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Bindal Gold Project: Feasibility testwork report

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Sammendrag

The general conklution from the testwork are that both L and F ore types will readily leach within 30 hours and that the better recoveries occur with the finely ground ore. Cyanide consumtion was found to be highest with the arsenic rich F-ore, possibly due to the interaction of cyanide and arsenic. In addition it is recomended that further investigations can be carried out to determine the optimum grind size and to establish reasons for the high cyanide consumption with the F-type ore.

Davy McKee

TERRA MINING AB

BINDAL GOLD PROJECT

FEASIBILITY TESTWORK REPORT

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May 1986
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CONTENTS

1.0 INTRODUCTION

Following discussions with Terra Mining AB, Davy McKee (Stockton) Ltd was contracted to carry out feasibility testwork on ore samples from the Bindal Gold Prospect, which is located in Northern Norway. The testwork was carried out by Warren Springs Laboratory at Stevenage, Hertfordshire, closely supervised by Davy McKee personnel.

The aims of the feasibility testwork were to provide basic metallurgical processing data for the different ore types expected at Bindal. This information could then be used to establish a conceptual process plant flowsheet which would be further developed by pilot plant testwork. However, due to a problem which arose with the representivity of the different ore types, only the arsenic rich Bindal F-Zone ore was tested, the L-Zone ore testwork being deferred until a later date.

The scope of the testwork programme included a preliminary investigation of the design parameters which would need to be defined before the full plant design could be established. Included in the testwork are crushing and grinding, froth flotation, roasting and cyanidation investigations. An exception to this work programme was the gravity testwork, which, although originally considered for inclusion in the scope, was not actually carried out. This was because there was a very limited quantity of ore which could be used and any gravity results obtained could not have been quantified. The quantity of free gold in the sample was expected to be very small, most of the gold was likely to be associated with the arsenopyrite.

Additionally, Terra Mining requested Davy McKee to review a testwork report on the Bindal L-Zone samples produced by Hitec Ore Processing Inc of Canada. It was requested that a summary and comment of this testwork be included with the current F-Zone testwork report to form a document summarising the overall situation with recent Bindal ore testwork.

2.0 TESTWORK SAMPLES

Initially three separate sample lots marked 'L', 'F' and '85N' were received at Warren Springs having a total weight of approximately 500 kg. However on instructions from Terra Mining AB, the samples marked 'L' and '85N' were not tested as they were thought to have an unrepresentatively low gold grade.

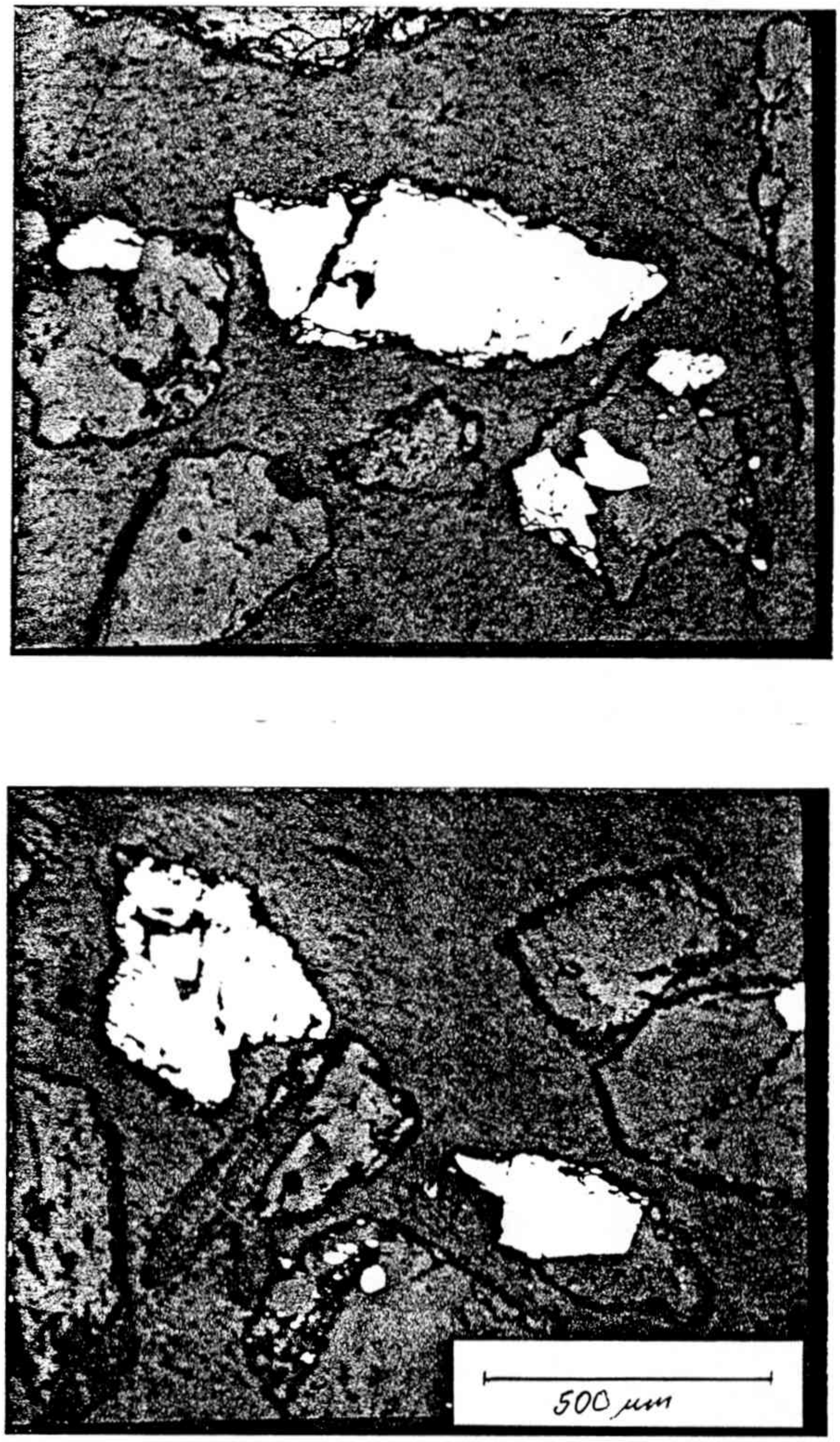
A further 20t consignment of 'L' type ore was sent to Warren Springs which was crushed, sampled and fire assayed. The following results were obtained:

Sample	Au g/t	As %
L1	1.50	1.35
L2	1.33	0.38
L3	1.14	0.34
L4	0.95	0.82

Due to the low gold assay figures obtained from this ore sample, it was decided that all planned testwork on the Bindal L-Sample should be suspended until further site investigations establish more accurate ore grades.

The work carried out by Warren Springs in the current testwork programme is based only on the Bindal F-type ore sample. This consisted of four bags labelled 85F001B, 85F0038, 85F004B and 85F007B which together weighed approximately 118 kg. The results and discussion of the testwork are only related to this sample, and as it is not representative of the entire Bindal deposit, the findings cannot be assumed to relate to all ore types.

However, when the geology and mineralogy of the Bindal L-zone are more fully investigated, it is intended that a more representative sample will be tested at Warren Springs. Results from this testwork could then be combined with the present results to give a more general view of the likely metallurgy of the deposit.



Photomicrograph of -600+300 µm fraction
(White grains - arsenopyrite, dark grains - gangue material)

Photomicrograph of -300 µm fraction

(White grains - arsenopyrite, dark grains - gangue material)

3.0 SAMPLE PREPARATION

A number of particles of rock were taken for Bond Crushability tests before the whole rock sample was crushed. On completion of this test this material was returned to the bulk of the sample.

The contents of the four bags were mixed together, stage crushed to -3.4 mm and 12 x 1 kg samples were riffled out for the Bond Crushability Test. The remainder was further stage crushed to -1.7mm and 40 kg riffled out and split into 1 kg samples for use in the testwork.

4.0 CRUSHABILITY AND GRINDABILITY TESTWORK

4.1 Crushability

A standard Bond Crushability Test was carried out on the 50 to 75 mm lump material as received. The Work Index determined was 14.7 kWh/tonne.

The complete results are shown in Table 1.0.

4.2 Grindability

A Bond Grindability Test was carried out on the -3.4 mm crushed material sample previously. The closing mesh size for the grindability test was 125 microns as determined by mineralogical examination and flotation testwork. The work index determined was 17.4 kWh/tonne.

The complete results are shown in Table 2.0 and Figure 2.0.

4.3 Grinding Tests

A series of grinding tests were carried out at a pulp density of 50% solids by weight. The 80% passing size versus grinding time was plotted (See Fig 1.0). This gave a 26 minute grinding time to give 80% passing 100 microns.

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TABLE 1.0 - TWIN PENDULUM IMPACT CRUSHABILITY TEST

Test No	Thickness Inches	Angle of Breakage	No of Pieces	Energy ft lb	ft lb/in
1	1.75	30	3	12.5	7.14
2	2.0	45	2	27.2	13.60
3	2.25	35	3	16.8	7.47
4	2.5	20	2	5.6	2.24
5	2.5	50	3	33.2	13.28
6	2.75	50	4	33.2	12.07
7	2.75	85	4	84.7	30.80
8	2.13	30	4	12.5	5.88
9	2.88	75	5	68.8	23.93
10	1.5	30	3	12.5	8.33
11	2.0	50	4	33.2	16.6
12	2.0	50	3	33.2	16.6
13	2.38	65	4	53.6	22.57
14	1.63	40	3	21.7	13.35
15	1.25	40	2	21.7	17.36
16	2.63	60	5	46.4	17.68
17	2.25	50	6	33.2	14.76
18	1.5	35	2	16.8	11.20
19	2.75	55	4	39.6	14.40
20	2.88	75	6	68.8	23.93
				Mean Value =	14.66

	(1)	(2)	(3)	(4)
Wt. empty jar	416.3	413.8	416.3	416.3
Wt. jar + ore	764.1	777.2	820.2	786.4
Wt. jar + water	916.2	910.0	916.3	916.4
Wt. jar + water + ore	1143.2	1146.6	1176.4	1165.6
S.G.	2.879	2.866	2.809	2.827

Average S.G.	2.85
Average Impact	14.66 ft.lb/in
Maximum Impact	30.80 ft.lb/in
Minimum Impact	2.24 ft.lb/in
Average Impact Omitting maximum and minimum	14.45

W.I. $\frac{2.59 \times 14.66}{2.85}$ = 13.32 kWh/ton

<u>Crushability Index</u> <u>14.68 kWh/tonne</u>

TABLE 2.0 - BOND GRINDABILITY TEST AT 125 UM

Size Distribution of Bond Mill Feed

Particle Size mm	wt.g	wt.%	wt.% Undersize
-3.400+2.800	65.3	13.1	86.9
-2.800+2.360	67.6	13.6	73.3
-2.360+1.700	105.0	21.1	52.2
-1.700+1.180	71.7	14.4	37.8
-1.180+0.600	75.6	15.2	22.6
-0.600+0.300	45.9	9.2	13.4
-0.300+0.180	21.5	4.3	9.1
-0.180+0.125	14.0	2.8	6.3
-0.125+0.090	8.4	1.7	4.6
-0.090+0.075	4.1	0.8	3.8
-0.075+-.053	8.3	1.7	2.1
-0.053	10.6	2.1	
	498.0	100.0	

80% passing size of feed = 2600 µm

Percentage minus 125 um in feed = 6.3%

Weight of unit volume of feed (700 ml) = 1283.4g

Weight of product required 1283.4/3.5 = 366.7g

Period	No of Revs	-125µm in P	-125µm in F	-125µm net P	g/rev
1	150	303.7	80.9	222.8	1.485
2	234	299.8	19.1	280.7	1.200
3	290	382.9	18.9	364.0	1.255
4	272	384.1	24.1	360.0	1.323
5	259	353.2	24.2	329.0	1.270
6	271	388.7	22.3	366.4	1.352
7	253	359.5	24.5	334.5	1.322
8	260	354.2	22.6	331.6	1.275
9	270	367.8	22.3	345.6	1.280
10	268	368.7	23.2	345.5	1.289
11	266	365.6	23.2	342.4	1.287

Average g/rev of the last three consecutive periods = 1.285g

Average circulating load of the last three consecutive periods = 249.4%

Size distribution of Bond Mill Product

Particle Size mm	Wt.g	Wt.%	Wt.% Undersize
-0.125+0.106	49.2	16.5	83.5
-0.106+0.090	29.1	9.8	73.7
-0.090+0.075	31.6	10.6	63.1
-0.075+0.063	34.1	11.5	51.6
-0.063+0.053	22.7	7.6	44.0
-0.053+0.045	4.6	1.5	42.5
-0.045	126.5	42.5	
	297.8	100.0	

80% Passing size of product = 103µm

Calculation of Bond Work Index (Wi)

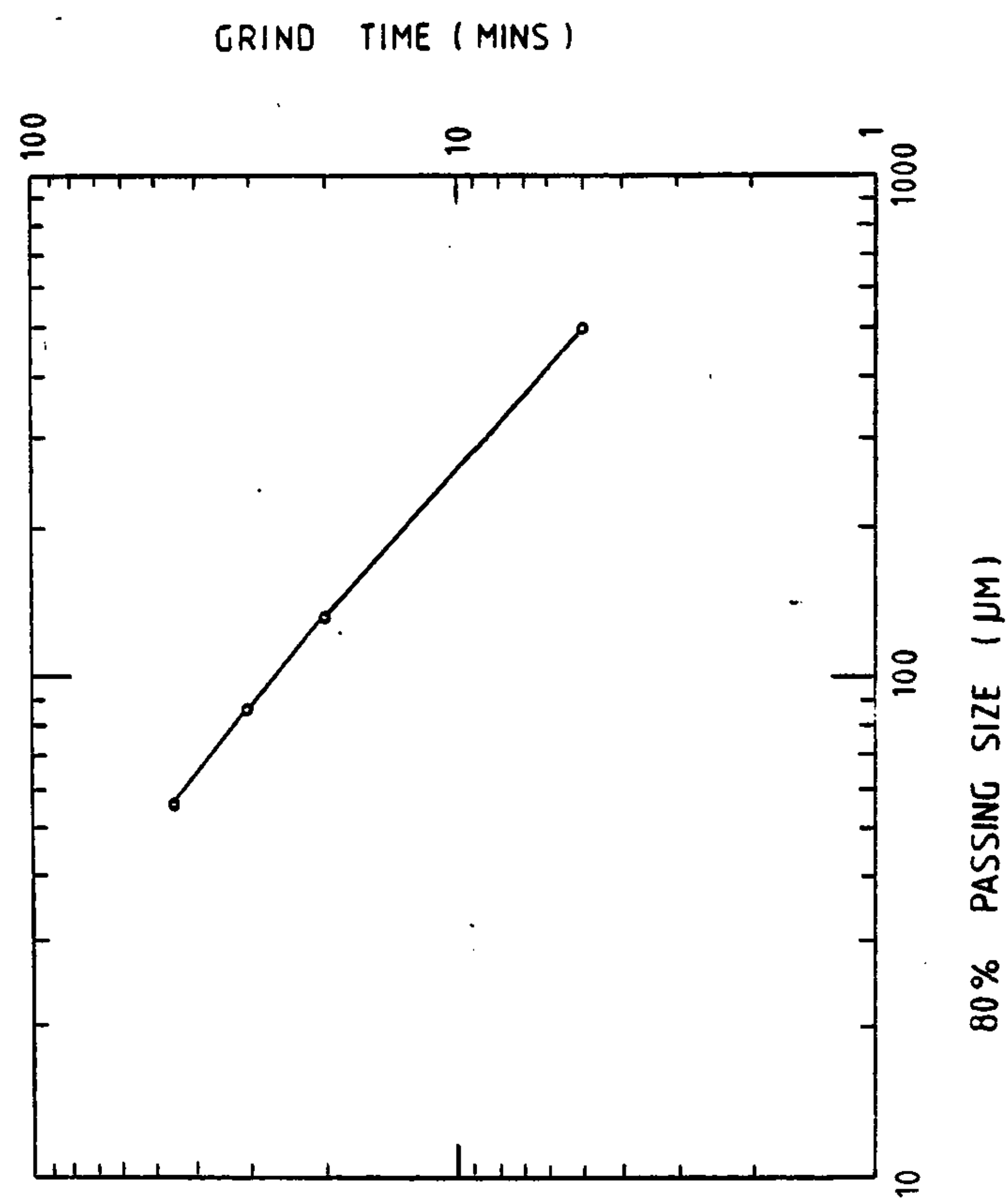
F = 80% passing size of Mill Feed = 2600µm
P = 80% passing size of Mill Product = 103µm
P = Limiting screen size used = 125µm
Gbp = g/rev = 1285g

$$Wi = \frac{44.5}{(P1)^{0.23} \times (Gbp)^{0.82} \times \left[\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right]}$$

$$Wi = \frac{44.5}{(125)^{0.23} \times (1.285)^{0.82} \left[\frac{x\ 10}{\sqrt{103}} - \frac{10}{\sqrt{2600}}\right]}$$

Wi = 15.769 kWh/ton (short)

<u>Bond Work Index = 17.38 kWh/tonne</u>



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	ch		ch	drawn		
				checked		
				approved (engineer)		
C	dr	D	dr	approved (client)		
	ch		ch			
				microfilmed		

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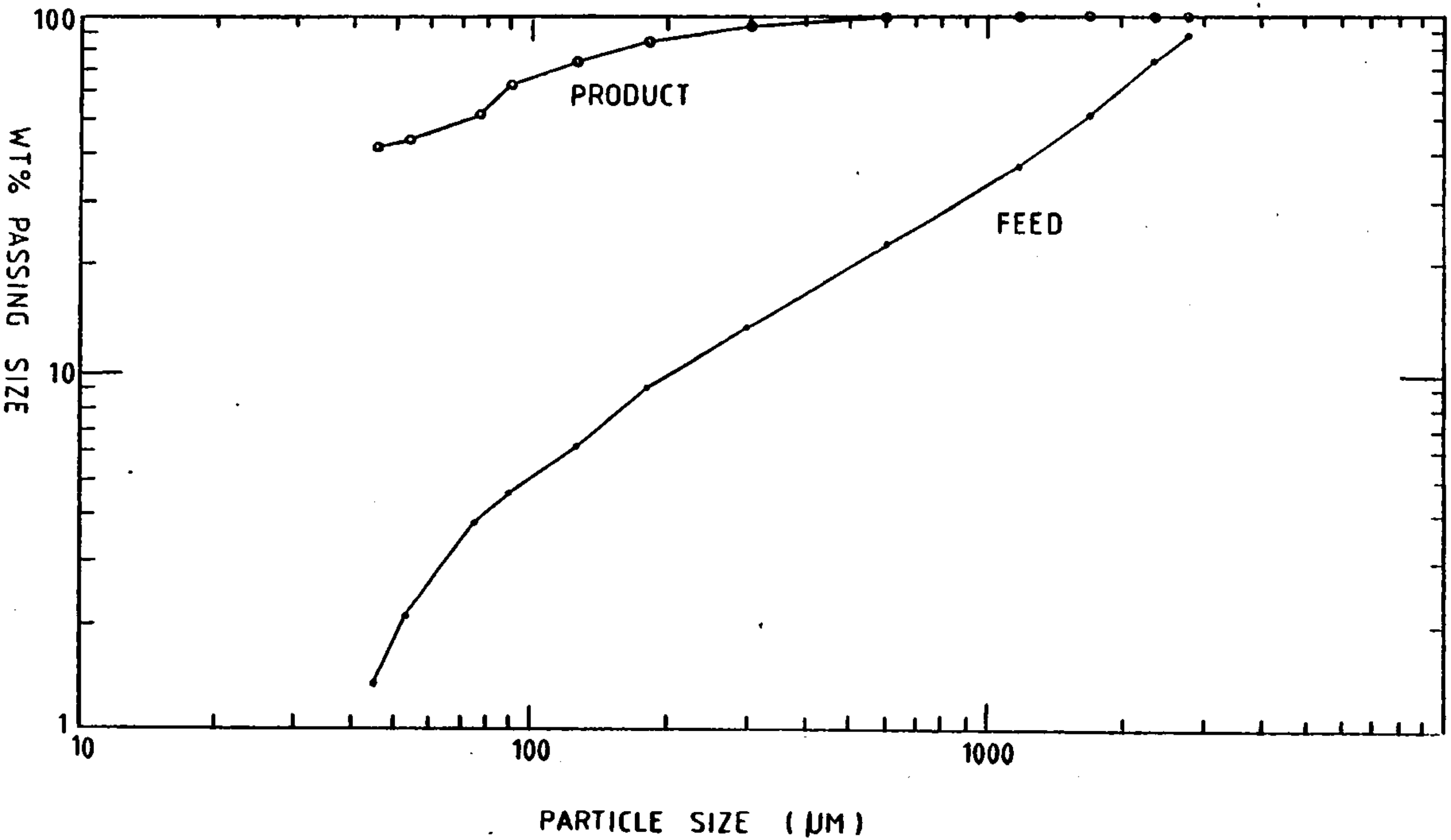
80% PASSING SIZE VERSUS
LABORATORY MILL GRIND TIME

scale | Drg. No FIGURE 1 | rev.

PENNINE LS



A dr	A ch	B dr	B ch	C dr	C ch	D dr	D ch

drawn	checked	approved (engineer)	approved (client)	microfilmed
by				
date				

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BOND MILL FEED AND PRODUCT SIZING

scale

Drg. No FIGURE 2

rev.

PENNINELS

5.0 MINERALOGICAL ANALYSIS

The feed and flotation products were examined microscopically in polished section grain mounts. Arsenopyrite was the only sulphide mineral found, the gangue being make up of silicates, dominantly quartz, feldspar and biotite.

No grains of native gold were seen during the examination of any of the sections.

The liberation size of the sulphide was indicated at 100 um by microscopic examination of the -600 +300 µm and -300 µm size fractions of a sample of the feed that had been ground for 5 minutes in a laboratory mill. The -600 +300 µm fraction contained liberated sulphide and coarse composites, while the -300 um fraction contained mainly liberated sulphide but some composite grains with inclusions of sulphide at about 100 µm.

The liberation size of 100 µm was also confirmed by the flotation testwork.

6.0 FLOTATION TESTWORK

A series of five batch flotation tests were carried out at a grind size of 80% minus 100 µm to determine suitable conditions for the production of a flotation concentrate which could be further processed to recover contained gold. A sixth test was carried out at a grind size of 80% -200 µm. Results are given in Table 3.0.

Potassium amyl xanthate (PAX) was selected as the collector and Dowfroth 250 as the frother, no other reagents were used in the testwork apart from small quantities of sulphuric acid for pH control. The dosage rates being determined by staged reagent addition floats.

6.1 Tests 1 and 2

The first two tests were carried out at natural pH of 7.8 and 8.0 and reagents were added stagewise after a 5 minute conditioning time for each stage. In Test 1 the addition of 20 g/t PAX per stage was found to be too high, as all sulphides floated readily in the first flotation stage, with very little in the second stage.

The second test was carried out under similar conditions but with a staged collector addition of 5 g/t PAX. This resulted in a heavy float in the first stage (as Test 1) with much lighter floats in the subsequent three stages of flotation.

It can be concluded from the first two flotation tests that a total collector addition of 20g/t PAX would be required to float the sulphide in the sample.

6.2 Tests 3 and 4

The effects of varying pH and cleaning the rougher concentrate were investigated in Tests 3 and 4. In Test 3 the natural pH was used (8.0), and in Test 4 the pH was modified with sulphuric acid to 6.0. In both tests a total of 20 g/t PAX and 20 g/t frother were used in the rougher float, with the concentrate being cleaned with no additional collector addition and a further 10 g/t frother. There was little difference between the two tests with the weight floated to the cleaner concentrate being 12.5 and 12.3% and gold recovery being 92.1% and 90.0%. There appears to be no advantage in modifying the pH conditions for flotation.

6.3 Test 5

The conditions used for Test 4 were used for Test 5 to produce a bulk sulphide concentrate for further cyanidation and roasting testwork (see sections 7 and 8). This float was done in two steps with two lots of 4 kg being floated at each stage. The two rougher concentrates were then combined and cleaned to give 1 kg of cleaner concentrate which was then split into 10 x 100g lots.

6.4 Test 6

It was decided to carry out a float test at a coarser grind using the same flotation conditions as Test 4 (ie. pH 6.0, total 30 g/t frother, 20 g/t PAX), but with a grind of 80% passing 200 µm. The weight floated to the cleaner concentrate was only slightly reduced being 11.1% for Test 6 and 12.3% for Test 4, however, the gold recovery was very much reduced being only 45.1% as against 90% in Test 4.

6.5 Summary

The highest gold recovery from flotation was obtained in Test 1 where, because of high reagent additions, the second stage acted as a scavenger. A total gold recovery of 99% was obtained by the combined concentrates in this test.

Overall the flotation results indicate that gold recoveries in excess of 90% of the contained gold can be obtained by flotation at neutral or near neutral pH, with comparatively low reagent dosages.

TABLE 3.0 - FLOTATION TESTWORK

Test 1 - Stage Addition 20g/ton Potassium Amyl Xanthate Per Stage

Conditions:

Grind	80% passing 100µm
Float	Natural pH 7.8
Stage 1	20g/t Potassium Amyl Xanthate 10g/t Frother DF250 Condition 5 mins Float 10g/t frother Float to exhaustion
Stage 2	20g/t Potassium Amyl Xanthate Condition 5 mins Float (nothing floated) 10g/t frother Float (only slime and mica)

Results

	Wt.g	Wt%	Au g/t	Au%dis.
Conc 1	136.6	13.34	30.65	97.8
Conc 2	19.4	1.89	2.66	1.2
Tail	868.0	84.77	0.05	1.0
	1024.0	100.0	4.18	100.0

Test 2 - Stage Addition 5g/ton Potassium Amyl Xanthate

Conditions:

Grind	80% passing 100µm
Float	Natural pH 8.0
Stage 1	5g/t Potassium Amyl Xanthate 20g/t frother DF250 Condition 5 mins Float 6 mins Heavy Float
Stage 2	5g/t Potassium Amyl Xanthate 10g/t frother Condition 5 mins Float 6 mins Light float

Stage 3 and 4 as Stage 2

Results

	Wt.g	Wt%	Au g/t	Au%dis.
Conc 1	135.9	13.35	26.49	85.69
Conc 2	13.3	1.31	16.01	5.07
Conc 3	11.4	1.12	7.84	2.13
Conc 4	9.2	0.92	6.65	1.49
Tail	848.0	83.32	0.28	5.65
	1017.8	100.00	4.13	100.00

Test 3 - Rougher Cleaner Float Using 20g/ton Potassium Amyl Xanthate

Conditions:

Grind	80% passing 100µm
Float	Natural pH 8.02
Rougher	20g/t Potassium Amyl Xanthate 20g/t frother DF250 Condition 5 mins Float 4 mins 10g/t frother Float 5 mins (9 mins in all)
Cleaner	pH 8.2 Float 7 mins

Results

	Wt.g	Wt%	Au g/t	Au%dis.
Cl Conc	127.9	12.53	31.11	92.1
Cl Tail	32.8	3.21	3.08	2.3
Ro Tail	860.0	84.26	0.28	5.6
	1020.7	100.00	4.23	100.0

Test 4 - As Test 3 but as pH 6.0

Froth more mobile

Results

	Wt.g	Wt%	Au g/t	Au%dis.
Cl Conc	125.6	12.3	34.67	90.0
Cl Tail	22.4	2.19	7.08	3.3
Ro Tail	873.0	85.5	0.37	6.7
	1021.0	100.00	4.85	100.0

Test 5 - 8kg Bulk Float Using the Same Conditions as Test 4

The dried cleaner concentrate was split into 10 100g samples for further roasting and cyanidation tests.

Test 6 - Ground to 80% Passing 200µm

Conditions

Grind 80% passing 200µm

Float As Test 4

Results

	Wt.g	Wt%	Au g/t	Au%dis
Cl Conc	112.7	11.1	19.59	45.14
Cl Tail	19.3	1.9	7.68	3.03
Ro Tail	883.0	87.0	2.87	51.83
	1015.0	100.0	4.81	100.00

7.0 ROASTING TESTWORK

A series of roasting tests were carried out on small samples of flotation concentrate weighing approximately 100g. The small scale of the testwork was agreed to try and maximise information obtained from the limited quantity of flotation concentrate available.

The testwork was carried out in a 40 mm diameter silica reactor housed in a vertical tube furnace, heated electrically. A thermocouple positioned in the feed bed was used to control the reaction temperature to within 25°C of the selected temperature which ranged from 500°C to 650°C.

The flotation concentrate samples were roasted in air, passed through the reactor at a rate of 0.5 litres/minute. This caused a partial fluidisation of the bed and resulted in some carry over of fine material. This loss was estimated at 1 to 2% of the original feed weight. The material was charged cold into the furnace, brought up to the selected reaction temperature and left at this temperature for a period of 6 to 7.5 hours.

The roasted product was dark brown indicating that the iron had oxidised to magnetite. White fumes were given off during the roasting and a yellow condensate formed at the mouth of the silica reactor, which was probably a mixture of As_2O_3 and sulphur. During the tests some contamination of the roasted producted occurred due to the condensate falling back onto the bed during cooling. Results from the tests are given in Table 4.0.

The theoretical air requirement to oxidise 100g of arsenopyrite is 160 litres and is summarised by the following reaction.

$FeAsS + 29/12\ O_2 = 1/2\ As_2O_3 + 1/3\ Fe_3O_4 + SO_2$

So the air passed during each of the period of each roasting test is approximately 40%, 13%, 13% and 22% in excess of theoretical quantity of air to oxidise the arsenopyrite. The resulting weight loss during roasting was approximately 48%.

Davy McKee

TABLE 4 - SUMMARY OF ROASTING CONDITIONS

Test No	Material Fed g	Residue Wt g	Wt Loss %	Reaction Temp ^{o}C	Reaction Time hrs	Warm-up Time	Excess Air During Reaction %
1	99.7	51.1	48.7	500+25	7.5	45	40
2	98.4	51.0	48.2	500+25	6	30	13
3	99.1	58.0	41.5	575+25	6	60	13
4	99.1	51.0	48.5	650+25	6.5	100	22

Comments:

Test 3 - Slight signs of sintering

Test 4 - Distinct lumps in product: sintering

8.0 CYANIDATION TESTWORK

A series of standard cyanidation bottle tests were carried out both on ground ore and on flotation concentrates at different grind sizes to determine the effects on gold recovery. A solution containing 0.1% w/v HCN was used at a pH of approximately 10.0 for the leaching. Where larger leach samples were used (approximately 1 kg) the free cyanide and gold content of the leach solution was monitored throughout, to try and establish information on leach characteristics. However, for several samples it was only possible to leach 100g batches which could not be readily monitored, and so only overall leach figures are presented.

8.1 Run of Mine Ore Leaching

Nominally 1 kg samples of ore were wet ground in a laboratory rod mill to produce samples for cyanidation. The following grind sizes were examined:

(a) 80% - 75 microns produced by grinding -1.7 mm ore for 35 minutes.

(b) 80% - 100 microns produced by grinding -1.7 mm ore for 26 minutes.

Each sample was slurried with a sufficient quantity of water to produce a 50% w/w mixture and sodium cyanide was added to give a leach solution containing 0.1% w/v NaCN; ie. 1 kg/tonne of ore. The pH of each pulp was maintained at 10-12 by addition of calcium oxide as required. Leaching was performed at not less than 21°C using a standard rolling bottle technique.

The pH, rate and extent of cyanide consumption were monitored by removing small samples of liquor at known times throughout the leach. Cyanide consumption was monitored by titration with standard 0.01M silver nitrate solution and the rate of gold leaching was determined by analysing the same titrated samples by atomic adsorption spectrophotometry.

It was intended to leach each grind size for up to 72 hours, but the 80% -100 um sample was only leached for 49 hours since it was evident that gold dissolution occurred very rapidly and was leached to completion within the period.

Although there is some degree of variation between the results obtained using ROM ore, it is quite apparent that the rate and extent of the gold leaching is high with approximately 90% of the contained gold being leached within 24 hours. More detailed leach testwork will help to optimise leaching conditions for run of mine ore. In particular, additional grinding testwork may help to identify the optimum grind size for cyanidation, although it is apparent that both the grind sizes already selected will leach readily, with a possible slight advantage of the finer grind size. Table 5, indicates the ROM ore leaching results.

8.2 Flotation Concentrate Leaching

Leaching tests were carried out on samples of flotation concentrate prepared during flotation test No.5 as detailed in Section 6.3. One sample was reground in a small rod mill for 15 minutes before leaching. 48 hour leaching tests were performed on 100g samples under the same leaching conditions indicated in the previous section ie. 0.1% w/v NaCN.

Similar leaching tests on a 50g scale were also performed on the flotation concentrate as received after roasting to oxidise the arsenopyrite. 28.5 hour leaching tests were carried out on three samples, each of which had been roasted under differing conditions.

Due to the very small scale of the testwork on flotation concentrates, sampling was not possible for any of the leaching tests, thus gold extraction was determined at the end of each test by solution and residue analysis.

Details of the various cyanidation tests are summarised in Table 5.0. The gold extraction figures quoted are calculated from head values derived from combined final leach liquor and residue analyses. It will be noted that the consumption of cyanide by the flotation concentrate is much greater than that for the Run of Mine ore, probably due to the presence of residual flotation reagents on the concentrate particles. Consequently, at the standard conditions the gold extraction was much lower, due to the presence of insufficient cyanide. Accordingly two further tests (8 and 9) on roasted and unroasted flotation concentrate were carried out, in which the cyanide concentration was increased from 0.1% w/v to 1.0% w/v at a 24 hour leach period. The gold extraction was improved particularly on the roasted sample, with a smaller improvement being found with the unroasted concentrate. Further testwork would have to be carried out to determine the correct cyanidation conditions for the flotation concentrates.

It should be noted that the calculated gold heads on the roasted samples are about 17% lower than anticipated, this could be due to some gold being carried over with the fume from the roasting process.

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TABLE 5 - SUMMARY OF CYANIDATION TEST RESULTS

Test No	Feed	Leach Time Hr	Free NaCN w/v%	Au in Soln. mg	Au in Residue g/tonne	Total Au Au Extracted %	Calculated Head mg	Calculated Head mg/tonne
1	80%-75µm ROM ore 1011g	4.25	0.045	4.50		97.5	4.616	
		24	0.047	4.14		89.7	4.616	
		28.75	0.053	3.83		83.0	4.616	
		44.5	0.049	4.48		97.1	4.616	
		51.5	0.049	4.69		101.6	4.616	
		68.5	0.040	4.19	0.43	90.9	4.616	4.62
2	80%-100µm ROM 1034g	5	0.071	4.36		87.6	4.977	
		22	0.053	4.64		93.2	4.977	
		29	0.051	4.45		89.4	4.977	
		46.5	0.044	4.21		84.6	4.977	
		49	0.036	4.42	0.56	88.8	4.977	4.98
3	Flotation Concentrate Unground	48	0.005	0.39	29.37	11.9	3.268	32.7
4	Flotation Concentrate 15min grind	48	0.002	0.34	28.27	10.8	3.139	31.4
5	Flotation Concentrate 6hr roast @ 500C	28.5	0.003	0.27	45.28	10.5	2.566	50.6
6	Flotation Concentrate 6hr roast @ 575C	28.5	0.004	0.58	41.43	19.6	2.958	51.4
7	Flotation Concentrate 6.5hr roast @ 650C	28.5	0.003	0.36	47.47	13.1	2.743	54.5
8	Flotation Concentrate Unground	24.0		2.13	15.24	58.0	3.661	36.6
9	Flotation Concentrate 7.5hr roast @ 500C	24.0		2.53	6.57	88.38	2.865	56.1

9.0 CONCLUSIONS AND RECOMMENDATIONS

1. The energy required to grind crushed run of mine ore to the liberation size of the sulphide was 17.4 kWh/tonne. This represents a significantly harder than average ore type indicating that a commercial milling operation will have to have a substantial crushing and grinding circuit and that comminution power costs may be significant.

2. Mineralogical examination showed that arsenopyrite was the only sulphide mineral present in the ore sample. A liberation size of 100 um was indicated for this mineral by both microscopic examination and by flotation testwork.

3. Cyanidation testwork on the ROM ore showed that a high gold recovery was possible, approximately 90% in a leach time of 24 hours. This confirms the amenability of the sample to direct cyanidation.

4. The arsenopyrite in the ore floated very readily with an addition of only 20 g/t Potassium Amyl Xanthate and 20 g/t frother at a natural pH of approximately 8.0.

5. The best gold recovery by flotation was obtained (in Test 1) where effectively a rougher/scavenger circuit was adopted. Combining rougher and scavenger concentrates would give up to 99% recovery to a concentrate weight of only 15% of the feed.

   If the flotation/cyanidation route is adopted the rougher/scavenger circuit would be recommended.

6. Cyanidation of flotation concentrates indicated that a high consumption of cyanide may be expected due to the presence of cyanicides in the form of residual flotation reagents. Further testwork is recommended to determine possible ways of reducing cyanide consumption and improve recoveries.

7. Cyanidation results on the roasted and unroasted flotation concentrate were generally inconclusive. Further testwork would be required to confirm what conditions would be required to maximise the gold recovery by roasting. The high cyanide consumption found on roasted concentrates may turn out to be of a similar order as that for run of mine ore.

8. Allowing for inaccuracies of sampling and assying during the small scale batch testwork; it can be expected that a recovery of approximately 90% of the contained gold could be achieved in a commercial plant.

9. If a combined flotation/cyanidation route is used this would reduce the possibility of environmental problems concerned with tailings disposal. This is because most arsenopyrite would be removed during flotation leaving he tailings with a low As content. The arsenopyrite residue could then be disposed of separately away from the site after cyanidation.

10. The current results are very encouraging indicating that a high gold recovery may be obtained. Limited further batch testwork on Bindal ore should be carried out especially on the L-type ore once a representative sample can be obtained. If this work is successful it is recommended that more detailed pilot testing should be carried out on a larger bulk sample of ore to more fully investigate the metallurgy of the ore.

10.0 HITEC LEACH TESTWORK OF BINDAL GOLD ORE

Hitec Ore Processing Inc of Ontario, Canada, carried out limited testwork on different samples of Bindal gold ore including both the L and F ore types. The results are presented in a report dated 4 May 1986 addressed to Mr Jan Bida of Terra Mining.

The three ore samples tested by Hitec were composites of 24 individual samples. The head grades of the L-samples were found to be very low (0.947 and 1.557 g/t) as found with the Warren Spring L-Samples, and so may be unrepresentative of that ore type. The grade of the F-type ore was 6.579 g/t and is probably much more representative.

10.1 Sample Preparation

It is assumed that the ore samples were passed through a jaw crusher set at approximately -3/16" (4.76mm) and the product was split into three fractions, one being reduced to -1/8" (3.18mm), one being pulverised to an unspecified size and one remaining at -3/16". These three size fractions were then cyanided using a standard rolling bottle technique.

10.2 Cyanide Leaching

Known weights of sample were mixed with the leaching solution using a standard one gallon wide necked bottle which was continuously rolled for up to 47 hours. At fixed periods small samples of solution were removed for assay checks and to monitor the pH and free cyanide in the solution.

Precise details of sample weight, pulp density and free cyanide levels are not given for any of the leach tests.

10.3 Results

The results from the Hitec cyanidation testwork are summarised in Table 6. In each case the head grade and ore type is given complete with sample feed sizes, cyanide consumption hours and percentage gold extraction after 47 hours.

Sample 1 (L-ore) results show that up to 83% of the gold could be recovered within 47 hours with a cyanide consumption of 1.1 kg/t. Sample 2 (L-ore) testwork indicated that approximately 100% of the gold could be recovered with a cyanide consumption of 1.12 kg/t. In both cases the best gold recoveries were obtained with the 'pulverised' ore with much lower recoveries being found with samples of a coarser grind. Overall the results obtained from the L-ore samples must be treated as indicative, as the head grade of the samples were very low being 0.947 and 1.557 g/t gold and so the accuracy of the sampling and assaying will be lower.

The results for Sample 3 (F-ore) also show that the best recoveries can be obtained with the pulverised ore (81%) however the cyanide consumption is higher than that for the L-ore at 1.28 kg/t. As the head grade (6.579 g/t Au) is higher than the L-ore it may be assumed that the overall accuracy of the results is better. The higher consumption of cyanide in this test was not investigated, but may well be due to the presence of arsenic in the F-ore which is known to be a cyanicide.

Overall, the majority of the gold appeared to be liberated from both ore types within the first 30 hours of the tests, (which is consistent with the findings from Warren Springs Laboratory). However it is very difficult to fully interpret all the leaching results as two of the tests on each sample were carried out at unusually coarse grind sizes, both of which produced poor gold recoveries. The best recoveries were found wth 'pulverised' ore, the size analysis of which is not defined in the report, making a direct comparison with other testwork very difficult.

10.4 Summary

The general conclusions from the testwork are that both L and F ore types will readily leach within 30 hours and that the better recoveries occur with finely ground ore. Cyanide consumption was found to be highest with the arsenic rich F-ore possibly due to the interaction of cyanide and arsenic. In addition it is recommended that further investigations be carried out to determine the optimum grind size and to establish reasons for the high cyanide consumption with F-type ore.

TABLE 6 - SUMMARY OF HITEC TESTWORK RESULTS

Sample 1	L-ore	Head Grade	0.947 g/t Au	
Feed Size (inches)		-3/16	-1/8	
(mm)		-4.763	-3.175	Pulverised
Cyanide Consumption after 47 hours		2.12 lb/ton 1.08 kg/t	2.16 lb/ton 1.10 kg/t	2.16 lb/ton 1.10 kg/t
% Gold Extraction after 47 hours		39.65%	68.83%	83.15

Sample 2	L-ore	Head Grade	1.557 g/t Au	
Feed Size (inches)		-3/16	-1/8	
(mm)		-4.763	-3.175	Pulverised
Cyanide Consumption After 47 hours		2.2 lb/ton (1.12 kg/t)	2.12 lb/ton (1.08 kg/t)	2.2 lb/ton (1.12 kg/t)
% Gold Extraction After 47 hours		35.04	70.17	100.0

Sample 3	F-Ore	Head Grade	6.579 g/t Au	
Feed Size (inches)		-3/16	-1/8	
(mm)		-4.763	-3.175	Pulverised
Cyanide Consumption After 47 hours		2.12 lb/ton (1.08 kg/t)	2.55 lb/ton (1.30 kg/t)	2.51 lb/ton (1.28 kg/t)
% Gold Extraction After 47 hours		37.11	58.78	80.98