Department of Energy, Mines and Resources
Ministère de l'Énergie, des Mines et des Ressources

CANADA

MINERAL PROCESSING DIVISION

Mines Branch
Direction des mines

File Number
Nº à rappeler

40 Lydia Street,
Ottawa 1, Ontario.
May 6, 1968

Mr. B. S. Imrie, P. Eng.,
General Manager,
Mount Nansen Mines Limited,
420-475 Howe Street,
Vancouver 1, B.C.

Dear Mr. Imrie:

Herewith a progress report on the status of the investigation carried out on the gold-silver ore from your Yukon property.

Two lots of ore were received, one containing marked individual samples of drill core and the other containing a bulk sample designated "Composite B". This report concerns only the latter sample.

The composite ore was crushed to -10 mesh, carefully mixed and riffled into 2000 gram test samples. One of these test samples was selected for a semi-quantitative spectrochemical analysis, a head sample analysis, a screen analysis and a mineralogical examination. The results of this work may be seen in the following tables. A mineralogical report will be sent to you as soon as that work has been finished.

Table 1

Spectrochemical Analysis

Principal constituent	-	Si AL
$>$ 2.0 percent	-	As
$>$ 1.0 "	-	Fe
$>$ 0.1 "	-	Sb, Ca, Mg, Cr, Ti, Ni
$>$ 0.01 "	-	Cu, Ag, Co

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Table 2

Head Sample Analysis

Gold (Au)	- 0.415 oz./ton
Silver (Ag)	-17.405 " "
Lead (Pb)	- 1.32 per cent
Zinc (Zn)	- 0.40 " "
Copper (Cu)	- 0.08 " "
Antimony (Sb)	- 0.43 " "
Arsenic (As)	- 1.91 " "
Iron (Sal Fe)	- 4.62 " "
Sulphur (tot S)	- 1.16 " "
Insoluble	-68.32 " "

Table 3

Screen Test Analysis

Mesh Size	Weight %	Assays		Distribution %	
		Au	Ag	Au	Ag
+14 mesh	27.4	0.265	12.645	18.3	20.9
+20 "	20.0	0.290	13.610	14.7	16.4
+28 "	12.1	0.285	15.935	8.7	11.6
+35 "	8.1	0.270	17.030	5.5	8.3
+48 "	5.8	0.450	19.300	6.6	6.8
+65 "	5.0	0.370	20.480	4.7	6.2
+100 "	3.8	0.525	21.575	5.0	5.0
+150 "	3.3	0.650	23.770	5.4	4.7
+200 "	2.8	0.850	25.480	6.0	4.3
-200 "	11.7	0.845	22.215	25.1	15.8
Head (calcd)	100.0	0.40	16.55	100.0	100.0

As, Fe and S analyses have not yet been received.

A series of flotation tests was done at different grinds at an initial pH of about 7.8 using Na_2CO_3, $CuSO_4$, Aero Promotor 301 and pine oil. These were preliminary tests and show that a grind between 85% and 90% -200 mesh will be necessary in order to ensure the best recovery of the gold and silver. The results of these tests were as follows:

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Table 4

Results of Flotation Tests 6, 7, 8, 9

Test No.	Grind % -200 m	Product	Weight %	Assays Au	Assays Ag	Distribution % Au	Distribution % Ag
6	65.0	Ro flot conc	28.7	1.06	54.53	79.5	89.7
		Ro Flot tail	71.3	0.11	2.53	20.5	10.3
		Head (calcd)	100.0	0.38	17.45	100.0	100.0
7	70.0	Ro flot conc	26.6	1.185	57.93	81.1	90.6
		Ro flot tail	73.4	0.10	2.185	18.9	9.4
		Head (calcd)	100.0	0.39	17.01	100.0	100.0
8	85.0	Ro flot conc.	35.5	0.921	45.34	83.5	93.4
		Ro flot tail	64.5	0.10	1.785	16.5	6.6
		Head (calcd)	100.0	0.39	17.24	100.0	100.0
9	90.0	Ro flot conc	40.2	0.837	39.54	83.6	93.8
		Ro flot tail	59.8	0.11	1.75	16.4	6.2
		Head (calcd)	100.0	0.40	16.94	100.0	100.0

Test 10

As previously stated it appears from the above results that a grind between 85% and 90% -200 mesh will be necessary in order to ensure adequate liberation of the gold and silver minerals.

This test duplicated Test 9 with some modification in the amounts of reagents used. The total flotation time was 28 minutes. The following tables shows the detailed flotation scheme used as seven separate concentrates were removed to determine the flotation rate. The results of this test are shown in Table 6 and 7.

Table 5

Flotation Scheme Test 10

OPERATION	Time Min.	% S	pH	Reagents - lbs/ton Na_2CO_3	$CuSO_4$	Aero 301	P.O.		REMARKS
Grind	40			10	0.5				2000 gram ground to 90% -200 mesh
Condition	3	35	8.2			0.05			
Flot conc 1	1						0.04		
" " 2	1						0.03		
" " 3	2								Flotation at 2400 rpm in 1000 gram cell
Condition	3		8.4			0.02			
Flot conc 4	4						0.03		
" " 5	4								
Condition	3		8.4			0.02	0.015		
Flot conc 6	8								
Condition	3		8.5			0.04			
Flot conc 7	8						0.03		

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Table 6

Results of Flotation Test 10

PRODUCT	Wt. %	Assays Au.		Ag	Dist. % Au	Ag
Flot conc. 1	3.4	1.95		235.49	17.2	47.9
" " 2	4.2	2.31		71.98	25.1	18.1
" " 3	4.1	1.78		49.39	19.0	12.1
" " 4	4.7	0.84		27.38	10.2	7.7
" " 5	4.6	0.38		10.89	4.5	3.0
" " 6	7.5	0.26		7.26	5.2	3.2
" " 7	7.6	0.20		4.30	3.9	2.0
Flot Tail	63.9	0.09		1.57	14.9	6.0
Head (calcd)	100.0	0.38		16.72	100.0	100.0

Table 7

Results of Size Analysis of Flot Tail from Test 10

PRODUCT	Wt. %	Assays Au		Ag	Dist. % Au	Ag
+ 100 Mesh	0.6					
+ 150 "	5.0	0.045		1.395	2.6	5.0
+ 200 "	12.5	0.05		1.57	6.6	12.6
+ 56 Microns	4.2	0.097		2.935	4.3	7.9
+ 40 "	18.9	0.07		1.53	14.0	18.6
+ 28 "	15.0	0.085		1.335	13.5	12.9
+ 20 "	12.1	0.10		1.30	12.8	10.1
+ 14 "	7.6	0.10		1.33	8.0	6.5
+ 10 "	4.5	0.129		1.41	6.1	4.1
- 10 "	19.6	0.155		1.77	32.1	22.3
Head (calcd)	100.0	0.095		1.55	100.0	100.0

The size analysis of the flotation tailing shows 32.1% of the gold and 22.3% of thejsilver in the -10 micron fraction. It is doubtful whether one can recover an appreciable amount of this gold and silver.

An attempt is now being made based on the results of Test 10 to produce a high grade gold-silver concentrate by repeated cleaning steps.

Further analysis of the concentrates from Test 10 for Sb, As and S will determine whether or not it would be best to produce a high grade concentrate from the first four concentrates followed by the flotation of a scavenger concentrate.

I shall endeavour to keep you informed as to the result of additional work as it becomes available.

Yours very truly,

T. F. Berry

T. F. Berry,
Non-Ferrous Minerals Section

TFB/am

Department of Energy, Mines and Resources
Ministère de l'Énergie, des Mines et des Ressources

CANADA

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MINERAL PROCESSING DIVISION

40 Lydia Street,
Ottawa 1, Ontario.
June 18, 1968

19.VI.68

Mr. B. S. Imrie, P. Eng.,
Manager,
Mount Nansen Mines Limited,
420-475 Howe Street,
Vancouver 1, B.C.

Dear Mr. Imrie: Progress Report No. 2

In the tests included in this report, two grinds were investigated: a coarse grind; 75%-200 mesh and a fine grind; 90%-200 mesh. There appears to be no point in grinding to the finer size since almost the same overall recovery of gold and silver was obtained at a grind of 75%-200 mesh.

In Test 20, in which the ore was ground to 75%-200 mesh, the rougher flotation concentrate was ground and refloated. There was no increase in the grade of the final concentrate, and the conclusion is that regrinding is of no benefit. Perhaps if the same rougher recovery can be obtained at a grind coarser than 75%-200 mesh, regrinding may be advantageous. This is something that will be determined.

The natural pH of this ore is about 6.6. The use of soda ash to ensure an alkaline float and copper sulphate does not appear to be necessary since about the same rougher recovery was obtained in the acid as in the alkaline circuit. Potassium amyl Xanthate (Z-6) along with Aero Xanthate 301 with perhaps a small addition of Aerofloat 208 promoter gave as good results as any flotation scheme found. These are standard reagents which present no feeding problems to a flotation circuit. Rather large amounts of pine oil frother were used in all of the tests but the substitution of Dow froth 250 for pine oil may result in a reduction of the amount of frother necessary.

The flotation tailings from Test 18 - (grind-75%-200 mesh) and Test 19 - (grind -90%-200 mesh) were cyandided for 24 hr and 48 hr. These results may be seen in Test 22. Regardless of the grind or the cyanidation time, the overall recovery by flotation and cyanidation of the gold and silver in these tests was almost identical. It follows from the results of these tests that if the same overall recovery from flotation and cyanidation can be obtained at a coarser grind, i.e. - 65%-200 mesh, then this is the procedure that should be followed, even though this may mean regrinding of a rougher concentrate to obtain the best concentrate grade. Also, since cyanidation of the flotation tailing appears to be advantageous, the flotation time may be reduced, and still result in the same overall recovery. These are tests which must be tried before any final conclusions can be reached.

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FILE: METALLURGY

The final concentrates in Tests 18, 19 and 20 were analysed for arsenic and antimony. The removal of the arsenic by roasting would present no problem if it alone was present. However, the presence of antimony may make roasting a difficult operation. There are methods of driving off the antimony in roasting. One of these involves the addition of salt (NaCN) to thecharge. These tests will have to be done before any firm conclusions can be reached.

The sulphide samples (Heustis ore body) were examined mineralogically but nothing new was discovered, except the presence of coarse electrum in the -65+100 mesh fraction of a sink product which was concentrated on a micro-superpan.

Perhaps it would be advantageous to outline the scope of the ensuing investigations.

1. Investigate the recovery possibilities at a coarser grind, at the natural pH of the ore, and determine whether regrinding of the rougher concentrate is necessary to obtain the best concentrate grade.

2. Determine whether a reduction in the flotation time can be made without affecting the overall recovery by flotation and cyanidation.

3. Investigate the possibility of a reduction in the frother required by substituting other frothers for pine oil.

4. Investigate the roasting of the final concentrate to reduce the aresenic and antimony content.

The results of these tests will be sent to you as they become available.

Yours very truly,

T. F. Berry

T. F. Berry,
Non-Ferrous Minerals Section

TFB/am
Encs.

Test 11

2000 gram sample - ground to 90% -200 mesh.

Flotation Procedure

OPERATION	Time Min.	% S	pH	Reagents - lb/ton Feed Na_2CO_3	$CuSO_4$	301	208	Na_2S	404	P.O
Grind	40	65		10.0	0.5					
Condition	3	35	8.0			0.04	0.03			
Float	4									0.06
Float	4					0.02	0.02			0.04
Float	4		8.4	(end)		0.02	0.02			0.04
Scavenger										
Condition	4		8.4			0.02		0.5	0.04	
Float	6									0.04
Condition	2							0.5	0.02	
Float	6		9.3	(end)						0.04

Results of Test 11

PRODUCT	Wt. %	Assays Au	Ag	Distn. % Au	Ag
Float conc	26.6	1.10	56.63	76.3	89.2
Scavenger conc	9.0	0.29	7.70	6.9	4.1
Float tail	64.4	0.10	1.75	16.8	6.7
Head (calcd)	100.0	0.38	16.88	100.0	100.0

Remarks:

This test was a repeat of Test 10 using Aerofloat Promoter 208 with the Aero Xanthate 301 and an attempt to scavenge additional values using sodium sulphide and Aero Promoter 404.

Test 12

2000 gram sample - ground to 90% -200 mesh.

Flotation Procedure

OPERATION	Time Min.	% S	pH	Reagents lb/ton Feed Na_2CO_3	$CuSO_4$	301	P.O	Sod Sil
Grind	40	65		10.0	0.5			
Condition	3	35	8.1			0.05		
Float	4						0.04	
Condition	3					0.02		
Float	4						0.03	
Condition	3					0.02		
Float	4						0.03	
1st cleaner	5							0.5
2nd cleaner	3							0.5
Condition	3					0.06		
Float	16						0.06	
Condition	3					0.02		
Float	2						0.03	
1st cleaner	5							1.0
2nd cleaner	2							1.0

Results of Test 12

Product	Wt. %	Assays Au	Ag	Dist % Au	Ag
Float conc	8.7	2.44	144.92	57.1	75.9
2nd cl tail	3.2	0.965	38.92	8.3	7.5
1st " "	8.7	0.365	9.29	8.6	4.9
Scavenger conc	1.0	0.72	33.09	1.9	2.0
2nd cl tail	3.1	0.375	5.85	3.2	1.1
1st " "	11.2	0.18	3.68	5.4	2.5
Float tail	64.1	0.09	1.60	15.5	6.1
Head (calcd)	100.0	0.37	16.61	100.0	100.0

Remarks:

This flowsheet followed Test 10 insofar as reagents in the rougher stage were concerned, except that somewhat heavier reagent additions were used. The rougher and the scavenger concentrate were each cleaned twice with sodium silicate as a gangue depressant.

Test No. 13 - 2000 gram sample - ground to 90%-200 mesh

Test No. 14 - " " " " " 75%- " "

10 lb Na_2CO_3/ton in grind
0.3 lb Aero promoter 404 stage added
0.2 lb Aerofloat promoter 242 stage added
Flotation time 22 minutes

Test No. 15 - 2000 gram sample - ground to 90%-200 mesh

Test No. 16 - " " " " " 75%- " "

10 lb Na_2CO_3/ton in grind
0.3 lb. Aero promoter 425 stage added
0.25 lb Aerofloat 242 stage added
Flotation time 22 minutes

Results of Tests 13-14-15-16

Test No.	Product	Wt. %	Assays Au	Assays Ag	Distn.% Au	Distn.% Ag
13	Ro float conc	26.8	0.93	50.60	75.2	91.7
	Scavenger conc	9.2	0.20	3.76	5.5	2.3
	Float Tail	64.0	0.10	1.38	19.3	6.0
	Head (calcd.)	100.0	0.33	14.79	100.0	100.0
14	Ro float conc	24.9	1.04	58.30	76.0	90.8
	Scavenger conc	8.2	0.22	5.09	5.3	2.6
	Float tail	66.9	0.095	1.57	18.7	6.6
	Head (calcd.)	100.0	0.34	15.98	100.0	100.0
15	Ro float conc	25.2	1.23	62.19	78.4	91.5
	Scavenger conc	7.2	0.245	5.00	4.4	2.1
	Float tail	67.6	0.10	1.61	17.2	6.4
	Head (calcd.)	100.0	0.40	17.12	100.0	100.0
16	Ro float conc	23.1	1.26	66.09	78.1	90.7
	Scavenger conc	8.2	0.24	5.34	5.3	2.6
	Float tail	68.7	0.09	1.81	16.6	7.3
	Head (calcd.)	100.0	0.37	16.95	100.0	100.0

TEST 18

2000 gram sample ground to 75%-200 mesh.

Flotation Procedure

Operation	Time Min.	% S	pH	Reagents lb/ton Feed 301	Z-6	P.O
Grind	26½	65		0.1		
Condition	5	35	6.6		0.1	
Float	10					0.1
Condition	5		6.1	0.05	0.1	
Float	5					0.06
Condition	5		6.2	0.05	0.1	
Float	5					-
1st cleaner	7.5		6.1			0.04
2nd "	4					
3rd "	4					

Results of Test 18

Product	Wt. %	Assays Au	Ag	As	Sb	Dist. % Au	Ag
Final conc	9.5	2.67	145.83	15.09	2.33	64.3	81.4
3rd cl tail	0.7	0.955	24.56			1.7	1.0
2nd " "	1.7	0.52	15.57			2.2	1.6
1st " "	19.9	0.235	5.165			11.9	6.0
Float "	68.2	0.115	2.495			19.9	10.0
Head (calcd.)	100.0	0.39	17.02			100.0	100.0

Remarks:

Floated at the natural pH of 6.6 the rougher recovery was less at this grind than that obtained when flotation was carried out in an alkaline pulp.

Test 19

2000 gram sample ground to 90%-200 mesh

This was a repeat of Test 18 as far as flotation reagents were concerned. The finer grind was the only change.

Results of Test 19

Product	Wt. %	Assays Au	Ag	As	Sb	Distn % Au	Ag
Final conc	8.6	2.76	164.0	15.63	2.45	63.0	83.7
3rd cl tail	1.0	0.915	26.81			2.4	1.6
2nd " "	3.6	0.36	7.95			3.4	1.7
1st " "	25.8	0.195	3.835			13.3	5.6
Float tail	61.0	0.11	2.06			17.9	7.4
Head (calcd)	100.0	0.38	16.85			100.0	100.0

Remarks:

This finer grind did not reduce the gold and silver in the float tailing. As in Test 18 flotation was carried out at the natural pH of the ore.

Test 20

2000 grams ground to 75%-200 mesh.
Rougher concentrate ground and cleaned three times.
Flotation tail cyanided.

Flotation Procedure

Operation	Time Min	% S	pH	Reagents lb/ton Feed 301	Z-b	P.O
Grind	26½	65		0.1		
Condition	5	33	6.5		0.1	1.0
Float	7					
Condition	3		6.3		0.1	
Float	8					0.06
Condition	3		6.3		0.1	
Float	5					0.04
Filter				0.05	0.05	
Regrind	10					
Reg Ro Float	10		6.6		0.05	0.06
1st cl	5					
2nd cl	2.5					
3rd cl	2					

Results of Test 20

Product	Wt %	Assays Au	Ag	As	Sb	Dist % Au	Ag
Final conc	8.7	2.78	158.05	16.25	2.45	66.5	81.7
3rd cl tail	0.5	1.165	64.93			1.6	1.9
2nd " "	1.5	0.73	28.62			2.8	2.5
1st " "	0.4	0.375	10.00			0.4	0.2
Reg. Ro tail	25.0	0.175	4.02			12.0	6.0
Float	63.9	0.095	2.025			16.7	7.7
Head (calcd)	100.0	0.36	16.84			100.0	100.0

Remarks:

Floated at natural pH of 6.5. In an attempt to increase the rougher recovery a high pulp level was carried and the froth was pulled heavily.

Cyanidation Tests - Test 22

In the following table the results of cyanidation tests on the ground ore at two different grinds and in the flotation tailings from Tests 18 and 19 are reported. The solution strength in each test was maintained as near as possible at 1.0 lb NaCN and 1.0 lb CaO per ton of solution.

Feed	Grind % -200m	Agitation time hr	Consumption lb/ton feed NACN	CaO	Feed Feed Au	Ag	Residue Au	Ag	Extr * % Au	Ag
Ore	75	24	1.58	5.6	0.40	16.84	0.242	8.89	39.5	47.2
Ore	90	24	1.04	5.62	0.40	16.75	0.225	8.645	43.8	48.4
Ore	75	48	1.74	8.02	0.40	16.84	0.23	8.285	42.5	50.8
Ore	90	48	1.40	7.78	0.40	16.75	0.22	8.15	45.0	51.3
Flotation tailing Test 18 and 19										
18	75	24	0.56	2.40	0.115	2.495	0.04	0.92	65.2	63.1
18	75	48	0.94	8.30	0.115	2.495	0.04	0.90	65.2	63.9
19	90	24	0.50	7.62	0.11	2.06	0.04	0.965	63.6	53.2
19	90	48	1.62	8.14	0.11	2.06	0.04	0.82	63.6	60.2

* Calculated by difference.

			Au%	Ag%
Overall recovery:	Test 18 - by flotation		- 80.1	90.0
	by cyanidation	(24 hr)	- 13.0	6.3
	" "	(48 hr)	- 13.0	6.4
	Test 19 - by flotation		- 82.1	92.6
	by cyanidation	(24 hr)	- 11.4	3.9
	" "	(48 hr)	- 11.4	4.4

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Ministère de l'Énergie, des Mines et des Ressources

Mines Branch
Direction des mines

File Number
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40 Lydia Street,
Ottawa 1, Ontario,
July 25th, 1968.

Mr. B.S. Imrie,
Manager,
Mount Nansen Mines Limited,
420-475 Howe Street,
Vancouver 1, B.C.

Dear Mr. Imrie:

Progress Report No. 3

In our last letter to you (June 18), we indicated the scope of this continuing investigation.

As the following table shows, very little will be gained by grinding this ore finer than 55% to 60% -200 mesh.

Test No.	Initial Grind % - 200 mesh	Flot. Ro Recovery %	
		Au	Ag
25	55.0	81.0	90.4
24	65.0	80.9	90.3
19	75.0	80.1	90.0
18	90.0	82.1	92.2

At this grind (55% to 60% -200 mesh) a time of about 20 minutes will ensure the optimum flotation recovery of the gold and silver. When such treatment is augmented by the cyanidation of the flotation tailing an overall recovery of about 94% of the gold and 96% of the silver should be obtained (Test 29).

The results of Tests 26 to 29 inclusive are illustrated in Figure 1, in which the recoveries of gold and silver and the concentrate grades have been plotted against flotation time. The top line in each graph represents the overall recovery of gold and silver obtained by cyaniding the flotation tailings. The flotation and cyanidation reagent consumptions are shown for each flotation time.

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FILE: METALL.

Test 25 and Test 29 were identical in all respects except that in Test 29 the rougher flotation concentrate was ground for 10 minutes prior to cleaning three times. A comparison of these two tests shows that regrinding is beneficial, resulting in an increased grade with about the same recovery. Based on the results of all work done to date, regrinding of a rougher flotation concentrate followed by three stages of cleaning will result in a final concentrate assaying about 2.8 oz Au/tin and about 160 oz Ag/ton. A higher grade than this can not be expected from this ore.

The results of Test 32 are graphically shown in Figure 2. In this test successive concentrates were removed at 1, 3, 6, 10, 15 and 20 minutes of flotation time. The cumulative recoveries of gold and silver and the cumulative concentrate grades have been plotted. A deliberate attempt was made in this test to reduce the amounts of flotation reagents which had been used in other tests. Additionally Dowfroth 250 was substituted for pine oil.

In the following table a comparison of the amount of reagents used in two tests and the rougher recoveries of gold and silver obtained is shown. The flotation time in each test was 20 minutes and the grind was about 55% -200 mesh.

Reagents lb/ton feed	Test 29	Test 32
301	0.25	0.20
Z-6	0.40	0.15
Pine oil	0.30	-
DF-250	-	0.18
Recovery %		
Au	83.5	78.0
Ag	92.2	88.3

The reduced recovery in Test 32 may be accounted for by the greatly reduced amount of Z-6 used. Dowfroth 250 seems to be a better frother than pine oil from the point of view of stability and general froth control. Certainly there is no doubt that less of this frother will be required than if you use pine oil.

All of the tests shown in this report were done at the natural pH of the ore (about 6.6). When one compares these results with those reported in Progress Report No 2 in which an alkaline pulp was obtained, one must conclude that almost no benefit is to be derived by the use of soda-ash.

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In an operation in which a cyanidation circuit is to be employed to recover additional gold and silver it is sometimes advantageous to grind in cyanide thus providing a longer contact time. Such a process, using NaCN and lime in the primary grind may interfere with the flotation stage of concentration. Work now in progress involves this step in the investigation. If these reagents are not detrimental to the flotation recovery and concentrate grade then the flotation tailing filter can be omitted with this product going directly to a primary thickener and thence to agitation. The cyanided pulp would then be filtered with the filtrate returning to the primary thickener. The barren solution from the Merrill-Crowe unit would provide make-up solution for the primary grind and agitation.

A supplementary mineralogical report on the high-grade sample from the Heustis ore body is in the printing stage and should be received shortly.

As soon as additional results are available they will be sent to you.

Yours very truly,

T. F. Berry

T.F. Berry,
Non-Ferrous Minerals Section.

Test 24 - 2000 grams ground to 65%-200 mesh

Test 25 - 2000 grams ground to 55%-200 mesh
Same flotation procedure as Test 24

Flotation Procedure

Operation	Time Min.	% S	pH	Reagents lb/ton Feed 301	Z-6	P.O.
Grind	21	65		0.1		
Condition	5	35	6.5		0.1	
Float	6					0.1
Condition	3			0.05	0.1	
Float	6					0.06
Condition	3			0.05	0.1	
Float	7		6.4			0.06
1st cleaner	7.5					0.04
2nd "	4					
3rd "	2.5					

Results of Test 24

Product	Wt. %	ASSAYS Au	Ag	DIST % Au	Ag
Final Conc	10.0	2.46	132.85	64.1	80.8
3rd Cl Tail	0.6	1.248	46.58	2.0	1.7
2nd Cl Tail	2.0	0.49	11.37	2.6	1.4
1st Cl Tail	20.3	0.23	5.195	12.2	6.4
Float Tail	67.1	0.11	2.38	19.1	9.7
Head (Calcd)	100.0	0.38	16.44	100.0	100.0

Results of Test 25

Product	Wt. %	ASSAYS Au	Ag	DIST % Au	Ag
Final Conc	10.4	2.29	128.54	63.4	80.2
3rd Cl Tail	0.7	1.17	31.25	2.2	1.3
2nd " "	2.7	0.43	16.65	3.1	2.7
1st " "	18.1	0.255	5.73	12.3	6.2
Float "	68.1	0.105	2.355	19.0	9.6
Head (Calcd)	100.0	0.38	16.68	100.0	100.0

Tests 26, 27, 28, 29

Rougher Flotation Scheme

Operation	Time Min.	% S	pH	Unit	Reagents lb/ton 301	Z-6	P.O
Grinding	16.5	65		12" steel	0.1	-	-
Test 26							
Conditioning	5	35	6.4	1000 gm cell	-	0.1	
Flotation	6				-	-	0.12
Test 27							
Conditioning	5	35	6.4	1000 gm cell	-	0.1	
Flotation	6				-	-	0.12
Conditioning	3				0.05	0.1	
Flotation	4				-	-	0.06
Test 28							
Conditioning	5	35	6.4	1000 gm cell	-	0.1	
Flotation	6				-	-	0.12
Conditioning	3				0.05	0.1	
Flotation	4				-	-	0.06
Conditioning	3				0.05	0.1	
Flotation	5				-	-	0.06
Test 29							
Conditioning	5	35	6.3	1000 gm cell	-	0.1	-
Flotation	6				-	-	0.12
Conditioning	3				0.05	0.1	-
Flotation	4				-	-	0.06
Conditioning	3				0.05	0.1	-
Flotation	5				-	-	0.06
Conditioning	3				0.05	0.1	-
Flotation	5				-	-	0.06

Regrinding & Cleaning

Regrinding	10						
1st Cleaner	5			1000 gm cell	-	0.05	-
2nd "	3			500 gm cell	-	-	0.04
3rd "	2			250 gm cell	-	-	-

Tests 26, 27, 28 and 29

Results of Test 26 - 6 minutes

Product	WT. %	Assays Au	Assays Ag	Dist % Au	Dist % Ag
Final conc	6.2	2.59	174.76	43.0	67.2
3rd cl tail	0.6	2.08	38.51	4.5	1.4
2nd cl tail	1.2	1.405	27.715	4.5	2.0
1st cl tail	3.5	0.95	9.70	8.9	2.2
Float tail	88.5	0.165	4.95	39.1	27.2
Head (calcd)	100.0	0.37	16.17	100.0	100.0
1.Ro conc	11.5	1.98	103.29	60.9	72.8
2.Cy Residue		0.065	1.60	60.6	66.7 *
Overall Recovery (1 + 2)				84.6	90.0

Results of Test 27 - 10 minutes

Product	WT. %	Assays Au	Assays Ag	Dist % Au	Dist % Ag
Final conc	7.2	2.60	167.72	47.6	71.0
3rd cl tail	0.7	1.58	47.60	2.8	2.0
2nd " "	3.0	1.35	38.80	10.3	6.8
1st " "	7.2	0.44	6.465	8.1	2.7
Float "	81.9	0.15	3.64	31.2	17.5
Head (calcd)	100.0	0.39	17.01	100.0	100.0
1.Ro conc	18.1	1.49	77.56	68.8	82.5
2.Cyanide Residue		0.05	1.27	66.7	65.1 *
Overall Recovery (1 + 2)				89.6	93.9

* calculated by difference

Note: The flotation tailings were cyanided for 24 hours at a dilution of 2:1 and a solution strength of 1.0 lb NaCN/ton and 1.0 lb CaO/ton.

Tests 26, 27, 28 and 29

Results of Test 28 - 15 minutes

Product	WT. %	Assays Au	Assays Ag	Dist % Au	Dist % Ag
Final conc	8.3	2.69	150.55	57.6	74.9
3rd cl tail	0.8	0.56	37.57	1.2	1.8
2nd " "	4.7	0.595	17.27	7.2	4.9
1st " "	12.3	0.32	7.955	10.2	5.9
Float "	73.9	0.125	2.84	23.8	12.5
Head (calcd)	100.0	0.39	16.68	100.0	100.0
1. Ro conc	26.1	1.13	55.92	76.2	87.5
2. Cyanide residue		0.04	1.04	68.0	63.3*
Overall recovery (1 + 2)				92.4	95.4

Results of Test 29 - 20 minutes

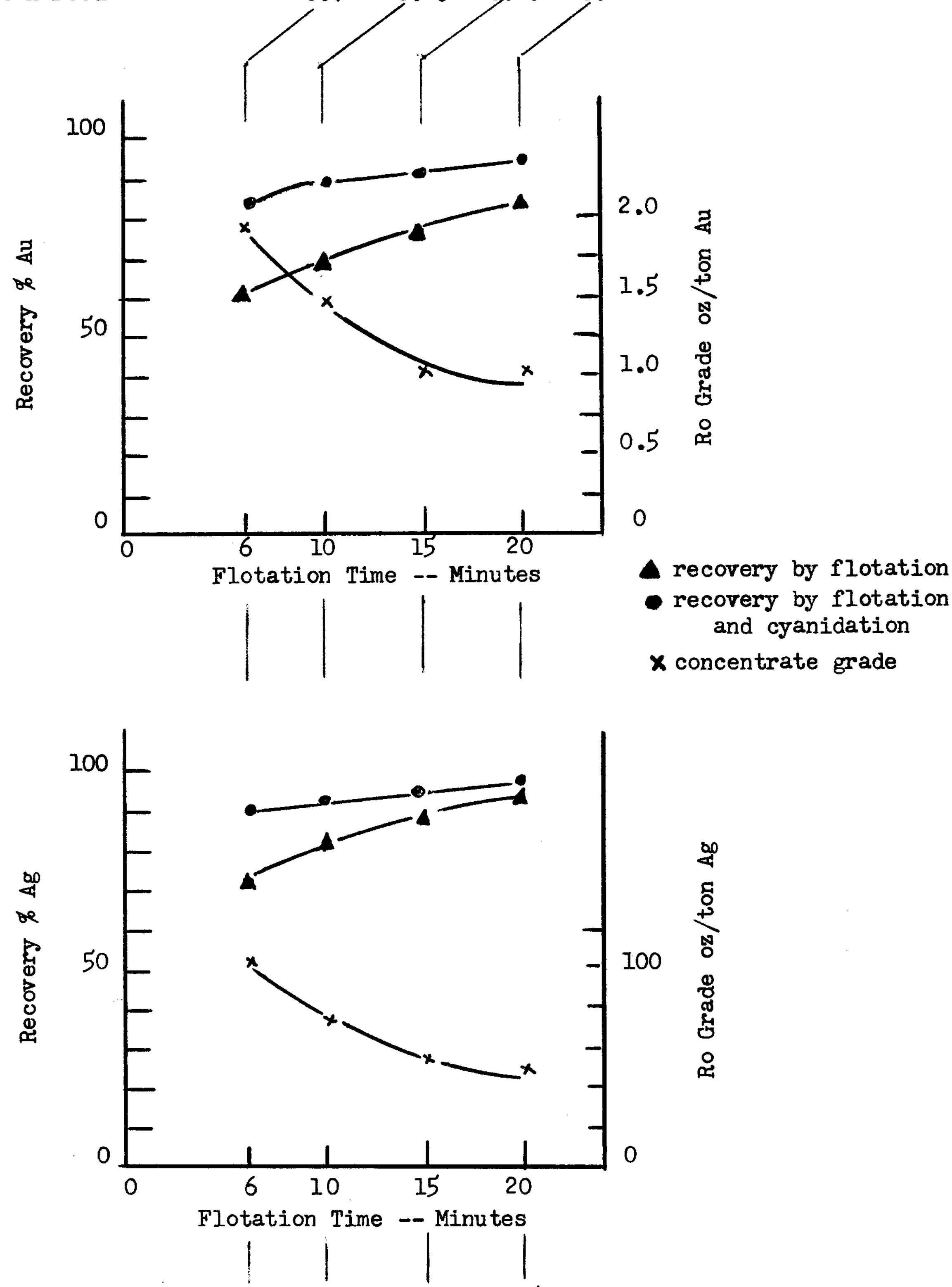
Product	WT. %	Assays Au	Assays Ag	Dist % Au	Dist % Ag
Final conc	8.2	2.88	154.29	59.8	75.2
3rd cl tail	0.6	1.52	64.61	2.3	2.3
2nd cl tail	2.8	0.70	28.08	5.0	4.7
1st cl tail	26.5	0.245	6.325	16.4	10.0
Float tail	61.9	0.105	2.145	16.5	7.8
Head (calcd)	100.0	0.39	16.83	100.0	100.0
1. Ro conc	28.1	1.17	51.61	83.5	92.2
2. Cyanide Residue		0.035	0.935	66.7	56.4 *
Overall recovery (1 + 2)				94.5	96.6

* calculated by difference

Note: The flotation tailings were cyanided for 24 hours at a dilution of 2:1 and a solution strength of 1.0 lb NaCN/ton and 1.0 lb CaO/ton.

FIGURE 1 - **Recovery and Grades Vs Flotation Time** Tests 26-29

Reagent Consumption lb/ton Feed	NaCN -	1.20	0.96	0.72	0.56
	CaO -	6.72	6.28	6.16	6.00



Flotation Reagents lb/Feed	301--	0.10	0.20	0.20	0.25
	Z-6--	0.15	0.20	0.30	0.40
	P.O.--	0.16	0.18	0.24	0.30

Test 32

2000 grams ground to 55%-200 mesh

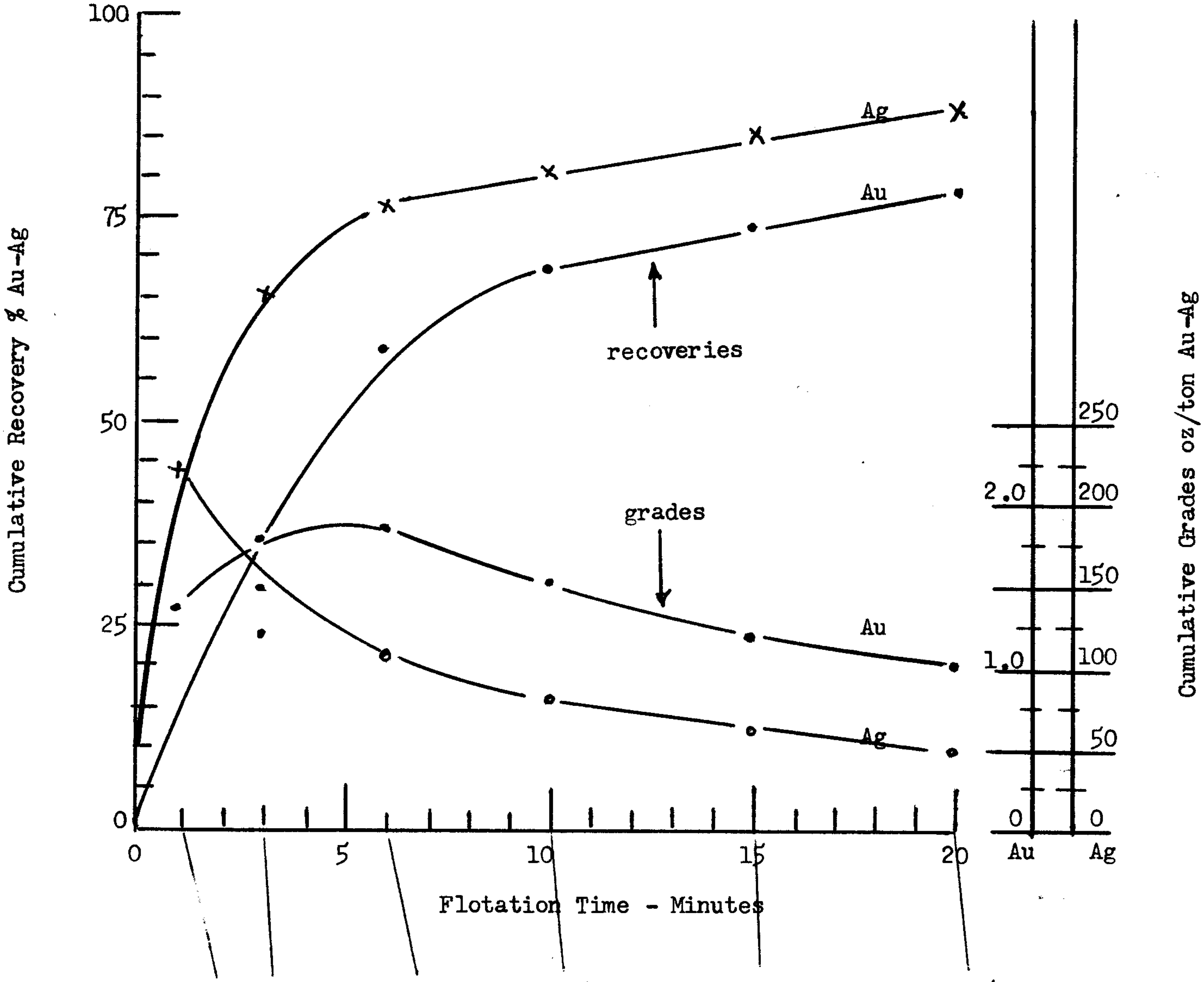
Flotation Scheme

Operation	Time Min	% S	pH	Reagents lb/ton 301	Z-6	D.F. 250
Grinding	16.5	65		0.05	-	0.04
Flotation No. 1	1	35	6.5	-	-	0.02
" No. 2	2					
Conditioning	3			0.05	0.05	0.02
Flotation No. 3	3					
Conditioning	3			0.05	0.05	0.02
Flotation No. 4	4					
" No. 5						
Conditioning	3			0.05	0.05	0.04
Flotation No. 6	5					

Results of Test 32

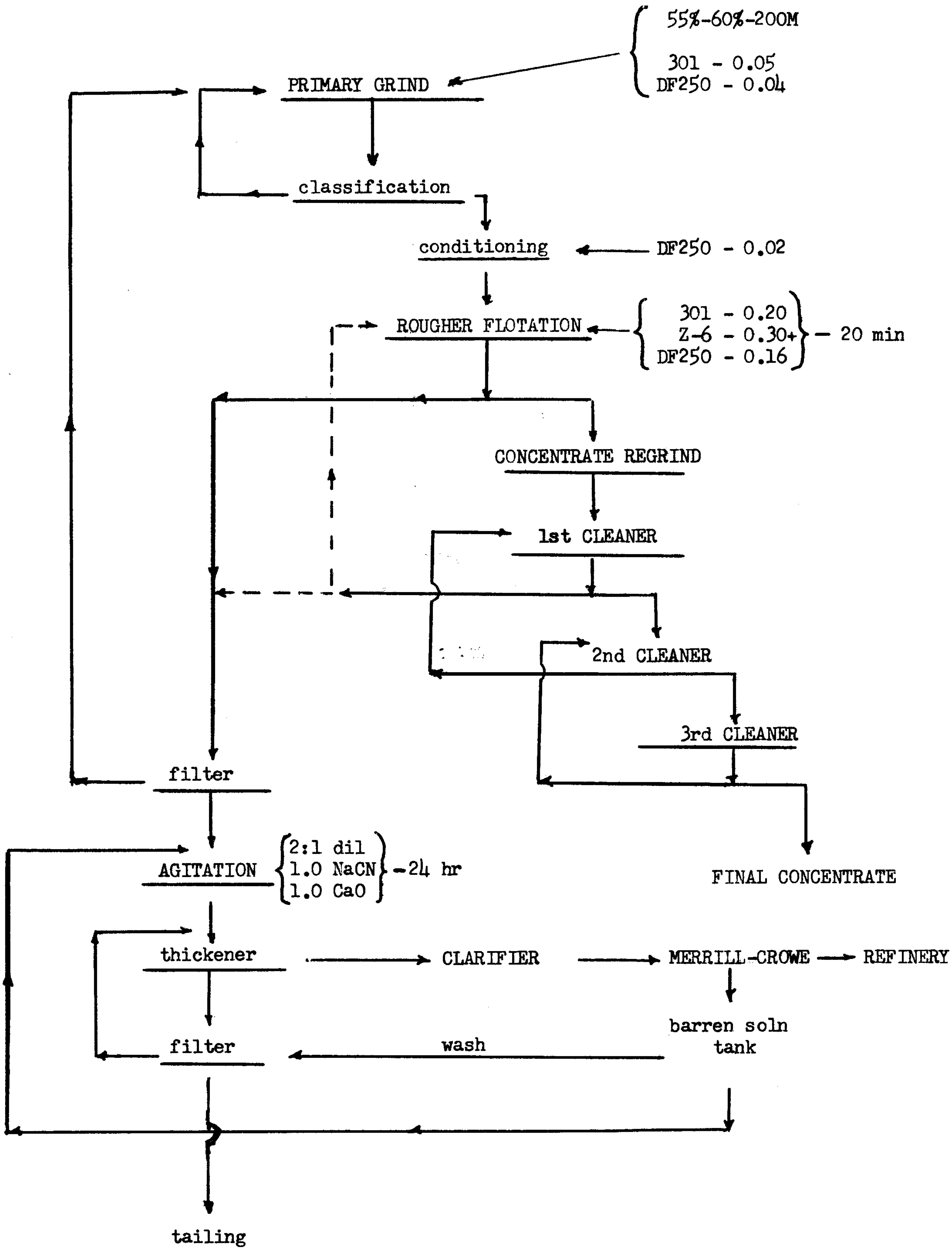
Product	WT. %	Assays Au	Assays Ag	Distn % Au	Distn % Ag
Flot conc No. 1	2.9	1.36	248.90	10.5	44.0
" " No. 2	3.2	1.60	108.73	13.6	21.2
" " No. 3	5.6	2.30	32.35	34.3	11.0
" " No. 4	5.3	0.71	17.60	10.1	5.7
" " No. 5	6.0	0.36	10.63	5.8	3.9
" " No. 6	5.0	0.28	8.30	3.7	2.5
" tail	72.0	0.115	2.665	22.0	11.7
Head (calcd)	100.0	0.38	16.41	100.0	100.0

FIGURE 2 - Results of Test 32



Reagents	301	0.05	-	0.05	0.05	-	0.05
lb/ton	Z-6	-	-	0.05	0.05	-	0.05
Feed	DF-250	0.06	-	0.02	0.02	0.02	0.04

FIGURE 3 PROPOSED FLOW SHEET



Department of Energy, Mines and Resources
Ministère de l'Énergie, des Mines et des Ressources
CANADA

Mines Branch
Direction des mines

File Number
Nº à rappeler

MINERAL PROCESSING DIVISION

40 Lydia St.
Ottawa 1, Ont.
Sept. 25, 1968

Mr. B.S. Imrie,
Manager,
Mount Nansen Mines Limited,
420-475 Howe Street,
Vancouver 1, B.C.

Dear Mr. Imrie:

Progress Report No. 4

As we indicated in our last report, in an operation in which a cyanidation circuit is to be employed to recover additional gold and silver from rougher flotation tailings it is sometimes advantageous to grind in cyanide thus providing a longer contact time. The tests shown in this report investigated the use of NaCN and lime in the primary grind and the rougher flotation circuit.

The ore used was the remaining half of the original shipment which when analysed gave the following results.

Head Sample Analysis

Gold (Au)	- 0.417	oz/ton
Silver (Ag)	- 15.87	" "
Lead (Pb)	- 0.99	per cent
Copper (Cu)	- 0.09	" "
Zinc (Zn)	- 0.41	" "
Iron (Sol Fe)	- 4.08	" "
Sulphur (Total S)	- 3.09	" "
Insoluble	- 79.05	" "

...2

FILE 30.IX. 68

SENT TO
1. D. CAMPBELL
2. H. JOHNSTON
3. S. GRAH..
4. . . COFFEE

B.I.

B.S. Imrie
Sept. 25
Page 2

<u>Test 33</u>

2000 grams ground to 55%--200 mesh

Flotation Procedure

Operation	Time min.	% Solids	pH	Reagents, lb/ton NaCN	CaO	301	Z-6	D.F. 250
Grind	16.5	65		1.0	1.0			
Condition	5	35	8.4			0.05		0.06
Float	3							
Condition	3		8.2			0.05	0.05	0.02
Float	5							
Condition	3		8.2				0.10	
Float	5							
Condition	3		8.1			0.05	0.10	0.04
Float	5							
Condition	3		8.0				0.05	0.02
Float	2							
Regrind	10							
Condition			7.8				0.05	0.04
Float								
Condition							0.05	0.02
Float	4							
1st cleaner	4		7.3					
2nd "	2		7.1					
3rd "	2		6.8					

Results of Test 33

Product	Wt. %	Assays Au	Ag	Dist % Au	Ag
Final conc	8.5	2.99	149.38	60.4	83.8
3rd cl tail	0.5	1.49	55.72	17.7	1.8
2nd cl tail	1.2	0.71	17.14	2.0	1.4
1st cl tail	3.5	0.38	8.87	3.2	2.0
? x Reg ro tail	18.9	0.195	3.77	8.7	4.7
Float "	67.4	0.05	1.41	8.0	6.3
Head (calcd)	100.0	0.42	15.15	100.0	100.0

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The filtrates from all of the flotation products were combined and assayed for gold and silver. Using 20 assay tons of filtrate only trace amounts of gold and silver were observed.

Samples of the rougher flotation tailing and of the regrind rougher flotation tailing were cyanided for 24 hours at a solution strength of 1.0 lb NaCN/ton and 1.0 lb CaO/ton and dilution of 2:1.

Results of Cyanidation Tests

Product	Reagents lb/ton ore		Head oz/ton		Residue oz/ton		Extraction	
	NaCN	CaO	Au	Ag	Au	Ag	Au	Ag
Ro float tail	1.0	2.1	0.05	1.41	0.03	0.625	40.0	55.7
Regrind ro float tail	1.6	5.75	0.198	3.77	0.11	0.875	43.6	76.8

The overall recovery of gold and silver by flotation and cyanidation is summarized below.

	Au %	Ag %
Final flotation concentrate	60.4	83.8
3rd cleaner tailing	17.7	1.8
2nd " "	2.0	1.4
1st " "	3.2	2.0
Regrind rougher tailing cyanide	3.8	3.6
Rougher flotation " "	3.2	3.5
	90.3	96.1

If the regrind rougher tailing were returned to the flotation circuit rather than to cyanidation the overall recovery would be somewhat higher.

...4

Test 34

Following the flotation scheme outlined in Test 33 28000 grams of ore was floated in two batches in a single Agitair cell and the rougher concentrates were combined reground, refloated and cleaned. This large amount of ore was used to obtain sufficient final concentrate for roasting tests.

The flotation tailing was sampled and then cyanided in an open agitator for 48 hours at a dilution of 2:1 and a solution strength of 1.0 lb NaCN/ton and 1.0 lb CaO/ton.

Results of Test 34

Product	Wt. %	Assays		Dist %	
		Au	Ag	Au	Ag
Final conc	8.8	3.21	141.15	67.7	83.6
3rd cl tail	1.3	0.615	14.89	1.9	1.3
2nd " "	2.3	0.415	8.83	2.3	1.4
1st " "	4.2	0.425	9.085	4.3	2.6
Reg. ro tail	8.6	0.29	4.89	6.0	2.8
Float "	74.8	0.10	1.59	17.8	8.3
Head (calcd)	100.0	0.42	14.85	100.0	100.0
Flot tail cy res.		0.04	0.795	10.7	4.2
Recovery %				92.9	95.9

Less material was floated during rougher flotation in this test than in Test 33. This was due in part to the difficulty in judging the optimum amount of reagents to be used. About half the amount of reagents used in Test 33 were used in this test.

The overall recovery in this test was approximately the same as in Test 33.

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Roasting tests have been started on the final concentrate produced in Test 34, and the results of these tests will be contained in our next report to you.

Enclosed with this report is a mineralogical report on the high-grade sample from the Heustis ore body.

Yours very truly,

T.F. Berry

T.F. Berry
Non-Ferrous Minerals Section

TFB/mn
Enclosure

Department of Energy, Mines and Resources
Ministère de l'Énergie, des Mines et des Ressources
CANADA

Mines Branch
Direction des mines

File Number
Nº à rappeler

Mineral Processing Division

40 Lydia Street,
Ottawa 1, Ontario,
December 17, 1969.

22 XII V 69

Mount Nansen Mines Limited,
420 - 475 Howe Street,
Vancouver 1, B.C.

6

Attention: Messrs. F. Bianconi and R. Saager

Gentlemen:

The current investigation has been completed and a final report has been written. This report has been delayed pending the results of some additional test work which we feel may offer alternative methods of treatment.

The present work indicates that flotation to produce a shipping concentrate will have to be supplemented by the cyanidation of the flotation tailing in order to ensure maximum recovery of the gold and silver. The Webber ore (Mines Branch Investigation IR-67-59) is more amenable to treatment by straight cyanidation than by flotation and since this ore will have to be treated sooner or later by cyanidation an alternative method of treatment which has been investigated is one in which the ground pulp was cyanided and then the cyanide residue washed and floated to recover a saleable concentrate.

For comparison purposes the flotation scheme used in Test 33 was followed with the exception that 1.0 lb $CuSO_4$/ton was added to flotation to condition the pulp after cyanidation and washing. The two schemes are:

Test 1 Grinding to 75% -200 mesh, flotation of a rougher concentrate, regrinding the rougher concentrate, four cleaner stages, cyanidation of the rougher flotation tailing.

Test 2 Grinding to 75% -200 mesh, cyanidation of ground pulp at 2:1 dilution, washing, conditioning cyanide residue with $CuSO_4$, flotation of a rougher concentrate, regrinding of the concentrate, four cleaner stages.

The results obtained using these two treatment schemes may be seen in the following tables.

Results of Test 1 (Test 33)

Product	Weight %	Assays oz/ton		Distribution %	
		Au	Ag	Au	Ag
Final concentrate	8.5	2.99	149.38	72.0	83.8
4th cl tailing	0.5	1.49	55.72	2.0	1.8
3rd " "	1.2	0.71	17.14	2.4	1.4
2nd " "	3.5	0.38	8.87	3.8	2.0
1st " "	18.9	0.195	3.77	10.2	4.7
Flotation "	67.4	0.05	1.41	9.6	6.3
Pregnant sol (calcd)	-	-	-	3.8 36.5	3.5 55.5
Cyanide Residue		0.03	0.625	5.8	2.8
Head (calcd)	100.0	0.42	15.15	100.0	100.0

Results of Test 2

Product	Weight %	Assays oz/ton		Distribution %	
		Au	Ag	Au	Ag
Pregnant solution*	-	0.102	2.654	43.2	33.0
Final concentrate	7.7	2.48	114.29	40.5	54.7
4th cl tailing	0.4	1.742	54.358	1.5	1.4
3rd " "	0.5	1.039	26.386	1.1	0.8
2nd " "	3.0	0.441	6.552	2.8	1.2
1st " "	23.2	0.11	1.845	5.4	2.7
Flotation "	65.2	0.04	1.53	5.5	6.2
Head (calcd)	100.0	0.472	16.08	100.0	100.0

*Assay expressed in oz per ton of solution.

Although a higher recovery of gold and silver was obtained in Test 2, the disposable tailing in Test 2 assayed 0.04 oz Au/ton and 1.53 oz Ag/ton compared with 0.03 oz Au/ton and 0.625 oz Ag/ton in Test 1. Also to be considered is the fact that in Test 2, 100% of the ore would have to be cyanided compared to about 67% in the Test 1 treatment. The concentrate Ag grade in Test 2 is considerably lower than in Test 1 which might make this difficult to market. Thus on balance it seems that flotation followed by cyanidation is the best treatment process.

In paragraph 2 of your letter you indicated that because of high shipping costs the minimum concentration ratio which could be tolerated would be, at a rough guess, 10:1. That is to say that the treatment of

10 tons of ore should not produce more than 1 ton of concentrate. In most of the flotation tests which were done, it was found that the concentrate ratio was in excess of this figure and in Test 33 and Test 34 which you specifically mention the concentration ratios considering the final concentrates only were 11.8:1 and 11.4:1, respectively. Because of the long flotation time necessary for maximum recovery a large amount of material remained in the cleaner fractions as a circulating load. While one may roughly estimate that about 50% of the gold and silver in the cleaner tailing products may be recovered in a continuous plant treatment, there is no way of determining exactly how much more weight would be in the final concentrate. A rough estimate is that the concentration ratio would be less than 10:1; that is that 10 tons of ore would produce something in excess of 1 ton of shipping concentrate.

Initially, the reason for roasting was to reduce if possible, the antimony contained in the final Au-Ag concentrate to avoid or reduce the smelter penalties. The only advantage gained by roasting tests which were completed is a reduction in the shipping weight brought about by the elimination of about 90% of the arsenic and sulphur. The weight loss amounted to as much as 28% with a corresponding increase in the grade. The following table shows results which are typical of those obtained in the roasting tests.

Element	Concentrate	Calcine
Gold (Au) oz/ton	3.21	4.39
Silver (Ag) "	141.15	198.88
Arsenic (As) per cent	16.86	1.52
Antimony (Sb) " "	2.10	1.16
Sulphur (Tot S) " "	26.94	2.00
Weight Loss " "		28.00

Cyanidation tests have been completed on the concentrate obtained in Test 34 and on calcines resulting from the roasting of that concentrate. The results of these tests were very unsatisfactory with the greatest extraction achieved being about 42% of the gold and 34% of the silver.

A series of roasting and cyanidation tests have just been completed and it is these tests which have resulted in the delay in the issuance of the final report and of this letter. We recognize that any flotation concentrate which may be produced may be too low-grade to make shipment to a smelter attractive. Therefore a series of roasting tests has been done as carefully as possible on a composite final flotation concentrate. In these tests varying* percentages of salt (NaCl) were added to the charges in an attempt to improve

*A process developed at the Mines Branch for recovering gold and silver from concentrates containing antimony as stibnite.

the cyanidation recovery of gold and silver from the calcines. The calcines produced were cyanided for 48 hours at a solution strength of 2.0 lb NaCN/ton with sufficient lime added to maintain an alkaline solution.

Test No.	Weight in gms Charge	NaCl	Calcine	Weight Loss%	Time at Temp hrs 450°C	650°C	Total Roasting time hr *	Assays oz/ton Au	Ag
Conc								2.67	155.91
3-A	200	0	152.2	23.9	1 1/2	1/2	2 1/2	3.44	207.38
3-B	200	10	152.9	27.2	1 3/4	1	3 1/4	2.99	199.19
3-C	200	20	144.9	34.1	1 3/4	3 1/2	5 3/4	2.52	177.52
3-D	200	40	134.3	44.0	1	2 3/4	4 1/2	2.22	144.93
3-E	200	20	162.5	26.1	1 1/2	1/2	2 1/2	2.78	173.83

*Roasting was done in a muffle furnace, with the fan on and the door ajar to permit almost continuous rabbling. The time at 450°C was determined by the speed with which the As_2O_3 evolved.

Note: Assays for As, Sb, Sol Fe, Tot S, to come.

In Test 3-D the low arsenic evolution time of 1 hr was the result of a depth of charge of less than 1/2 inch. In the other tests the charge depth was 1 inch. The calcine in Test 3-A was a light brick red in colour and in all of those tests in which salt was used, including those tests in which the sulphur had not been roasted to completion, the calcines were dark maroon.

The calcines from the roasting tests were weighed, sampled and assayed for gold and silver. A 100 gram calcine charge from each test was ground for 20 minutes with a small amount of lime and was cyanided for 48 hours at a dilution of 5:1 and a solution strength of 2.0 lb NaCN/ton. Lime was added to maintain a protective alkalinity during cyanidation. The solutions were titrated frequently in an attempt to maintain the cyanide concentration. With tests 3-A, 3-B and 3-E the end points for the cyanide determination were difficult to see. This is undoubtably caused by sulphur in solution resulting from incomplete roasting.

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Test No.	Weight gms		Weight Loss %	Reagent Consum lb/ton		R.P. ml 10/N $KMnO_4$/L	Assays oz/ton				Corrected for Weight Loss		Extraction %	
							Charge		Residue					
	Charge	Residue		NaCN	CaO		Au	Ag	Au	Ag	Au	Ag	Au	Ag
3-A	100	95.0	5	42.8	21.0	360	3.44	207.38	1.315	96.76	1.249	91.92	63.7	55.7
3-B	100	88.0	12	22.8	15.0	220	2.99	199.19	1.865	18.30	1.641	16.10	45.1	91.9
3-C	100	84.1	15.9	24.4	6.0	80	2.52	177.52	0.40	8.70	0.34	7.32	86.5	95.9
3-D	100	77.3	22.7	28.0	4.0	80	2.22	144.93	0.405	5.08	0.313	3.93	85.9	97.3
3-E	100	81.6	18.4	35.4	24.0	340	2.78	173.83	0.755	21.51	0.616	17.55	77.8	89.9

The results from cyaniding the calcines in which salt had been added look very encouraging. If this method was used for processing your ore it would eliminate the shipping and marketing of a gold-silver concentrate in favour of a recovery plant on the property.

The silver-bearing minerals have been identified as Pyrargyrite - $Ag_3Sb_2S_3$, Freibergite - $(Cu,Fe,Ag)_{12} Sb_4S_{13}$, Andorite - $(Pb, Ag) Sb_3S_6$, and Electrum, an alloy of gold and silver. The only gold -bearing mineral found in the ore was Electrum. In addition there are other minerals such as Boulangerite - $Pb_5Sb_4S_{11}$, Bournonite - $PbCuSbS_3$, Arsenopyrite FeAsS, Galena - PbS, Chalcopyrite - $CuFeS_2$, Pyrite - FeS_2 and Sphalerite ZnS. All of these minerals are intimately associated one with the other, to very fine sizes, and we believe it would be difficult, if not impossible, to selectively float a lead-silver concentrate from the gold or to float an arsenopyrite-pyrite concentrate containing the gold from the lead and silver.

We hope that this report will answer the questions which you pose in your letter to us of October 16,1969 regarding the treatment of the ore. The final report containing the additional test results will be sent to you as soon as possible.

Yours very truly,

T.F. Berry

T.F. Berry,
Non-Ferrous Minerals Section.

TFB/cw