calcination and Cyanidation

In all of the roasting tests which follow the feed to the furnace was the arsenopyrite-pyrite concentrate from Tests 14 and 15. In each test the charge was placed in a flat roasting dish in an electric muffle furnace, the temperature of which was controlled within narrow limits by an indicating pyrometer with a thermocouple approximately 3 in. above the surface of the charge. An exhaust fan connected to the furnace provided an excess of air for the oxidation of the arsenic and the antimony and insured the elimination of the volatile gases of these elements from the roasting area.

Test 16

The charge in this test was a 758 g sample of the arsenopyrite-pyrite concentrate from Test 14. The roasting conditions and the observations were as follows:

TABLE 17

Results of Roasting Test 16

Elapsed time min	Temp °C	Observations
0	175	Charge in - no fan - door closed
15	240	No fan - door open ½ in. - rabbling - no SO_2 odour
30	280	" " " " " "
45	340	Fan on - door open 3 in. - rabbling - no SO_2 odour
60	370	Fan on - door open 3 in. - rabbling - slight SO_2 odour
75	370	Slight odour arsenic
90	380	Slight odour arsenic no visible fumes
100	410	" " " " " "
125	430	Heavy fumes - door open 6 in.-rabbling
195	440	No visible fumes
210	500	Door open 10 in.-rabbling
240	600	Hold for 30 min
270		Pull charge

The loss in weight during this roasting test was 17.5%.

The entire calcine was ground to 98.2% -200 M (85.8% -325 M) and a sample was riffled out for a chemical analysis the results of which may be seen in the following table with an analysis of the roaster feed.

TABLE 18

Results of Analysis of Roaster Feed and Calcine

Element		Roaster Feed	Calcine
Gold (Au) oz/ton		0.80	1.00
Antimony (Sb)	%	1.16	0.86
Arsenic (As)	%	6.08	1.52
Sulphur (Total S)	%	25.53	1.20
" (SO_4)	%	-	1.98

Four cyanidation tests were carried out on the calcine the results of which may be seen in the following self-explanatory table.

TABLE 19

Results of Cyanidation of Calcine from Roasting Test 16

Test No.	Agitation Time hr	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		RP*	Residue oz/ton Au	Extraction** % Au	Remarks
		NaCN	CaO	NaCN	CaO				
16A	48	0.50	-	4.80	-	100	0.225	77.5	No lime
16B	"	"	0.50	5.80	23.00	120	0.225	77.5	
16C	"	"	-	1.00	-	128	0.220	78.0	(6 hr lime wash prior to cyanidation
16D	"	1.00	-	1.20	-	108	0.220	78.0	

* RP - Reducing power - ml N/10 $KMnO_4$/1

** extraction calculated on a head assay of 1.00 oz Au/ton
the dilution in all tests was 10:1

Test 17

A 738 g sample of the arsenopyrite-pyrite flotation tailing from Test 14 was the charge used in this test. A summary of the roasting conditions and the observations made may be seen in the following table.

TABLE 20

Results of Roasting Test 17

Elapsed Time min	Temp °C	Observations
0	450	Fan on - door open 10 in. continuous rabbling
10	400	Temp drop 50°C no odour or visible fumes
20	415	Slight visible fumes
25	425	Charge glowing - heavy fumes
30	445	" " " "
75	450	No visible fumes - rabbling - raise temp
135	600	" " " "
165	600	Pull charge

The loss in weight during this test was 19.0%.

The entire calcine was ground in the same manner as in Test 16 and a sample was riffled out for chemical analyses.

TABLE 21

Results of Analysis of Roaster Feed and Calcine

Element	Roaster Feed	Calcine
Gold (Au) oz/ton	0.80	0.99
Antimony (Sb) %	1.16	0.69
Arsenic (As) %	6.08	1.48
Sulphur (Total S) %	25.53	1.24

Two cyanidation tests were carried out on this calcine, the results of which may be seen in the following table.

TABLE 22

Results of Cyanidation of Calcine from Roasting Test 17

Test No.	Agitation Time hr	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		Residue oz/ton Au	Extraction % Au	Remarks
		NaCN	CaO	NaCN	CaO			
17A	48	1.00	-	2.80	-	0.20	79.8	Lime wash
17B	"	1.00	-	2.60	-	0.20	79.9	Lime wash - 0.3 g PbO added

Test 18

A 738 g sample of the arsenopyrite-pyrite flotation tailing from Test 14 was the charge used in this roasting test.

All of the roasting conditions were identical to those which prevailed in Test 17 with the exception that the finishing temp was 700°C.

The calcine was ground to 99.5% -200 M (94.5% -325 M) and a head sample was riffled out for a chemical analysis. The analytical results may be seen in the following table.

TABLE 23

Results of Analysis of Roaster Feed and Calcine

Element	Roaster Feed	Calcine
Gold (Au) oz/ton	0.80	1.03
Antimony (Sb) %	1.16	0.61
Arsenic (As) %	6.08	1.30
Sulphur (Total S) %	25.53	0.61

The results of cyanidation tests on samples of the calcine from Test 18 may be seen as follows.

TABLE 24

Results of Cyanidation of Calcine from Roasting Test 18

Test No.	Agitation Time hr	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		Residue oz/ton Au	Extraction % Au	Remarks
		NaCN	CaO	NaCN	CaO			
18A	48	0.5	0.5	3.0	26.0	0.21	79.6	Straight Cyanidation
18B	"	"	"	1.0	16.4	0.22	78.6	6 hr lime wash 50 lb CaO/ton calcine
18C	"	"	"	1.0	15.6	0.22	78.6	6 hr lime wash 50 lb CaO/ton calcine with 1.0 lb PbO added
18D	30	"	"	0.2	0.2	0.27	73.8	24 hr lime wash 48 lb/ton calcine with 2.0 lb PbO added

The reducing power in these tests was negligible.

The dilution in all tests was 10:1.

The CaO consumption indicated is exclusive of lime used in washing.

Test 19

To a 600 g sample of the arsenopyrite-pyrite flotation concentrate from Test 14, 60 g of salt was added and the charge was thoroughly mixed. The roasting conditions and observations may be seen in the following table.

TABLE 25

Results of Roasting Test 19

Elapsed Time Min	Temp °C	Observation
0	300	Charge in
15	350	Fan on, door open 5 in. rabbling, no fumes visible
35	400	Fan on, door open 5 in. rabbling, no fumes no odour
40	410	Visible fumes - continuous rabbling
50	430	Heavy white fumes " "
100	430	No fumes visible door open 10 in.
205	650	Temp raised slowly, rabbling, door open
265	650	Pull charge, cool

The calcine was ground to 99.8% -200 M and two samples were riffled out for cyanidation for 48 hr and 72 hr at a solution strength of 0.5 lb NaCN and 0.5 lb CaO/ton.

The results which were obtained were no better than those obtained in the previous roasting and cyanidation tests.

Test 20

The charge in this test was a 700 g sample of the arsenopyrite-pyrite flotation concentrate from Test 15. The roasting conditions and observations may be seen as follows.

TABLE 26

Results of Roasting Test 20

Elapsed Time Min	Temp °C	Observations
0	480	Charge in, fan on, door open 10 in.
7	440	Continuous rabbling during temp drop
20	470	As_2O_3 evolving continuous rabbling
25	480	" " " "
55	480	No arsenic fumes visible
175	650	Raise temp, rabble intermittently
235	650	Pull charge, cool

The weight loss during this roast was 16.8%.

The calcine was ground to approximately 99.0% -200 M and 2- 200 g samples were riffled out for cyanidation. A head sample was also obtained for a chemical analysis.

TABLE 27

Results of Analysis of Roaster Feed and Calcine

Element		Roaster Feed	Calcine
Gold (Au) oz/ton		0.73	0.84
Antimony (Sb)	%	0.31	0.13
Arsenic (As)	%	5.21	1.15
Sulphur (Total S)	%	14.52	1.32

The results of the cyanidation tests on the calcine from Test 20 may be seen in the following table.

TABLE 28

Results of Cyanidation of Calcine from Roasting Test 20

Test No.	Agitation Time hr	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		Residue oz/ton Au	Extraction % Au	Remarks
		NaCN	CaO	NaCN	CaO			
20A	72	1.0	1.0	1.6	30.0	0.11	86.8	
20B	"	"	"	2.3	43.6	0.105	87.5	grind in lime 5 lb/ton calcine

The reducing power in these tests was negligible.

The dilution in each test was 5:1.

The CaO consumption is exclusive of lime used in grinding.

Test 21

The only difference between this roasting test and Test 20 was the addition of 70 g of salt to the 700 g sample of the arsenopyrite-pyrite flotation concentrate from Test 15. The weight loss during the roast was 15.2% of the weight of the original charge.

The calcine produced was ground to approximately 99.0% -200 M and a head sample was riffled out for a chemical analysis. In Table 29 which follows the chemical analysis shown as the roaster feed has been calculated to allow for the salt dilution.

TABLE 29

Results of Analysis of Roaster Feed and Calcine

Element	Roaster Feed	Calcine
Gold (Au) oz/ton	0.66	0.68
Antimony (Sb) %	0.28	0.13
Arsenic (As) %	4.74	0.92
Sulphur (Total S) %	13.21	2.23

The following table shows the results of two cyanidation tests done on this calcine.

TABLE 30

Results of Cyanidation of Calcine from Roasting Test 21

Test No.	Agitation Time hr	Charge Weight g	Residue Weight g	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		Residue oz/ton Au	Extraction % Au	
				NaCN	CaO	NaCN	CaO		Apparent	Actual*
21-A	72	200.0	187.0	1.0	1.0	1.6	51.5	0.24	64.7	66.6
21-B	"	"	186.0	"	"	1.5	49.4	0.24	64.7	66.6

The reducing power in these tests was negligible.

The dilution in each test was 5:1.

* The apparent extraction was corrected to allow for the weight loss during cyanidation.

In the two tables which follow may be seen the results of screen tests on the roaster feeds and on the calcines from the roasting tests. A microscopic examination of the calcines did not reveal any of the glassy substance usually associated with fusion but it is probably true to say that there was some incipient fusion.

TABLE 31

Results of Screen Tests

Mesh Size	Roaster Feed Weight %	Calcines Weight %			
		Test 16	Test 17	Test 18	Test 19
+ 65M	-	17.5	14.2	17.6	23.0
- 65M +100"	1.0	7.4	8.8	7.4	7.0
-100" +150"	7.8	11.3	11.7	10.6	11.3
-150" +200"	18.4	16.3	17.1	14.7	11.4
-200" +325"	32.5	32.7	28.1	19.3	16.4
-325"	40.3	14.8	20.1	30.4	30.9
TOTAL	100.0	100.0	100.0	100.0	100.0

TABLE 32

Results of Screen Tests

Mesh Size	Roaster Feed Weight %	Calcines Weight % Test 20	Calcines Weight % Test 21
+ 65M	-	17.2	15.5
- 65M +100"	0.2	3.2	4.2
-100" +150"	2.2	4.2	3.7
-150" +200"	5.8	7.0	6.2
-200" +325"	14.1	16.2	13.6
-325"	77.7	52.2	56.8
TOTAL	100.0	100.0	100.0

Test 22 A-B

In all previous roasting tests, the roaster feed was an arsenopyrite-pyrite flotation concentrate from which as much of the stibnite as possible had been removed.

In order to determine the effect of a salt addition on the roasting and subsequent cyanidation of an arsenopyrite-pyrite concentrate containing a large percentage of antimony, an artificial roaster charge was prepared from the rejects of the flotation products obtained during the investigation. This concentrate was divided into two equal fractions of 520 g, hereafter designated Test 22A and Test 22B. To Test 22B 52.0 g of salt was added.

Each test charge was roasted separately in a muffle furnace with the roasting time, temperature and the observations being nearly identical during each roast. The following table shows the roasting conditions which prevailed during each test.

TABLE 33

Results of Roasting Test 22A-B

Elapsed Time Min	Temp °C	Observations
0	450	Charge in, door open, fan on, temp drop
5	420	Slight fumes, rabbling
15	450	Heavy fumes, continuous rabbling
65	450	No fumes visible, temp raised to 650°C with door closed and intermittent rabbling
95	650	Door open continuous rabbling
120	650	No fumes or odour detected, charge pulled, dull red colour

In the following table, the results of a chemical analysis on the roaster feed and on the two calcines are shown. The calculated assays shown in the roaster feed column Test 22B allow for the charge dilution occassioned by the addition of the salt.

TABLE 34

Chemical Analysis of Roaster Feed and Calcines

Element	Roaster Feed		Calcine	
	Test 22A	Test 22B*	Test 22A	Test 22B
Gold (Au) oz/ton	0.82	0.74	1.16	1.00
Antimony (Sb) %	6.18	5.61	5.40	3.04
Arsenic (As) %	8.30	7.55	1.15	7.05
Sulphur (Total S) %	28.10	25.00	0.85	1.54

*Calculated

As might be expected, when a concentrate containing such a high sulphur percentage was given a dead sweet roast, there was a large weight loss. This amounted to 29% and 30.6% respectively in Test 22A and Test 22B.

Each calcine was given a 30 min grind in a ball mill using steel balls and although no size analysis of the ground pulp was made a visual observation indicated that the grind was extremely fine.

The two ground calcines were cyanided for 72 hr at a dilution of 10:1 and a solution strength of 1.0 lb NaCN and 1.0 lb CaO/ton. The following table shows the results which were obtained by cyanidation.

TABLE 35

Results of Cyanidation of Calcines from Test 22A-B

Test No.	Agitation Time hr	Charge Weight g	Residue Weight g	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		Residue oz/ton Au	Extraction % Au	
				NaCN	CaO	NaCN	CaO		Apparent	Actual*
22-A	72	200.0	200.0	1.0	1.0	1.80	18.60	0.875	24.6	-
22-B	"	"	176.0	"	"	1.60	16.40	0.46	54.0	60.0

The reducing power of the solutions in these tests was negligible.

*The apparent extraction was corrected to allow for the weight loss during cyanidation.

The results of these tests indicated that roasting with salt increased the extraction of gold by over 50%.

In order to show the effect of salt on the roasting and subsequent cyanidation of a bulk sulphide concentrate the following test was done.

Test 23

Two 2000 g samples of -10 M ore were ground to approximately 95.0% -200 M (75.0% -325 M) and floated using the flotation procedure outlined in Test 7. The rougher concentrates were combined and the whole was cleaned 3 times. The results of this phase of the test were as follows.

TABLE 36

Results of Test 23 (Flotation)

Product	Weight %	Assays				Distribution %			
		oz/ton	%						
		Au	Sb	As	S	Au	Sb	As	S
Final concentrate	10.7	1.00	2.03	7.51	27.47	63.8	57.2	66.6	74.5
3rd cl tailing	0.9	0.62	0.24	4.17	11.10	3.3	0.6	3.1	2.5
2nd " "	1.4	0.35	0.24	2.34	7.05	2.9	8.8	2.7	2.5
1st " "	12.6	0.14	0.12	0.92	2.33	10.5	4.0	9.7	7.4
Flotation "	74.4	0.044	0.15	0.27	0.69	19.5	29.4	17.9	13.1
Head (calcd)	100.0	0.168	0.38	1.20	3.94	100.0	100.0	100.0	100.0

The final concentrate was roasted with 10% salt added using the same conditions outlined in Test 22. There was a weight loss of 26.4% of the total charge.

The following table shows the chemical analysis of the roaster feed and calcine. The feed analysis has been corrected for the dilution caused by the addition of the salt.

TABLE 37

Chemical Analysis of Roaster Feed and Calcine

Element	Roaster Feed	Calcine
Gold (Au) oz/ton	0.91	0.99
Antimony (Sb) %	1.84	1.79
Arsenic (As) %	6.83	0.99
Sulphur (Total S) %	24.97	3.23

The calcine was ground for 30 min in a ball mill with steel balls. The results of screen tests on the roaster feed and on the roaster discharge and ground calcine were as follows.

TABLE 38

Results of Screen Tests

Mesh Size	Roaster Feed Weight %	Roaster Discharge Weight %	Calcine (ground) Weight %
+ 65M	-	2.4	-
- 65M +100"	0.6	2.9	-
-100" +150"	3.3	10.5	-
-150" +200"	8.2	14.9	0.3
-200" +325"	23.2	32.4	2.0
-325"	64.7	36.9	97.7
Total	100.0	100.0	100.0

The ground calcine was cyanided 72 hours at a dilution of 10:1 and a solution strength of 1.0 lb NaCN and 1.0 lb CaO/ton. The results which were obtained may be seen in the following table.

TABLE 39

Results of Cyanidation of Calcine from Test 23

Test No.	Agitation Time hr	Charge Weight %	Residue Weight %	Reagents Added lb/ton Soln		Consumption lb/ton Calcine		Residue oz/ton Au	Extraction % Au	
				NaCN	CaO	NaCN	CaO		Apparent	Actual*
23	72	250	227	1.0	1.0	13.6	49.5	0.355	64.1	70.6

The reducing power of this solution was negligible.

*The apparent extraction was corrected to allow for the weight loss during cyanidation.

CONCLUSIONS

The results of the test work indicated that this is a very refractory ore and difficulty will be experienced in obtaining a high recovery of the gold.

Neither straight amalgamation nor straight cyanidation of the ore is feasible.

Approximately 90.0% of the gold can be recovered by flotation in a bulked sulphide concentrate as may be seen in the results of Test 15 shown in Table 16. In this test the removal of a stibnite concentrate from the bulked sulphide concentrate by differential flotation resulted in a loss of 13.5% of the gold, so that the arsenopyrite-pyrite flotation concentrate, representing the roaster feed, contained only 76.7% of the gold.

In Test 15 a stibnite concentrate assaying 30.50% Sb and representing 71.4% of the Sb was obtained.

Roasting tests were carried out on samples of the arsenopyrite-pyrite flotation concentrate from tests 14 and 15 followed by cyanidation of the calcines. Of all these tests the best results were obtained in Test 20 and are shown in Table 28. In this test an extraction of 87.5% of the gold in the calcine was obtained which when referred to flotation Test 15 represented only 67.1% of the gold in the ore.

In contrast, the results of Test 22-A show that if a large percentage of Sb is allowed to remain in the roaster feed a very low extraction of the gold may be expected when the calcine is cyanided.

Several roasting tests were done in which salt was mixed with the roaster feed prior to calcination. The results of these tests show that there is no advantage to be gained by adding salt to a roaster feed from which the stibnite has been substantially removed. However, the results of Test 22A-B and Test 23 show that a considerable advantage is gained by roasting with salt, concentrates containing large percentages of antimony. In Test 23 the gold extraction from the cyanidation of the resulting calcine was 70.6%. However, the overall recovery of gold in this test was less than that obtained in Test 20.

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TFB/Jg