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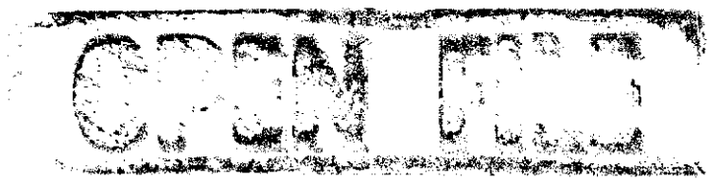
R 684 GOLD RECOVERY TESTS-
BEACONSFIELD

DEPARTMENT OF MINES

17/73



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PART B.
Reference 3/1



AMAX IRON ORE CORPORATION
(MINERALS EXPLORATION DIVISION)
55 MACQUARIE ST., SYDNEY. N.S.W. 2000

15th October, 1974.



R684

# Gold Recovery Tests - Beaconsfield

## Allstate Exploration N.L.



## Introduction

Gold recovery tests were required by Allstate Exploration N.L. on five samples of diamond drill taken from intersections of the Beaconsfield orebody. Three of the samples were from holes put down by the Tasmanian Government Department of Mines and two samples were from Allstate drilling.

## Sample Preparation

All the available individual core samples were used to make the intersection composites for the recovery tests, and therefore were not weighted proportionately. The following table shows the assays for each intersection and the calculated heads of the samples used in the recovery tests.

| D.D.H. No. | B4 | | B4A | | B4B | | A601-11 | | H6WL | |
|---|---|---|---|---|---|---|---|---|---|---|
| SAMPLE | 660984 | TEST | 662821 | TEST | 671015 | TEST | 740911 | TEST | 740852 | TEST |
| Au g/t | 90 | 167 | 60 | 56.5 | 40 | 61 | 11 | 14.6 | 38 | 35.3 |
| Ag g/t | 7 | 18.1 | 9 | 3.8 | 16 | - | - | 2.9 | 11 | 6.2 |
| As % | 1.5 | 0.95 | 0.4 | 0.51 | 0.05 | - | 1 | 0.74 | 0.7 | 0.72 |
| Cu % | 1.1 | 1.43 | 0.9 | 0.98 | 1.1 | 1.3 | 0.8 | 0.89 | 0.5 | 0.50 |
| S% | 7.5 | 8.4 | 4.9 | 4.4 | 3.6 | - | 5 | 5.0 | - | 13.2 |

Each sample was roll crushed in closed circuit with a 1 mm. screen until all the material passed through the screen.

## Test Work and Results

### 1. Test N1 - sample Reg. No. 671015 - D.D.H. B4B

Core from diamond drill hole B4B was selected for a preliminary test, because of its low arsenic content, and was in this aspect different from the remaining samples.

The whole sample was passed over a Denver laboratory mineral jig (No. 1M). The jig tailing was ground in the 8" diam. x 8" Warman laboratory ball mill, and was then floated in the Denver D1 laboratory flotation cell, using 3 kg/tonne of sodium carbonate and 0.2 kg/tonne of sodium ethyl xanthate, and pine oil as frother to produce a bulk sulphide concentrate. This concentrate was re-floated in a cleaning operation. The cleaned concentrate was re-floated using sodium cyanide at the rate of 0.1 kg/tonne of rougher feed as a pyrite depressant to form a cppper concentrate. The tail thus formed was a pyrite concentrate.

The initial flotation test to produce the bulk sulphide concentra was very active and at the time was thought to be due to the choice of frother.

At this stage of the investigation, copies of Amdel reports MP 3634/74 and MP 3929/74 were received and it was noted that no gold was found by visual examination. The jig concentrate was then examined under the microscope and free gold, gold/ pyrite composites and

gold/quartz composites were found and were removed by handpicking.

The remainder of the jig concentrate was ground with a pestle and mortar to pass through a 250 µm screen. The ground product was then floated using Aerofloat 238 as collector and Teric 401 as frother to produce a copper concentrate. A pyrite concentrate was then floated off using sulphuric acid and potassium amyl xanthate. In the flotation of the jig concentrate there was no sign of over-activity in flotation as had been noticed in the bulk-sulphide flotation of the ground jig tail.

The results of the bulk-sulphide flotation test were as follows:-

| Product | Mass | Assays | | Distribution % | |
|---|---|---|---|---|---|
| | % | Au g/t | Cu % | Au | Cu |
| F7C | 1.82 | 339 | 24.9 | 10.1 | 34.5 |
| F7T | 6.35 | 178 | 7.9 | 18.5 | 38.1 |
| F6C | 8.17 | 214 | 11.7 | 28.6 | 72.6 |
| F6T | 4.51 | 26 | 0.87 | 2.0 | 3.0 |
| F5C | 12.68 | 147 | 7.8 | 30.6 | 75.6 |
| F5T | 84.48 | 9.5 | 0.07 | 12.9 | 4.5 |
| J1T | 97.16 | 27 | 1.08 | 43.5 | 80.1 |
| J1C | 2.84 | 1210 | 9.2 | 56.5 | 19.9 |
| H | 100.0 | 61 | 1.3 | 100.0 | 100.0 |

Flotation of the jig concentrate after hand-picking gold from it, gave the following results.

| Product | Mass | Assays | | Distribution % | |
|---|---|---|---|---|---|
| | % | Au g/t | Cu % | Au | Cu |
| H/F2C | 0.0 | ** | | 27.1 | - |
| F3C | 0.71 | 250 | 22.5 | 2.8 | 12.5 |
| F4C | 2.68 | 800 | 4.5 | 26.6 | 7.4 |
| F4T | 0.05 | - | - | - | - |
| J1C | 2.84 | 1240 | 8.9 | 56.5 | 19.9 |

** Clean metallic gold, not assayed for silver. Assumed 100% Au for calculation purposes.

2. Tests N2 and N3 - sample Reg. No. 660984 D.D.H. B4

Before treatment of this sample, the depth of ragging in the jig was altered so that less sulphide material would be pulled into the concentrate and a higher grade of concentrate would be produced.

The sample from diamond drill hole B4 was passed over the jig and the weight of concentrate was greatly reduced. The free and composited gold was hand-picked out of the jig concentrate and the jig bed in the jig at the end of the test, and the remaining concentrate and jig bed material was put with the jig tail.

The jig tail was then halved by riffling, and one half was ground for five minutes in the ball mill, and the other half was ground for fifteen minutes in the ball mill.

After 15 minutes conditioning with 1 kg/tonne of sodium sulphite, flotation, with 0.05 kg/tonne of Aerofloat 238 and Teric 401 as a frother, produced a copper concentrate, and 0.1 kg/tonne

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of potassium amyl xanthate was used to produce the pyrite concentrate. However, the over-active condition was again present in the copper flotation as in the bulk sulphide flotation in test M1. This overactivity resulted in the copper concentrates being low grade.

The results of the two tests with coarse and fine grinding are as follows:-

| | | Mass % | Assays Cu% | Au g/t | Ag g/t | S% | As% | Distribution % Cu | Au | Ag | S | As |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Free gold | | | * | * | | | | 65.1 | | | |
| N2 | P1C | 16.4 | 5.7 | 192 | 22 | 34.7 | 3.32 | 77.7 | 19.5 | 23.2 | 68.5 | 56.7 |
| Coarse Grind | P2C | 12.2 | 2.0 | 75 | 34 | 16.6 | 2.47 | 17.3 | 5.7 | 26.5 | 24.4 | 31.4 |
| | P2T | 71.4 | 0.10 | 22 | 11 | 0.82 | 0.16 | 5.0 | 9.6 | 50.3 | 7.1 | 11.9 |
| | calculated head | 100.0 | 1.41 | 160 | 15.6 | 8.30 | 0.96 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |
| | Free gold | | | * | * | | | | 60.4 | | | |
| N3 | P1C | 15.2 | 7.9 | 203 | 29 | 28.8 | 2.15 | 82.8 | 17.8 | 21.5 | 51.4 | 35.2 |
| Fine Grind | P2C | 19.1 | 1.2 | 153 | 29 | 20.4 | 2.98 | 15.8 | 17.0 | 27.2 | 45.8 | 61.3 |
| | P2T | 65.7 | 0.03 | 13 | 16 | 0.36 | 0.05 | 1.4 | 4.8 | 51.3 | 2.8 | 3.5 |
| | calculated head | 100.0 | 1.45 | 173 | 20.5 | 8.51 | 0.93 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |

* Clean metallic gold, not assayed for silver.

Assumed 100% Au for calculation purposes.

The results of tests N2 and N3 confirm that the over-active condition during copper flotation produced a low grade of concentrate. The results also indicate that the fine grind has produced better recoveries of sulphur and all the metals except silver.

3. Tests N4 & N5 - sample Reg. No.662821 - D.D.H. B4A
Tests N6 & N7 - sample Reg. No.740911 - A601 - 11
Tests N8 & N9 - sample Reg. No.740852 - H6WL 1 - 14

Each of the above samples was treated similarly in the following manner. The samples were passed over the jig, and the free and composited gold was hand-picked out of the concentrate and the jig bed, and the remaining material from these products was put with the jig tail. The jig tail was then riffled in half, and each half was ground in the laboratory ball mill for 15 minutes.

Because of the over-active condition of the flotation tests on the first two samples, it was decided to try pre-flotation. The ground sample was conditioned for fifteen minutes with 1 kg/tonne of sodium sulphite, and then floated without any collector type flotation reagent and with only sufficient frother, Teric 401, to maintain a froth.

During five minutes flotation, a flotation concentrate was made which was dirty in appearance. This concentrate may contain a low density free-floating non-metallic mineral such as talc, sericite, or similar material.

A copper concentrate was then produced with the addition of 0.05 kg/tonne of Aerofloat 238 and five minutes flotation, and a pyrite concentrate was produced with the addition of 0.1 kg/tonne of potassium amyl xanthate and ten minutes flotation.

The results of these tests are as follows:-

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| Pyrite Conc. | Calcine weight as % of pyrite conc. | Calcine Assay % As | As Distribution in calcine | As Distribution in exhaust | % Recovery in solution Au | % Recovery in solution Ag |
|---|---|---|---|---|---|---|
| B4A (N4 & N5) | 72.8 | 1.7 | 36.9 | 63.1 | 82.8 | nil |
| A601-11 (N6 & N7) | 81.9 | 3.2 | 79.4 | 20.6 | 48.4 | nil |
| H6WL1 - 14 (N8 &N9) | 69.1 | 0.82 | 27.6 | 72.4 | 79.6 | 10.0 |

NB

Cyanidation tests were also done on the flotation tailings.

Composite flotation tailings for each diamond drill hole were made by taking 500 grams each of flotation tailings from the duplicate tests and mixing, viz., from N4 and N5, N6 and N7, and N8 and N9.

The tailings were agitated for 24hours in a litre of solution containing 0.5% sodium cyanide, and lime to give a pH of 11 to 12.

The results of the cyanidation tests on flotation tailings are as follows:-

| Flotation Tailing | Cyanide Consumption kg/tonne of tailing | Gold recovery in solution gms/tonne | Gold recovery in solution % |
|---|---|---|---|
| B4A (N4 & N5) | 1.6 | 0.55 | 54.5 |
| A601 - 11 (N6 & N7) | 0.2 | 0.66 | 22.9 |
| H6WL1 - 14 (N8 & N9) | 0.2 | 1.80 | 31.3 |

Conclusions

Free gold can be recovered by jigging. The amount recovered in this manner is largely dependent on the head value of the ore.

There is a free-floating gangue mineral present in the ore. It should be advantageous to remove this material by pre-flotation. Some copper tends to float with this material, but gold does not. When ore becomes available, more extensive flotation test work should be carried out in order to determine the best method of coping with this free-floating material and endeavouring to plave more copper into the copper concentrate.

Copper flotation yielded concentrates assaying about 400 grams of gold per tonne and about 15% copper. Cleaning and recleaning of the copper concentrate should appreciably raise these assays. About 20% of the gold present is recovered in the copper concentrate and this figure is somewhat dependent on the head grade of the ore.

The pyrite concentrate carried appreciable quantities of gold, After calcining the pyrite concentrates and cyanidation, about 80% of the gold in the pyrite concentrates can be recovered.

There is little point in cyanidation of the flotation tailings as there is not much gold present, and recovery of this gold is low.

Diamond drill hole A601-11 was the lowest in grade, and it was also the most difficult to treat. About 16% of the gold present in this ore found its way into the flotation tailing. Cyanidation of the tailing gave the lowest recovery - 22.9%, and cyanidation of the calcine of the pyrite concentrate also gave the lowest recovery - 48.4%.

It is planned to try cyanidation of the pyrite concentrate without roasting to compare recoveries. The result of this test will be reported separately.

Senior Metallurgist.

Chief Chemist & Metallurgist.

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607607

19th November, 1974.

Allstate Exploration NL.,
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Dear Sirs,

R 684

Please find attached results of further test work on above project.

Yours faithfully,

(H.K. Wellington)
Chief Chemist & Metallurgist.

Encls.

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R 684 Part (ii)

Gold Recovery Tests Beaconsfield

Allstate Exploration N.L.

Tonks = 116, 117

Cyanidation Tests

A cyanidation test was carried out on pyrite concentrate made in tests N8 and N9 from diamond drill core from sample 740852. A composite of 50 grams from each of these tests was agitated for 16 hours in a beaker by means of a magnetic stirrer in 600 mls of solution containing 0.2% potassium cyanide and 0.1% lime. The results are as follows together with the results of the cyanidation of the calcine for comparison.

| | % Recovery in solution | |
|---|---|---|
| 740852(H6WL 1-14) | Au | Ag |
| calcine | 79.6 | 10.0 |
| pyrite concentrate | 86.0 | 14.0 |

There was insufficient pyrite concentrate from the other bore holes to compare direct cyanidation of the pyrite concentrate with cyanidation of the calcine.

The result obtained on 740852 indicates that on this sample of ore there is no benefit gained in calcining the pyrite concentrate prior to cyanidation.

A sizing analysis on the flotation tail from test N3 on 662821 (B4A) is as follows:-

| Screen Aperture(µm) | % mass | % mass cum. |
|---|---|---|
| +212 | 0.2 | 0.2 |
| +150 | 1.5 | 1.7 |
| +106 | 8.9 | 10.6 |
| +75 | 13.1 | 23.7 |
| +53 | 13.8 | 37.5 |
| +38 | 13.5 | 51.0 |
| -38 | 49.0 | 100.0 |

(L.J. Rhodes)
Senior Metallurgist

(H.K. Wellington)
Chief Chemist & Metallurgist.

| | | Mass | Assays | | | | | Distribution % | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | % | Cu % | Au g/t | Ag g/t | S % | As % | Cu | Au | Ag | S | As |
| | Free Gold | | | * | * | | | | 57.7 | | | |
| | F1C | 2.79 | 10.5 | 8 | 13 | 12.8 | 0.34 | 30.2 | 0.4 | 8.7 | 8.9 | 1.8 |
| M4 | F2C | 2.51 | 15.6 | 520 | 61 | 22.8 | 3.3 | 40.4 | 22.6 | 36.9 | 14.2 | 16.1 |
| | F3C | 10.47 | 2.0 | 100 | 11.5 | 26.0 | 3.4 | 21.6 | 18.2 | 29.0 | 67.5 | 69.0 |
| | F3T | 84.23 | 0.09 | 0.75 | 1.25 | 0.45 | 0.08 | 7.8 | 1.1 | 25.4 | 9.4 | 13.1 |
| | Calc. head | 100.0 | 0.97 | 57.7 | 4.2 | 4.0 | 0.52 | 100.0 | 100.0 | [illegible] | 100.0 | 100.0 |
| | Free Gold | | | * | * | | | | 60.2 | | | |
| | F1C | 2.15 | 9.1 | 18 | 2[illegible] | 12.1 | 0.38 | 19.8 | 0.7 | 1.3 | 5.5 | 1.7 |
| M5 | F2C | 2.76 | 18.0 | 372 | 53 | 20.2 | 2.9 | 50.1 | 18.6 | 42.5 | 11.8 | 16.1 |
| | F3C | 10.05 | 2.2 | 102 | 8.7 | 35.0 | 3.3 | 22.4 | 18.6 | 25.3 | 74.3 | 66.8 |
| | F3T | 85.04 | 0.09 | 1.25 | 1.25 | 0.47 | 0.09 | 7.7 | 1.9 | 30.9 | 8.4 | 15.4 |
| | Calc. head | 100.0 | 0.99 | 55.2 | 3.4 | 4.7 | 0.50 | 100.0 | [illegible] | [illegible] | 100.0 | 100.0 |
| | Free Gold | | | * | * | | | | 15.1 | | | |
| | F1C | 2.70 | 7.6 | 5 | 4 | 7.3 | 0.48 | 22.5 | 0.9 | 4.0 | 4.0 | 1.8 |
| | F2C | 3.34 | 14.8 | 85 | 24 | 23.6 | 1.2 | 54.0 | 19.0 | 29.5 | 16.2 | 5.4 |
| M6 | F3C | 11.01 | 1.5 | 65.3 | 1.3 | 28.3 | 3.3 | 18.1 | 48.3 | 5.4 | 64.1 | 49.7 |
| | F3T | 82.95 | 0.06 | 3 | 2 | 0.92 | 0.38 | 5.4 | 16.7 | 61.1 | 15.7 | 43.1 |
| | Calc. head | 100.0 | 0.91 | 14.9 | 2.7 | 4.9 | 0.73 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |
| | Free Gold | | | * | * | | | | 15.7 | | | |
| | F1C | 3.55 | 5.7 | 6.7 | 2.7 | 7.4 | 0.50 | 23.4 | 1.6 | 3.1 | 5.1 | 2.4 |
| | F2C | 2.30 | 19.2 | 88 | 50 | 24.8 | 1.3 | 51.0 | 14.2 | 37.0 | 11.1 | 4.0 |
| M7 | F3C | 14.63 | 1.3 | 57 | 10 | 26.3 | 3.3 | 21.9 | 53.2 | 47.1 | 74.8 | 64.8 |
| | F3T | 79.52 | 0.04 | 2.75 | 0.5 | 0.58 | 0.27 | 3.7 | 15.3 | 12.8 | 9.0 | 28.8 |
| | Calc. head | 100.0 | 0.87 | 14.3 | 3.1 | 5.1 | 0.75 | 100.0 | 100.0 | 100.0 | 100.0 | 100.0 |
| | Free Gold | | | * | * | | | | 17.8 | | | |
| | F1C | 1.41 | 2.3 | 9.7 | 17.5 | 9.9 | 0.37 | 6.5 | 0.5 | 4.0 | 1.1 | 0.7 |
| M8 | F2C | 2.56 | 13.7 | 428 | 102 | 37.3 | 0.74 | 71.0 | 37.7 | 42.4 | 7.8 | 2.6 |
| | F3C | 29.71 | 0.33 | 36.5 | 10 | 36.9 | 2.2 | 19.8 | 37.2 | 48.2 | 87.8 | 89.4 |
| | F3T | 66.32 | 0.02 | 3 | 0.5 | 0.60 | 0.08 | 2.7 | 6.8 | 5.4 | 3.3 | 7.3 |
| | Calc. head | 100.0 | 0.49 | 29.1 | 6.2 | 12.4 | 0.73 | 100.0 | 100.0 | 100.0 | [illegible] | [illegible] |
| | Free Gold | | | * | * | | | | 12.5 | | | |
| | F1C | 1.36 | 2.4 | 4.8 | 18.3 | 9.8 | 0.35 | 6.5 | 0.1 | 4.0 | 1.0 | 0.7 |
| M9 | F2C | 2.80 | 12.4 | 412 | 79 | 43.6 | 0.85 | 69.6 | 27.9 | 35.9 | 8.7 | 3.4 |
| | F3C | 26.62 | 0.37 | 70.5 | 10 | 44.9 | 1.9 | 19.7 | 45.3 | 43.2 | 85.4 | 71.5 |
| | F3T | 69.22 | 0.03 | 8.5 | 1.5 | 1.0 | 0.25 | 4.2 | 14.2 | 16.9 | 4.9 | 24.4 |
| | Calc. head | 100.0 | 0.50 | 41.4 | 6.2 | 14.1 | 0.71 | 100.0 | 100.0 | [illegible] | [illegible] | [illegible] |

* Clean metallic gold not assayed for silver. Assumed to be 100% Au for calculation purposes.

Subsequently the free gold taken by treatment of the sample H6WL1 - 14 was assayed to find the silver content, and was found to be 99.7% Au and 0.3% Ag. Because of this assay no adjustments were made to the metal balances where the free gold had been assumed to be 100% gold.

The results show that the pre-flotation concentrate contains between 20% and 30% of the copper for the diamond drill holes B4B and A601 - 11 and 6.5% of the copper for diamond drill hole W6WL1 - 14 Less than 1% of the gold present in the drill holes tested, reports in the pre-flotation concentrate.

After having taken off the pre-flotation concentrate, there was no trouble in producing a copper concentrate of reasonable grade, ranging from 12.4% to 19.2% in all the tests. The concentrate produced was a rougher concentrate, and there should be no problem in upgrading this to a concentrate of saleable grade by cleaning and recleaning flotation. Over 70% of the copper has been received in the pre-flotation concentrate plus the copper concentrate. A further 20% of the copper is recovered in the pyrite concentrate.

Gold recovery is a little irregular varying from 83% to 99% in the jig and flotation concentrates, depending largely on the head grade. It may be more useful to examine the tailing assays. In all tests the gold assay of the tailing was 3 grams/tonne or less except test N9 in which the assay was 8.5 grams/tonne. The calculated gold head assays of tests N8 and N9 do not duplicate very well, and re-assaying of the products in the tests did not make any appreciable difference. The calculated sulphur head assays in these tests did not duplicate very well either.

Silver recoveries were also erratic varying from 69% to 94% in the flotation concentrate except in test N6 where the recovery was 39%.

The major proportion of the arsenic reported in the pyrite concentrate.

4. Cyanidation Tests.

Cyanidation tests on roasted pyrite concentrates were then undertaken to determine the recovery of gold from these concentrates.

Composite pyrite concentrates for each diamond drill hole were made by taking 50 grams each of pyrite concentrates from the duplicate tests and mixing, viz., from N4 and N5, N6 and N7, and N8 and N9.

The concentrates were then roasted at about 900°C until sulphur dioxide could no longer be detected.

The calcines were then agitated for 16 hours in beakers by means of magnetic stirrers, in 600 mls of solution containing 0.2% potassium cyanide and 0.1% lime.

The results of the cyanidation tests on calcine are as follows:-