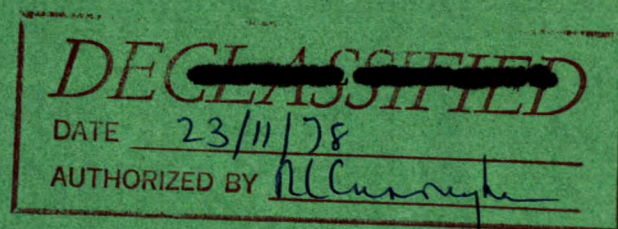


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CANADA

DEPARTMENT OF MINES AND TECHNICAL SURVEYS

OTTAWA

MINES BRANCH INVESTIGATION REPORT IR 62-49

**INVESTIGATION OF A GOLD ORE FROM
McKINNEY GOLD MINES LTD.,
ROCK CREEK, B. C.**

by

T. F. BERRY

MINERAL PROCESSING DIVISION

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Mines Branch Investigation Report IR 62-49
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SUMMARY OF RESULTS

The results of the investigation show that a gold recovery in excess of 95% can be attained by straight cyanidation of the ore and over 98% by amalgamation followed by cyanidation of the amalgamation residue.

In the flowsheet outlined in Test 5, an overall recovery of about 97% of the gold and about 85% of the silver was obtained by the amalgamation of a jig concentrate followed by the flotation of the jig tailing and the subsequent cyanidation of the flotation tailing.

In this test a final lead concentrate assaying 60.20% Pb and a final zinc concentrate assaying 54.27% Zn were produced. The recovery of these elements was lower than might be expected in a full-sized plant.

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INTRODUCTION

In a letter dated August 7, 1961, from the consulting engineering firm of H. Hill and L. Stark and Associates Ltd., 844 West Hastings Street, Vancouver 1, B.C., Mr. L. P. Stark asked the Mines Branch to undertake an investigation on a gold ore from McKinney Gold Mines Ltd., Rock Creek, B.C.

Location of Property

The property of McKinney Gold Mines Ltd. is reached by a seven mile branch road running north from Highway No. 3, three miles east of Bridesville in south central B.C. Rock Creek, a station on the Kettle Valley branch of the C.P.R., is nine miles east of the above road junction on Highway No. 3.

Shipment

On August 22, 1961, a 160-lb shipment of lump ore was received at the Mines Branch in Ottawa.

Nature of Investigation Requested

As this was essentially a gold ore, the primary purpose of the investigation was to develop a flowsheet for the recovery of the gold. Of secondary importance was the concentration of the small amounts of lead and zinc in the ore.

Sampling and Analysis

Several representative pieces of the ore were selected for a mineralogical study and the remainder of the shipment was crushed to -10 mesh. A head sample was riffled out for a chemical analysis. A semi-quantitative spectrographic analysis of a small portion of the head sample showed only lead and zinc in economic amounts.

TABLE 1
Results of Chemical Analysis of Head Sample

Element		
Gold	(Au)	1.79 oz/ton
Silver	(Ag)	1.72 "
Lead	(Pb)	0.82 %
Zinc	(Zn)	1.14 "
Copper	(Cu)	0.12 "
Nickel	(Ni)	0.02 "
Iron (Soluble Fe)		4.80 "
Sulphur (Total S)		3.60 "
Insoluble		74.60 "

From Internal Report MS-AC-61-748.

MINERALOGICAL EXAMINATION^{*}

Method of Examination

Four polished sections were prepared and the minerals identified by microscopical and X-ray diffraction studies. A -65+200 mesh fraction of the sample was separated into fractions by means of heavy liquids.

Results of Examination

The metallic minerals present in the sample are pyrite, chalcopyrite, galena, sphalerite, pyrrhotite, and a mineral tentatively regarded as a telluride. The non-metallic minerals are quartz, altered feldspar, dolomite, rutile, and garnet.

Pyrite, the principal metallic mineral, occurs as masses and as sub-hedral grains disseminated in the gangue. The pyrite grains range between 0.02 mm and 0.6 mm in diameter and contain numerous veinlets and tiny rounded blebs of chalcopyrite, galena, sphalerite, and a cream-coloured mineral that may be a telluride (see Figures 1 and 2). The blebs and tiny veinlets of the cream-coloured mineral are so small that they cannot be positively identified, but their optical properties are similar to those of gold-silver telluride.

The chalcopyrite, galena and sphalerite also occur as larger irregularly-shaped grains (see Figure 3). The chalcopyrite and galena grains range up to 0.5 mm, and the sphalerite grains up to 0.2 mm, in diameter. The sphalerite contains rounded chalcopyrite blebs up to 5 microns in diameter (see Figure 4).

^{*} Internal Report No. MS-61-89, by W. Petruk, September 19, 1961.

Pyrrhotite was not observed in the polished sections but it is present in the -65+200 mesh fraction.

The rutile occurs as small grains in the dolomite-quartz gangue, the individuals showing marked parallel alignment.

A heavy liquid separation of the ore shows that it is composed of about 90% quartz, about 5% dolomite, altered feldspar, garnet and rutile, and about 5% metallic minerals.

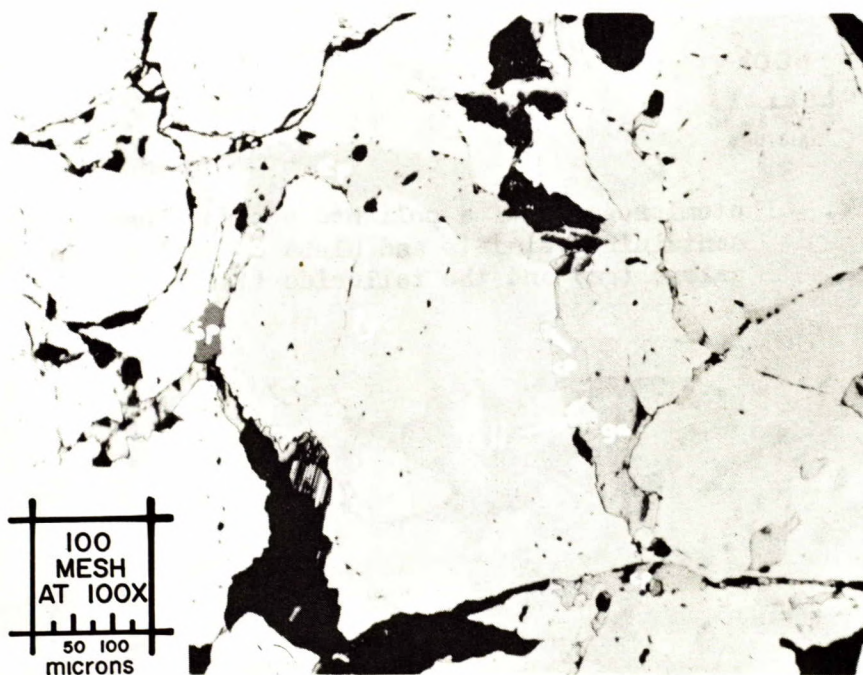


Figure 1. - Photomicrograph of a polished section showing pyrite (py) containing veinlets and blebs of chalcopyrite (cp), galena (ga), sphalerite (sp) and telluride (?) (tell).

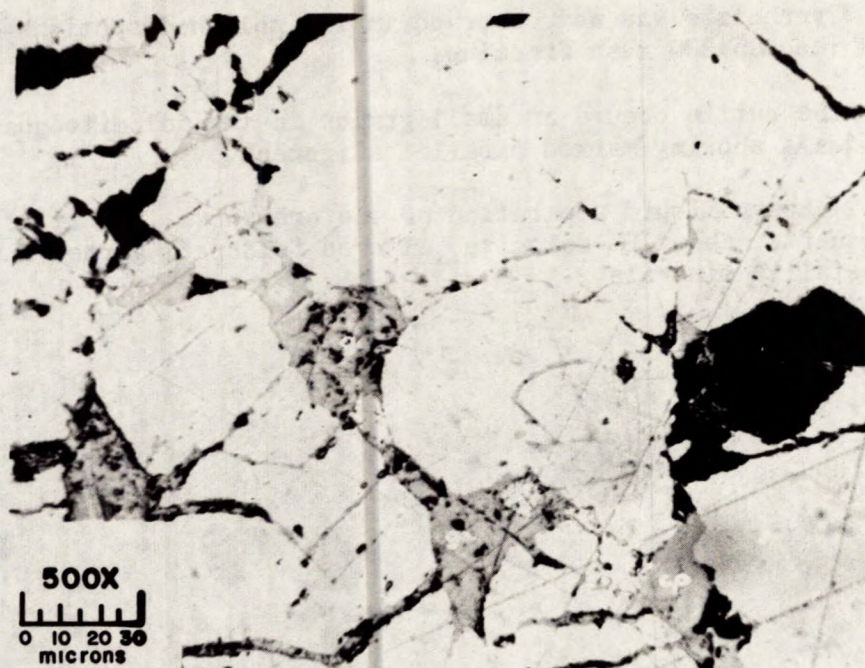


Figure 2. - Photomicrograph of a polished section showing pyrite (py) containing veinlets and blebs of chalcopyrite (cp), galena (ga) and the telluride (?) (tell).

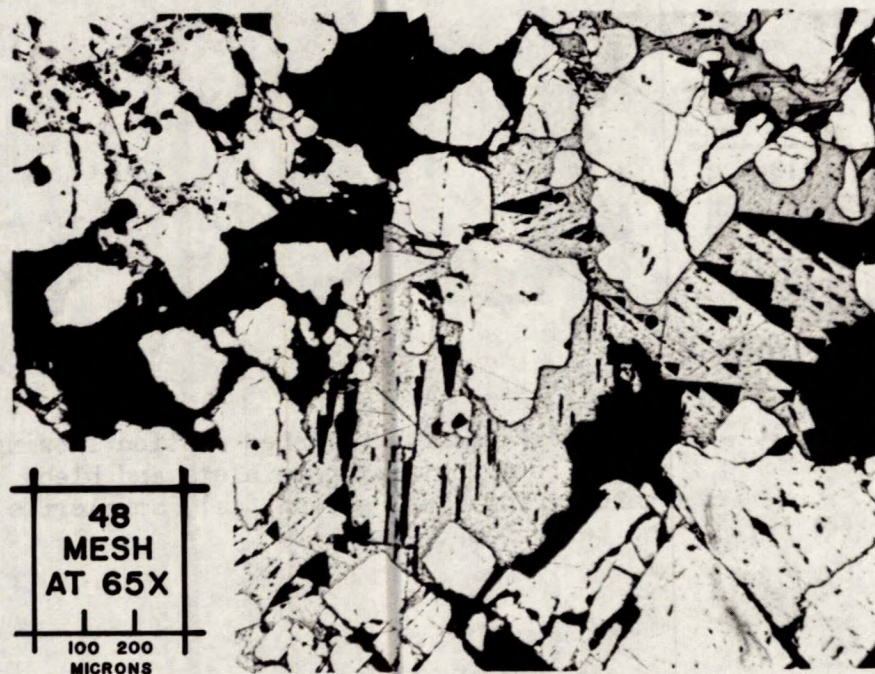


Figure 3. - Photomicrograph of a polished section showing irregularly shaped grains of galena (ga) and chalcopyrite (cp) in pyrite (py).

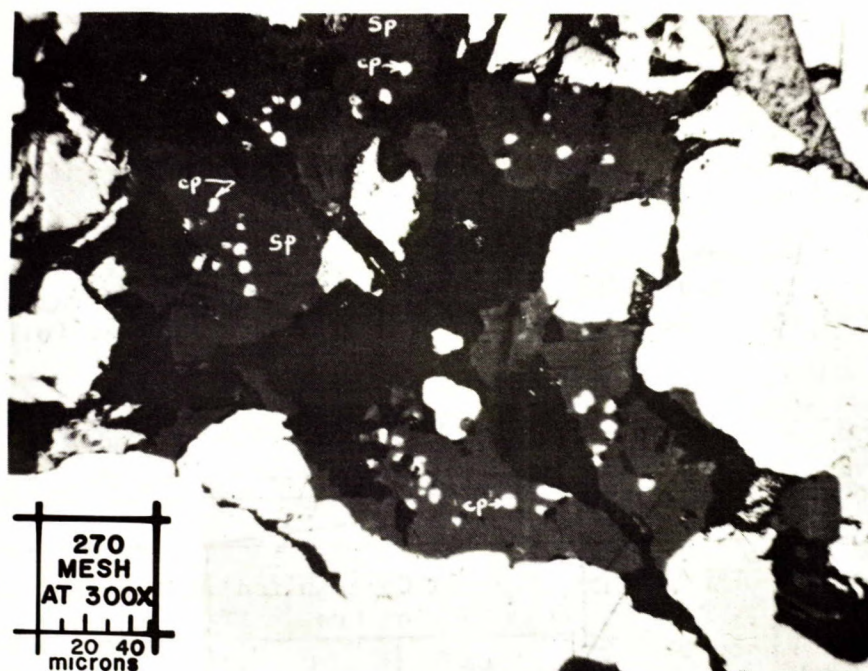


Figure 4. - Photomicrograph of a polished section showing sphalerite grains (sp) containing rounded blebs of chalcopyrite (cp).

DETAILS OF INVESTIGATION

Test 1

A 1500-g sample of -10 m ore was ground to 80.7% -200 m and was amalgamated for 3 hr with 1.0 lb lime/ton and 25 ml of mercury. The results which were obtained may be seen in the following table:

TABLE 2

Results of Amalgamation of Ground Ore

Feed	1.790 oz Au/ton
Amalgamation residue	0.176 "
Extraction	1.614 "
*Extraction %, Au	90.2

*Calculated by difference.

Test 2 and 3

The amalgamation residue from Test 1 was split into two fractions for cyanidation.

Each sample was cyanided for 48 hr at a solution strength of 1.0 lb NaCN/ton and 1.0 lb lime/ton. In Test 2 the water:solids ratio was 5:1 and in Test 3 it was 10:1. The results of these tests were as follows:

TABLE 3

Results of Cyanidation of Amalgamation Residue from Test 1

Test No	Grind % -200m	Agitation Time hr	Reagent Consumption lb/ton Ore		Tail Assay [*] oz/ton	Extraction ^{**} %
			NaCN	CaO	Au	Au
2	80.7	48	1.30	4.10	0.02	88.6
3	"	"	2.00	5.60	0.02	88.6

* From Internal Report MS-AC-61-774.

** The extraction was calculated by difference on a head assay of 0.176 oz Au/ton.

The overall extraction of gold in the first three tests may be summarized as follows:

By amalgamation - 90.2 %

By cyanidation - 8.7 %

Total - 98.9 %

Test 4

A 6000-g sample of -10 m ore was riffled out and jigged in 2000-g lots in a Denver laboratory jig. The jig tailing was retained separately for grinding and flotation.

The combined jig concentrate, weighing 318.0 g, was ground for 1 hr and amalgamated for 2 hr with 1.0 lb lime/ton of concentrate and 10 ml of mercury. The amalgam and a small portion of the amalgamation residue were assayed for gold.

TABLE 4
Results of Amalgamation of Jig Concentrate from Test 4

Jig concentrate (calcd)	21.58 oz Au/ton
Amalgamation residue	9.12 oz Au/ton
*Amalgam	12.46 " "
Extraction %, Au	57.7

Assays from Internal Report MS-AC-61-989.

*Amalgam assay expressed in oz Au/ton of
jig concentrate.

The rather low extraction of 57.7% is due probably to the presence of floured mercury, some of which may have been inadvertently washed into the amalgamation residue during the cleaning operation.

The jig tailing was ground to 66.0% -200 m in 2000-g lots with 1.0 lb NaCN/ton and 0.3 lb ZnSO₄/ton.

Three lead rougher concentrates were floated and combined, using Aerofloat promoter 31 and cresylic acid in the amount of 0.04 and 0.02 lb/ton respectively. The combined lead rougher concentrate was cleaned after conditioning with 0.1 lb NaCN/ton and then was recleaned without additional reagents in a 50-g flotation cell.

After the lead flotation the pulp was conditioned with lime to a pH of 10.0 and then was further conditioned for 5 min with 1.0 lb CuSO₄/ton. Sodium Aerofloat and cresylic acid in the amount of 0.1 and 0.02 lb/ton respectively were used to float a zinc rougher concentrate which was cleaned twice with no additional reagents being used.

In Table 5, the results of the flotation of the jig tailing are shown, along with the extraction of gold by amalgamation.

TABLE 5

Results of Flotation of Jig Tailing from Test 4

Product	Weight %	Assays [*]			Distribution		
		oz/ton	%		%		
		Au	Pb	Zn	Au	Pb	Zn
Hg amalgam ^{**}	-	0.664	-	-	34.1	-	-
Pb rec1 concentrate	1.8	37.37	26.48	19.30	34.7	62.6	28.8
" " tailing	0.4	45.50	4.40	15.00	9.4	2.3	5.0
" c1 "	1.6	7.45	5.83	9.11	6.2	12.2	12.1
Zn rec1 concentrate	0.6	2.73	1.06	39.64	0.8	0.8	19.7
" " tailing	1.6	1.135	0.46	16.73	0.9	1.0	22.2
" c1 "	2.8	1.61	0.86	2.38	2.3	3.2	5.5
Flotation "	91.2	0.247	0.15	0.09	11.6	17.9	6.7
Head (calcd)	100.0	1.943	0.76	1.21	100.0	100.0	100.0

^{*}From Internal Report MS-AC-61-989.

^{**}Assay expressed in oz Au/ton of feed.

Three samples of the flotation tailing were riffled out and were cyanided for 24 hr, 48 hr and 72 hr at a solution strength of 0.5 lb NaCN/ton and 0.5 lb lime/ton and at a dilution of 2:1. The results were as follows:

TABLE 6

Results of Cyanidation of Flotation Tailing from Test 4

Grind % -200m	Agitation Time hr	Reagent Consumption lb/ton Ore		Tail Assay [*] oz/ton	Extraction ^{**} %
		NaCN	CaO	Au	Au
66.0	24	0.98	2.20	0.0075	97.0
"	48	1.14	2.56	0.005	98.0
"	72	1.20	2.84	0.004	98.4

^{*}From Internal Report MS-AC-62-54.

^{**}The extraction was calculated by difference on a head assay of 0.247 oz Au/ton.

The overall gold recovery from Test 4 may be summarized as follows:

By jigging and amalgamation	- 34.1 %
*By lead flotation	- 48.7 %
By cyanidation, 48 hr	- 11.4 %
Total	- 94.2 %

*The lead middling products contain 15.6% of the gold. It has been assumed that 90% of this gold is recoverable.

Test 5

In Test 4 the extraction by amalgamation of the jig tailing was low, probably due to floured mercury in the amalgamation residue.

In the present test approximately 8000 g of -10 m ore was ground in lots of 2000 g each for about 5 min. Each lot was jigged to recover a jig concentrate. The four jig concentrates and the jig bed were combined and the resultant concentrate, weighing 356.0 g, was ground for 15 min in a porcelain mill.

The ground pulp was amalgamated for 3 hr with 2.5 lb lime/ton of concentrate and 20 ml of new mercury. The amalgam and a sample of the amalgamation residue were assayed for gold. During the amalgam cleaning, a careful check was made for floured mercury. The results of this phase of Test 5 were as follows:

TABLE 7

Results of Amalgamation of Jig Concentrate from Test 5

Jig concentrate (calcd)	22.935 oz Au/ton
Amalgamation residue	1.145 " "
*Amalgam	21.790 " "
Extraction %, Au	95.0

Assays from Internal Report MS-AC-62-712.

*Amalgam assay expressed in oz Au/ton of jig concentrate.

The remaining amalgamation residue was added in equal parts to the four jig tailing fractions. Each fraction was then ground for 15 min and the lead and zinc were recovered by flotation. The combined lead rougher concentrate and the combined zinc rougher concentrate were each cleaned in a 500-g

Denver flotation cell, and the cleaner concentrates were recleaned in a 50-g cell. This was an attempt to establish the fact that high-grade concentrates could have been produced in a conventional flotation cell if enough material had been available for the cleaning operation.

The lead and zinc cleaner tailings were added to the rougher tailing for subsequent cyanidation. This was to determine whether it was feasible to recover the gold in this fraction at this point or whether, due to the percentage of lead and zinc in this fraction, it would be necessary to re-cycle the cleaner tailing to the head of the flotation circuit. Unfortunately, due to an oversight, the cleaner tailing fractions were not assayed prior to their being mixed with the rougher tailing. However, since the flotation schemes of Test 4 and 5 were almost identical, when one compares the flotation tailing results in Test 4 with those obtained in the present test, it is obvious that a major portion of the lead and zinc in the feed to cyanidation is due to the inclusion of the lead and zinc cleaner tailings.

In Test 5 the following flotation scheme was used:

To grinding:	Sodium sulphite	-	1.0 lb/ton ore	
	Zinc sulphate	-	0.5 " " "	
	Aerofloat 208	-	0.05 " " "	
	Aerofloat 31	-	0.04 " " "	

To lead flotation:	Aerofloat 242	-	0.02 " " "	
(6 min)	Cresylic acid	-	0.02 " " "	

To lead cleaner flotation:

Sodium cyanide	-	0.02 " " "	
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To lead recleaner flotation:

No reagents.

To zinc conditioner:

(10 min)	Lime	-	0.5 " " "	(pH - 10.1)
	Copper sulphate	-	1.0 " " "	
	Sodium Aerofloat	-	0.09 " " "	

To zinc flotation:

(4 min)	Dowfroth 250	-	0.02 " " "	
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No reagents were added to the zinc cleaner or recleaner cells.

The following table shows the results of this flotation test and the extraction of gold by amalgamation:

TABLE 8
Results of Flotation of Jig Tailing from Test 5

Product	Weight %	Assays [*]				Distribution			
		oz/ton		%		%			
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Hg amalgam ^{***}	-	0.99	-	-	-	64.7	48.2	-	-
Pb rec1 conc	0.4	36.88	61.29	60.20	5.80	9.6	14.2	41.0	2.4
" " tailing	0.9	23.88	33.79	20.17	16.47	14.0	17.7	30.9	15.0
" cl "	(3.4)	-	-	-	-	-	-	-	-
Zn rec1 conc	0.4	0.88	2.00	0.67	54.27	0.2	0.5	0.4	22.0
" " tailing	0.4	1.31	2.83	1.51	29.03	0.4	0.7	1.0	11.8
" cl "	(2.1)	-	-	-	-	-	-	-	-
Flotation ^{***}	97.9	0.175	0.33	0.17	0.49	11.1	18.7	26.7	48.8
Head (calcd)	100.0	1.53	1.72	0.59	0.98	100.0	100.0	100.0	100.0

^{*} From Internal Reports MS-AC-62-712 and MS-AC-62-863.

^{**} Assay expressed in oz Au/ton of feed. Based on head assay of 1.72 oz Ag/ton, the recovery of Ag by amalgamation was calculated by difference.

^{***} Includes lead and zinc cleaner tailing.

Three samples of the flotation tailing were cyanided for 24 hr, 48 hr, and 72 hr at a dilution of 2:1 and a solution strength of 0.5 lb NaCN/ton and 0.5 lb lime/ton.

The following table shows the results of the cyanidation tests:

TABLE 9

Results of Cyanidation of Flotation Tailing from Test 5

Grind % -200m	Agitation Time hr	Reagent Consumption lb/ton ore		Tail Assay [*] oz/ton		Extraction ^{**} %	
		NaCN	CaO	Au	Ag	Au	Ag
66.0	24	0.88	1.00	0.02	0.16	88.5	51.5
"	48	1.20	1.68	0.017	0.12	90.3	63.6
"	72	1.32	1.92	0.015	0.12	91.4	63.6

^{*}From Internal Report MS-AC-62-863.

^{**}The extraction was calculated by difference on a head assay of 0.175 oz Au/ton and 0.33 oz Ag/ton.

The overall recovery of the gold and silver in Test 5 may be summarized as follows:

		Au	Ag
By jigging and amalgamation	-	64.7 %	48.2 %
[*] By lead flotation	-	22.2 %	24.8 %
By cyanidation (48 hr)	-	10.0 %	11.9 %
Total	-	96.9 %	84.9 %

^{*}The lead recleaner tailing contained 14.0% of the gold and 17.7% of the silver. It has been assumed that 90% of this gold and 60% of this silver will be recovered in plant operation, either in the final lead concentrate or in the flotation tailing, in which product they are subject to cyanidation.

Test 6

A 5000-g sample of -10 m ore was ground to about 65.0% -200 m and was cyanided for 72 hr at a dilution of 2:1 and a solution strength of 0.5 lb NaCN/ton and 0.5 lb lime/ton. The pregnant solution was retained and was assayed to determine the extraction of gold by cyanidation. The cyanide residue was subjected to flotation to recover the lead and the zinc. The flotation conditions and the reagents used were identical to those shown in Test 4. The results obtained in this test may be seen in Table 7.

TABLE 10

Summary of Results of Test 6

Product	Weight %	Assays [*]			Distribution		
		oz/ton	%		%		
		Au	Pb	Zn	Au	Pb	Zn
Pregnant solution	(23.4 litres)	0.332	-	-	95.9	-	-
Pb rougher conc	2.2	0.70	18.76	10.00	1.0	50.7	17.9
Zn cl conc	1.8	0.305	3.12	29.78	0.3	6.9	43.6
" " tailing	2.4	0.220	3.44	11.50	0.3	10.2	22.5
Flotation tailing	93.6	0.0425	0.28	0.21	2.5	32.2	16.0
Head (calcd)	100.0	1.616	0.81	1.23	100.0	100.0	100.0

^{*} From Internal Report MS-AC-62-240.

Test 7

A 5000-g sample of -10 m ore was ground to about 65.0% -200 m and a lead and zinc concentrate were recovered, using the same reagents and conditions shown in Test 4. The flotation tailing was cyanided for 48 hr at a solution strength of 1.0 lb NaCN/ton and 1.0 lb lime/ton at a dilution of 2:1. The following table summarizes the results which were obtained:

TABLE 11
Summary of Results of Test 7

Product	Weight %	Assays [*]			Distribution		
		oz/ton	%		%		
		Au	Pb	Zn	Au	Pb	Zn
Pb cl concentrate	1.3	33.89	30.80	9.63	26.9	50.6	10.8
" " tailing	3.8	13.29	3.12	9.91	30.8	24.7	32.4
Zn " concentrate	0.8	12.22	3.30	44.75	6.0	3.3	30.8
" " tailing	1.5	1.70	0.85	4.16	1.6	1.6	5.4
Pregnant solution	(21.22 litres)	0.53	-	-	32.2	-	-
Cyanide residue	92.6	0.045	0.17	0.26	2.5	19.8	20.6
Head (calcd)	100.0	1.64	0.79	1.16	100.0	100.0	100.0

* From Internal Report MS-AC-62-240.

CONCLUSIONS

The gold in this ore is readily recoverable by straight cyanidation or by amalgamation followed by cyanidation.

The results of Test 1 show that about 90% of the gold is free-milling. It is important that the flowsheet for treating this ore should include a method of recovering this free-milling gold as soon after liberation as possible. This is normally done by placing a jig between the ball mill and the classifier, or by amalgamation of the ground ore.

In Test 5, a jig concentrate was recovered from a sample of coarsely ground ore which, when amalgamated, yielded 95% of the gold in the concentrate, representing a recovery of nearly 65% of the gold in the ore. A calculated recovery of 48.2% of the silver in the ore was obtained in this test by amalgamation. Flotation of the jig tailing recovered an additional 9.6% of the gold and 14.2% of the silver in a final lead concentrate assaying 36.88 oz Au/ton, 61.29 oz Ag/ton and 60.20% lead. Cyanidation of the flotation tailing recovered an additional 10% of the gold and 11.9% of the silver.

If it is assumed that 90% of the gold and 60% of the silver in the lead recleaner tailing will be recovered in the final lead concentrate and/or by cyanidation of the final flotation tailing, this flowsheet will

give an overall recovery of nearly 97% of the gold and nearly 85% of the silver.

Although the lead and zinc contents of this ore are low, the investigation disclosed that saleable grades of these concentrates can be produced. It is expected that the recovery of these elements will be higher in a plant operation in which the middling fractions are re-circulated than could be obtained in laboratory tests.

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