Mineral Sciences V1

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MINES BRANCH INVESTIGATION REPORT IR 74-4

CONCENTRATION AND RECOVERY OF GOLD AND SILVER FROM A TELLURIDE ORE SUBMITTED BY KLEANZA MINES LIMITED, TERRACE AREA, BRITISH COLUMBIA

by

A. STEMEROWICZ AND R. W. BRUCE

MINERAL PROCESSING DIVISION

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COPY NO. 9

JANUARY 1974

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CONCENTRATION AND RECOVERY OF GOLD AND SILVER FROM A TELLURIDE ORE SUBMITTED BY KLEANZA MINES LIMITED, TERRACE AREA, BRITISH COLUMBIA

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A. Stemerowicz* and R. W. Bruce**

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SUMMARY OF RESULTS

The ore contained 0.41 oz/ton gold and 0.75 oz/ton silver both of which occurred as telluride inclusions in pyrite. Jigging of the ore crushed to minus 10 mesh followed by flotation of the jig tailing ground to 31% minus 200 mesh gave combined gold and silver recoveries of 86.2% and 86.0% respectively in a pyrite concentrate assaying 1.91 oz/ton gold and 3.39 oz/ton silver. Roasting and cyanidation of a pyrite concentrate of similar grade resulted in gold and silver extractions of 87.1% and 35.8% respectively.

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INTRODUCTION

Location of Property

The property is located approximately 7 miles east of Terrace, B.C. on Kleanza Mountain at an elevation of 2150 ft (Latitude 54^o - 32' N., Longitude 128^o - 27' W.).

Exploration work done on the property includes surface trenching, diamond-drilling and the driving of three adits. Proven ore has been conservatively estimated at 4300 tons. The probability of finding additional ore is good as the mineralized vein is open to extension in length and depth. There is also some evidence indicating the existence of a second vein.

Shipment

The ore sample, which was received on February 10, 1972, weighed 390 lb. It consisted of lump ore contained in 4 sacks and was submitted by Mr. R. H. Bates, President, Kleanza Mines Ltd (N.P.L.), P. O. Box 580, Terrace, B. C.

Nature of Investigation Requested

In his letter of January 7, 1972 Mr. Bates requested an investigation on the ore to determine the following:

(1) The fineness of grind required for the production of a pyrite concentrate from the point of view of (a) marketing the concentrate directly and (b) cyaniding the concentrate and marketing the bullion.

(2) Is it necessary to roast the concentrate before cyanidation?

(3) Would the tellerium content in the ore interfere with the concentration or treatment of the ore?

In the same letter it was stated that:

- The proposed mining rate was 25 to 30 ton/day.
- Natural gas and electric power were available at the property.
- Cost of hauling concentrates by surface transport to Trail, B. C. was $42/ton versus $8/ton by water transport along the coast to Tacoma, Wash., U. S.A.
- Approximately $70,000 was available for purchase of used mill equipment.
- The purchase of used crushing and gravity concentration equipment had been investigated and it was found that this equipment was readily available at a cost below the amount quoted above.

Sampling and Analysis

After selecting several mineralized lumps for mineralogical examination, all of the sample was crushed to approximately minus 1/4 inch and riffled into quarters. One of the quarters was further crushed to minus 10 mesh and riffled into 16 portions. One of these portions was chosen at random as a head sample while the remaining portions were used for the concentration tests.

TABLE 1

Head Sample Analysis *

Gold (Au)	0.41 oz/ton
Silver (Ag)	0,75 oz/ton
Iron (Fe)	11.35 %
Sulphur (S)	12.16 %

* From Internal Reports MS-AC-72-58 and 64.

TABLE 2

Semi-Quantitative Spectrographic Analysis of Head Sample*

Range %	Elements
Principal constituent	Si, Fe, Al
0.2 to 01	Ni, Mg, Ca
0.1 to 0.01	Ba, Ti, Mo, Mn, Ag
Not detected	Cd, Be, B, P, Sb, As, Ge, Pb, Sn, Na, Ga, Nb, Ta, In, Bi, Ba, V, Zr, Zn, Sr

* From Internal Report No. SL 4372

Mineralogical Examination*

A study of polished sections made from the mineralized hand specimens, minus 10-mesh head sample, and a sample of jig concentrate indicated that mineralization consisted essentially of massive and spongy varieties of pyrite in milky quartz. The gold and silver were present as tellurides which occurred as tiny inclusions and fillings in spongy pyrite. The gold values were in petzite (Ag, Au, Te) whereas the silver values were in hessite (Ag, Te), petzite and volynskite (Ag, Bi, Te). One very small grain of native gold, approximately 2 microns in diameter, was found. It occurred together with a telluride inclusion within pyrite.

OUTLINE OF INVESTIGATION

Pyrite Concentration

In addition to flotation, two other concentration methods were tried for the production of a precious metal-bearing pyrite concentrate, viz., (1) jigging at minus 10 mesh followed by flotation from the reground jig tailing and (2) tabling at a coarse grind (ca 30% minus 200 mesh).

* From Mines Branch Investigation Report IR 72-20 by A. E. Johnson.

Pyrite flotation was tried with the ore ground to 31%, 40% and 56% minus 200 mesh. Other variations in test procedure were the addition of soda ash to the grind and conditioning with copper sulphate prior to flotation.

Amalgamation

A sample of jig concentrate was ground and subjected to amalgamation. The object of the test was to determine what proportion, if any, of the precious metals was present in the form of free grains of native gold and silver.

Cyanidation

Samples of pyrite concentrate were cyanided both in the new state and after roasting. In order to ensure a complete roast, roasting was continued for $6\frac{3}{4}$ hours with the temperature raised to 650° C during the last hour.

Test Data

Screen analyses, details of test procedure and metallurgical balances for jigging, flotation and cyanidation tests are given in the appendix.

EVALUATION AND DISCUSSION OF RESULTS

Pyrite Concentration

Jigging and Flotation

Jigging of the ore at the minus 10-mesh size followed by flotation of the tailing ground to 31% minus 200 mesh gave significantly higher gold and silver recoveries than could be obtained by straight flotation or by tabling the ground ore. Results for jigging and flotation are given in Table 3.

TABLE 3

Results of Jigging and Flotation (Test 1)

Product	Wt %	Assays* Au	Ag	Fe	S	SiO_2	Distribution % Au	Ag	S
Jig conc	15.7	1.73	3.16	44.27	50.47	6.39	59.1	60.6	76.4
Flot conc	5.1	2.47	4.12	40.71	45.71	6.37	27.1	25.4	22.3
Flotation tail	79.2	0.08	0.15		0.18		13.8	14.0	1.3
Feed (calcd)	100.0	0.46	0.82		10.39		100.0	100.0	100.0
Jig + flot conc	20.8	1.91	3.39	43.40	49.31	6.38	86.2	86.0	98.7

* From Internal Reports MS-AC-72-65 and 86, expressed in per cent, except Au and Ag which are in oz/ton.

Flotation

Table 4 compares results of flotation obtained at different grinds and with other variations in procedure.

TABLE 4

Comparison of Results for Flotation

Test No	Feed	Grind % -200 m	Remarks	Rougher conc Assays Au	Ag	S	Flot tailing Assays* Au	Ag	S
1A	Jig tail	31	Soda ash to grind	2.46	4.29	44.96	0.08	0.15	0.19
1B	" "	31		2.48	3.95	46.45	0.08	0.14	0.16
2A	Ore	56		**			0.11	0.25	0.38
2B	"	56	Conditioned with $CuSO_4$	1.86	3.11	47.76	0.11	0.23	0.41
2C	"	40					0.11	0.22	0.30
3	"	31		1.78	3.31		0.11	0.24	

* From Internal Reports MS-AC-72-65, 79 and 171, expressed in per cent, except Au and Ag which are in oz/ton.
** Separate assays not available for each concentrate as they were combined for cleaning.

As can be seen from the above comparison, none of the test variables had any significant effect on flotation results.

Note that the precious metal content of the concentrate floated from the jig tailing was higher for the same sulphur content than the concentrates floated from the ore. However, when this concentrate was combined with the jig concentrate (See Table 3) the resultant precious metal content was similar to that of the flotation concentrates produced from the ore.

An attempt to upgrade rougher flotation concentrates by cleaning was unsuccessful (See Test 2 in appendix). Although sulphur content in the cleaner concentrate increased to 51.4% from 47.8% and the silica content decreased to 1.5% from 7.1%, there was no accompanying increase in silver and gold content. This would seem to indicate that some of the precious metals are present as free telluride particles liberated from the pyrite and that these particles have a tendency to be rejected to the tailing during the cleaning operation.

Practically all of the pyrite in the ore was recovered by flotation in Test 2 (97.4% sulphur recovery) as against gold and silver recoveries of only 83.1% and 79.5% respectively. This is further evidence indicating that slow-floating, free telluride particles are being liberated from the pyrite. The higher precious metal recoveries obtained by employing a combination of jigging and flotation can be attributed to the fact that about 3/4 of the pyrite was recovered by jigging as large minus 10-mesh grains thereby minimizing the extent of liberation of the tellurides from pyrite.

Precious Metal Losses in Flotation Tailing

Table 5 gives the distribution of gold and silver losses in the various size fractions of the flotation tailing from Test 3.

TABLE 5

Distribution of Precious Metal Losses in Various
Size Fractions of Flotation Tailing (Test 3)

Tyler mesh size	Wt %	Assays,oz/ton*		Distribution %	
		Au	Ag	Au	Ag
+ 48	30.6	0.11	0.15	28.5	19.1
+ 100	29.8	0.07	0.11	17.6	13.7
+ 200	9.0	0.05	0.08	3.8	3.0
+ 400	13.2	0.04	0.06	4.5	3.3
- 400	17.4	0.31	0.84	45.6	60.9
Total	100.0	0.12	0.24	100.0	100.0

* From Internal Report MS-AC-72-177

Despite the very coarse grind employed in Test 3, about 46% of the gold and 61% of the silver losses in the flotation tailing were distributed in the minus 400-mesh fraction, which for reasons given previously, would be in the form of slimed tellurides. The extreme friability of the tellurides would explain the difficulty experienced in recovering these minerals by flotation.

Tabling

The results of a table test at a coarse grind are given in Table 6. In this test two 2000-gram lots of minus 10-mesh ore were ground in a 7 x 14 lab rod mill for 15 minutes at 65% solids (screen analysis of table tailing, 29% minus 200 mesh). The ground ore was then run over a 2-ft x 4-ft diagonal deck Deister lab table.

TABLE 6

Results of Tabling (Test 4)

Product	Wt %	Assays, oz/ton*		Distribution %	
		Au	Ag	Au	Ag
Table conc	18.42	1.56	2.77	57.9	56.9
Table midds	15.85	0.28	0.47	8.9	8.3
Table tailing	65.73	0.25	0.48	33.2	35.0
Feed (calcd)	100.0	0.50	0.90	100.0	100.0

* From Internal Report MS-AC-72-137

From the above results, it can be seen that tabling was not as effective as flotation for concentrating the pyrite. Gold and silver recoveries in the table concentrate were lower by about 22% and 21% respectively than those obtained by flotation at the same grind (Test 3).

Amalgamation of Jig Concentrate

Table 7 gives the results of amalgamating the jig concentrate. In this test a 200-gram sample of jig concentrate from Test 1 was finely ground and amalgamated for 1 hour with new mercury.

TABLE 7

Results of Amalgamation of Jig Concentrate

Product	Wt %	Assays,oz/ton*		Distribution %	
		Au	Ag	Au	Ag
Amalgam**	-	0.06	0.23	3.5	7.3
Amalgam residue	100.0	1.67	2.93	96.5	92.7
Feed (jig conc)	100.0	1.73	3.16	100.0	100.0

* From Internal Report MS-AC-72-75

** Assays obtained by difference and expressed as oz/ton feed.

Only a small amount of precious metals was recovered by amalgamation. It can, therefore, be concluded that the proportion of the gold and silver present in the ore in the native state is insignificant.

Cyanidation of Pyrite Concentrate

A comparison of results obtained by cyaniding raw, finely reground pyrite concentrate and calcines from a complete roast of the concentrate is given in Table 8.

TABLE 8

Comparison of Results for Cyanidation of Pyrite Concentrate

Test No	Feed	Product	Wt %	Assays, oz/ton*		Distribution %	
				Au	Ag	Au	Ag
1	Raw conc, Test 2	Pregnant soln**	-	1.33	2.20	70.7	70.5
		Residue	100.0	0.55	0.92	29.3	29.5
		Feed	100.0	1.88	3.12	100.0	100.0
2	Calcines	Loss in roasting**	32.8	-	0.10	-	3.3
		Pregnant soln**	-	1.64	1.12	87.1	35.8
		Residue	67.2	0.24	1.90	12.9	60.9
		Feed	100.0	1.88	3.12	100.0	100.0

* From Internal Report MS-Ac-72-97

** Assays obtained by difference and expressed as oz/ton feed.

Roasting of the pyrite concentrate prior to cyanidation results in an appreciable increase in gold extraction over that obtained by cyaniding the raw concentrate. Note, however, that roasting results in a much lower extraction of silver.

CONCLUSIONS

The pyrite in the ore can be readily concentrated by jigging at minus 10 mesh followed by flotation of the jig tailing reground to about 30% minus 200 mesh. However, precious metal concentration was not as great as that of the pyrite. For a sulphur recovery of 98.7% in the combined jig and flotation concentrates the gold and silver recoveries were 86.2% and 86.0% respectively. The lower precious metal recoveries were attributed to the liberation of some of the tellurides from the pyrite and their subsequent loss in the tailing as fine slimed particles.

Tabling at a coarse grind was not an effective method for concentrating the pyrite. Precious metal recoveries in the table concentrate were about 20% lower than those obtained by flotation.

For optimum gold extraction it was necessary to roast the pyrite concentrate prior to cyanidation. Silver extraction was adversely affected by roasting but this loss was greatly offset by the gain in gold extraction.

Assuming a free market price of \$100/oz for gold and \$2.50/oz for silver the gross value of precious metal production from this ore would be as follows:

(1) <u>Shipment of Jig + Flotation Concentrate to Smelter:</u>

Value of gold = 100.00 x 0.41 x 0.862 = \$ 35.34

Value of silver = 2.50 x 0.75 x 0.86 = 1.61

Total = 36.95/ton ore

(2) <u>Roasting and Cyanidation of Jig + Flotation Concentrate:</u>

Value of gold = 35.34 x 0.871 = \$ 30.78

Value of silver = 1.61 x 0.358 = 0.58

Total = 31.36/ton ore

Difference (1) - (2) = \$36.95 - 31.36 = \$5.59/ton ore

or $\frac{5.59}{0.208}$ = \$26.90 /ton concentrate.

To the \$26.90 per ton in favour of shipping the pyrite concentrate to a smelter should be added roasting, cyaniding and other costs associated with the production and sale of precious metal bullion. This amount should then be adequate enough to cover low-cost water freight (\$8/ton), treatment and other charges. Even at the higher \$42/ton freight rate, shipping the concentrate to a smelter might be preferable to roasting and cyanidation of the concentrate and the subsequent production of precious metal bullion. This is because the high capital cost of a treatment plant would not be warranted for a 25 ton/day short-term operation. Also, experienced and skillful operators would be required to ensure the success of this scheme.

ACKNOWLEDGEMENTS

The authors wish to acknowledge the assistance of the following members of the Mineral Sciences Division: Dr. A. E. Johnson who carried out the mineralogical examination of the ore, Mr. G. Mason who did the spectrographic analysis of the head sample, Messrs P. E. Moloughney and J. A. Graham who did the fire assaying and Messrs D. Cumming, J. Cloutier, G. Richardson, B. Kobus and J. Hole who carried out the chemical analyses. The jigging, amalgamation, tabling and cyanidation tests were performed by Mr. J. C. Banks, technician with the Mineral Processing Division.

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APPENDIX

Screen Analyses

Tyler Mesh Size	Weight % Retained				
	Flotation Tailing				Table Tail
	Test 1A	Test 2A	Test 2C	Test 3	Test 4
+ 48	11.7	-	0.3	30.6	29.8
+ 65	17.3	0.2	5.5	29.8	17.4
+ 100	20.5	5.9	23.6	*	11.0
+ 150	11.0	19.6	17.5	*	6.7
+ 200	8.8	18.8	13.0	9.0	6.2
- 200	30.7	55.5	40.1	30.6	28.9
Total	100.0	100.0	100.0	100.0	100.0

* Not determined

Abbreviations Used in Flotation Test Reports

RM Rod mill

Xan Potassium ethyl xanthate

DF 250 Dowfroth 250 - Dow Chemical Co. frother.

JIGGING TEST REPORT

Sample: Kleanza Mines Ltd.

Test No: 1

Object of test:

Preliminary test on minus 10 mesh crushed ore

Feed rate, g/hr: 2000

Date: March 8, 1972

Test by: J. C. B.

OPERATING CONDITIONS

Unit: Denver lab Jig No. 1M

Speed, RPM: 250

Stroke, in.: 3/16

Supporting Screen: 8 mesh

Water feed rate, cc/min:

Top: 400

Bottom: 1000

Ragging:

Type: steel shot

Size in: 1/8 and 3/16 (1:1)

Weight, g: 71

METALLURGICAL BALANCE

Product	Weight		Assays		Distribution %	
	g	%	Au	Ag	Au	Ag
Jig conc	622.7	15.72	1.73	3.16	59.3	59.7
Jig bed	171.5	4.33	0.43	0.74	4.1	3.8
Jig tailing	3167.3	79.95	0.21	0.38	36.6	36.5
Feed (calcd)	3961.5	100.00	0.46	0.83	100.0	100.0

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 1	SAMPLE: Kleanza Mines Ltd.	DATE: March 8, 1972.
OBJECT OF TEST: Jigging + Flotation - Overall Metallurgical Balance		CHARGE: 4000 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton									
Jigging - see jig report														
Pyrite flotation - see Tests 1A and 1B														

PRODUCT	WT %	ANALYSIS *							DISTRIBUTION %				
		Au	Ag	Fe	S	SiO_2			Au	Ag		S	
Jig conc	15.72	1.73	3.16	44.27	50.47	6.39			59.1	60.6		76.4	
Flot conc A	2.52	2.46	4.29	40.33	44.96	10.12			13.5	13.2		10.9	
Flot conc B	2.54	2.48	3.95	41.09	46.45	2.65			13.6	12.2		11.4	
Flot tailing A	39.62	0.08	0.15		0.19				6.9	7.2		0.7	
Flot tailing B	39.60	0.08	0.14		0.16				6.9	6.8		0.6	
Feed (calcd)	100.00	0.46	0.82						100.0	100.0		100.0	
Combined jig + flot conc	20.78	1.91	3.39	43.40	49.31	6.38			86.2	86.0		98.7	

REMARKS: * In this and all subsequent reports analysis is expressed in per cent, except Au and Ag which are in oz/ton.

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 1A SAMPLE: Kleanza Mines Ltd. 1/2 split of Test 1 jig tailing

DATE: March 10, 1972
CHARGE: 1456 g
TESTED BY: A. S.

OBJECT OF TEST: To float a pyrite conc from jig tailing

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton Na_2CO_3	xan	DF250
Grinding	15*	65		7 x 14 RM	1.0		
Conditioning	2		8.5	1000-g cell		.014	
Pyrite rougher							
Stage 1	1/2						0.027
" 2	1/2					0.014	
" 3	1					0.014	
" 4	1					0.014	

PRODUCT	WT %	Analysis Au	Ag	Fe	S	SiO_2	Distribution % Au	Ag	S
Flot rougher conc	5.99	2.46	4.29	40.33	44.96	10.12	66.2	64.6	93.8
" " tail	94.01	0.08	0.15		0.19		33.8	35.4	6.2
Feed (calcd)	100.0	0.22	0.40		2.87		100.0	100.0	100.0

REMARKS: * Screen analysis of rougher tailing, 31% minus 200 mesh

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 1B	SAMPLE: Kleanza Mines Ltd. 1/2 split of Test 1 jig tailing	DATE: March 10, 1972
OBJECT OF TEST: As in Test 1A but without soda ash added to grinding		CHARGE: 1482 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton									
						Xan								
Grinding	15	65		7 x 14 RM										
Conditioning	2		7.2	1000-g cell		0.014								
Pyrite rougher -														
as in Test 1A														

PRODUCT	WT %	ANALYSIS							DISTRIBUTION %				
		Au	Ag	Fe	S	SiO_2			Au	Ag	S		
Flot rougher conc	6.01	2.48	3.95	41.09	46.45	2.65			66.5	64.3	94.9		
" " tail	93.99	0.08	0.14		0.16				33.5	35.7	5.1		
Feed (calcd)	100.0	0.22	0.37		2.94				100.0	100.0	100.0		

REMARKS:

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 2	SAMPLE: Kleanza Mines Ltd. -10 mesh ore	DATE: March 23, 1973
OBJECT OF TEST: To float a pyrite conc from the ore with the addition of copper sulphate and fineness of grind as variables.		CHARGE: 3 x 2000 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton									
						DF250								
Grinding — See sheets 1,2,3														
Conditioning — See sheets 1,2,3														
Pyrite roughers — See sheets 1,2,3														
Pyrite cleaner	2			1000-g cell		0.02								

PRODUCT	WT %	ANALYSIS Au	Ag	S	SiO_2	Fe	Te		DISTRIBUTION % Au	Ag	S		
Pyrite conc	19.7	1.88	3.12	51.39	1.54	45.44	0.034		73.7	69.7	91.8		
Pyrite cleaner tail	2.8	1.68	3.08	22.23	45.94	21.79			9.4	9.8	5.6		
Pyrite rougher tail A	25.6	0.11	0.25	0.38					5.6	7.3	0.9		
" " " B	25.8	0.11	0.23	0.41					5.6	6.7	1.0		
" " " C	26.1	0.11	0.22	0.30					5.7	6.5	0.7		
Feed (calcd)	100.0	0.50	0.88	11.03					100.0	100.0	100.0		
Pyrite ro conc (calcd)	22.5	1.86	3.11	47.76	7.07	42.50			83.1	79.5	97.4		

REMARKS: Pyrite rougher concentrates from Tests A, B and C combined for cleaning.

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 2A	SAMPLE: Kleanza Mines Ltd. -10 mesh ore	DATE: March 23, 1973
OBJECT OF TEST: To try a 45 min grind		CHARGE: 2000 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton									
					Xan	DF250								
Grinding	45*	65		7 x 14 RM										
Conditioning	2		6.7		0.02	0.02								
Pyrite rougher														
Stage 1	½													
" 2	½				0.01									
" 3	½				0.01									
" 4	½				0.01									
" 5	1				0.01									
" 6	1				0.01									

PRODUCT	WT %	ANALYSIS							DISTRIBUTION %					

REMARKS: Light, clean frother - barren at end of rougher float

* Screen analysis of rougher tailing, 56% minus 200 mesh.

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 2B	SAMPLE: Kleanza Mines Ltd. - 10 mesh ore	DATE: March 23, 1973.
OBJECT OF TEST: To try conditioning with copper sulphate prior to pyrite flotation		CHARGE: 2000 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton			
					$CuSO_4$	Na_2CO_3	Xan	DF250
Grinding	45	65		7 x 14 RM				
Conditioning	10		6.9		0.25	1.0		
Pyrite rougher								
Stage 1	1/2						0.04	0.02
" 2	1/2						0.02	
" 3	1/2						0.02	
" 4	1/2						0.04	
" 5	1						0.04	

PRODUCT	WT %	ANALYSIS							DISTRIBUTION %				

REMARKS: Froth appeared to be depressed despite high amounts of xanthate added
- effect of copper sulphate at low pH?

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 2C	SAMPLE: Kleanza Mines Ltd. - 10 mesh ore	DATE: March 23, 1973
OBJECT OF TEST: To try a 30 min grind		CHARGE: 2000 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton									
					Xan	DF250								
Grinding	30*	65		7 x 14 RM										
Conditioning	2		6.4		0.02	0.02								
Pyrite rougher - as in Test 2A														

PRODUCT	WT %	ANALYSIS							DISTRIBUTION %					

REMARKS: Active, well-mineralized and bright froth.
* Screen analysis of rougher tailing, 40% minus 200 mesh.

MINES BRANCH FLOTATION TEST REPORT

TEST NO. 3	SAMPLE: Kleanza Mines Ltd. - 10 mesh ore	DATE: May 11, 1972
OBJECT OF TEST: Pyrite flotation at a coarse grind (15 min) to compare with Tests 2A and 2C		CHARGE: 2000 g
		TESTED BY: A.S.

OPERATION	Time min	% Solids	pH	Unit used	Reagents, lb per ton: Xan	DF 250
Grinding	15*	65		7 x 14 RM		
Conditioning					0.02	0.02
Pyrite rougher						
Stage 1	1/2					
" 2	1/2				0.02	
" 3	1				0.02	
" 4	1				0.02	
" 5	1				0.02	

PRODUCT	WT %	ANALYSIS Au	Ag	DISTRIBUTION % Au	Ag
Pyrite rougher conc	19.90	1.78	3.31	80.1	77.4
" " tail	80.10	0.11	0.24	19.9	22.6
Feed (calcd)	100.00	0.44	0.85	100.0	100.0

REMARKS: * Screen analysis of rougher tailing, 31% minus 200 mesh.

MINES BRANCH
CYANIDATION TEST REPORT

Test No: 1	Sample: Kleanza Mines Ltd., Test 2 pyrite conc
Test By: JCB	

OBJECT OF TEST: To try cyanidation of raw, reground pyrite conc.

TEST DATA

Date	Time hr	NaCN, lb/ton Solution		Lime lb/ton Solution		Charge:	
		Added	Titrn	Added	Titrn	Solids, g	553
						Water, cc	1000
Apr.4/72	0	1.0		1.0		Dilution (Water : Solids)	1.8:1
" "	1	0.8	0.15	1.2	0.05		
" "	4	0.4	0.70	0.8	0.30	Nominal Solution Strength,	
" "	6.75	0.2	1.0	0.8	0.50	lb/ton Solution:	
Apr.5/72	23.5		1.05	1.0	0.3	NaCN	1.0
" "	26.5		0.95	0.8	0.55	Lime	1.0
" "	30.75	0.1	0.9	0.8	0.60	Reagent Consumption,	
Apr.6/72	48		0.9		0.42	lb/ton Solids Cyanided:	
Total		2.5		6.4		NaCN	2.9
						Lime	10.8
						Reducing power*	340

METALLURGICAL BALANCE

Product	Wt %	Assays, oz/ton		Distribution, %	
		Au	Ag	Au	Ag
Pregnant solution	–	1.33	2.20	70.7	70.5
Cyanide residue	100.0	0.55	0.92	29.3	29.5
Feed (assay)	100.0	1.88	3.12	100.0	100.0

Remarks: Pyrite concentrate reground at 53% solids for 30 min in an 8 - in.- dia Abbé porcelain mill with full charge of steel balls. (Screen analysis on Sheet 2).

* No. of cc of 0.1N $KMnO_4$ solution required to oxidize all reducing agents present in 1000 cc of pregnant soln.

Cyanidation Test Report

Test No. 1

Screen Analysis
of Reground Pyrite Concentrate

Tyler Mesh Size	Wt % retained
+ 200	0.7
+ 325	7.1
+ 400	5.5
+ 500	21.2
- 500	65.5
Total	100.0

MINES BRANCH
CYANIDATION TEST REPORT

Test No: 2	Sample: Kleanza Mines Ltd. Test 2 pyrite conc
Test By: J.C.B.	

OBJECT OF TEST: To try roasting of pyrite conc followed by cyanidation of the calcines

TEST DATA

Date	Time hr	NaCN, lb/ton Solution		Lime lb/ton Solution		Charge:	
		Added	Titrn	Added	Titrn	Solids, g	300
						Water, cc	1000
Apr.4/72	0	1.0		0.5		Dilution (Water : Solids)	3.3: 1
" "	1	0.4	0.75	0.4	0.1		
" "	4		1.05	0.4	0.2	Nominal Solution Strength,	
" "	6.75	0.1	1.0	0.2	0.4	lb/ton Solution:	
Apr.5/72	23.5		1.0		0.5	NaCN	1.0
" "	26.5					Lime	0.5
" "	30.75		1.05	0.3	0.4	Reagent Consumption,	
Apr.6/72	48		1.02		0.5	lb/ton Solids Cyanided:	
Total		1.5		1.8		Na CN	1.6
						Lime	4.3
						Reducing Power	32

METALLURGICAL BALANCE

Product	Wt %	Assays, oz/ton		Distribution, %	
		Au	Ag	Au	Ag
Loss in roasting *	32.78	-	0.10	-	3.3
Pregnant solution	-	1.64	1.12	87.1	35.8
Cyanide residue	67.22	0.24	1.90	12.9	60.9
Feed (pyrite conc)	100.00	1.88	3.12	100.0	100.0
Pregnant solution **	-	2.49	1.66	87.4	37.0
Cyanide residue	100.0	0.36	2.83	12.6	63.0
Feed (calcines)***	100.0	2.85	4.49	100.0	100.0

Remarks: * Overall metallurgical balance, assays of roasting loss, pregnant solution and cyanide residue obtained by difference and expressed as oz/ton of feed.

** Metallurgical balance for cyanidation only, assays of pregnant solution obtained by difference and expressed as oz/ton of feed.

*** % sulphur, 0.15

Cyanidation Test Report

Test No. 2

Roasting Procedure

Elapsed time hr	Elapsed time min	Temp. °C	Remarks
Start		room	Charge in, controller set at 475°, door open 2 in.
1	15	475	
4	15	475	Controller set at 650°
5	45	650	
6	45	650	Shut power off, calcine cooled in furnace.

Calcine Treatment Before Cyanidation

(1) After cutting out a small sample for analysis, calcines ground for 10 min in an 8 in.-dia porcelain mill with 1/2 charge of steel balls, 500 cc water and 1 gram lime.

(2) Ground calcines were filtered, and the filter cake was washed once with cold water.

(3) Washed cake repulped in agitation bottle to make up a volume of 1000 cc.