GS 1986 026

DEPARTMENT OF MINERAL RESOURCES

NEW SOUTH WALES

MINERAL RESOURCES DEVELOPMENT LABORATORY

IRONCLAD MINE, CARGO, NSW

- PRELIMINARY STUDIES OF GOLD EXTRACTION FROM GOLD ORES

J. R. S. Kmetoni

Senior Scientific Officer

January 1986

Mineral Resources Development Laboratory

Report 86/1

PO Box 76, Lidcombe, NSW, 2141

GS 1981 026

Ironclad Mine

Cargo, NSW

Part 1. Preliminary study of gold extraction from Underground (sulphide) ore.

Part 2. Preliminary study of gold extraction from Open pit (oxidized) ore.

Part 3. Conclusions.

D000870820

GS 1986 026

Officers participating in the investigation:

Mineral Processing: Dr J. R. S. Kmetoni, K. Ferguson, F. Pall

XRD analyses: Dr J. R. S. Kmetoni, K. Ferguson

Microscopy: Dr J. R. S. Kmetoni

Heavy Metals analyses: I. Hodges

XRF analyses: D. Karaolis, L. Dowswell

Sulphur analyses: J. Hobbs, A. Hanna

GS 1986 028

Abstract

Preliminary studies of gold extraction from gold ore from Ironclad Mine at Cargo, New South Wales were carried out.

Gold ore samples were of two different types: primary sulphide ore from underground workings and secondary oxidized ore from open pit workings.

In the initial phase of the investigation the basic characteristics of the ores, related to the contained gold, were established. The amenability of the ores to suitable gold extraction processes was tested, including cyanidation, flotation, gravity separation, roasting and combinations of the methods.

It is concluded that it is technically feasible to achieve recoveries of gold well in excess of 90% from both ores.

Because of the different characteristics of each of the two ores, no single tested method was suitable for gold extraction from both ores. A combined treatment flowsheet is proposed. However the viability of such an operation will depend upon instalation/processing costs and market prices for products.

D000870830

GS 1986 026

Contents

		Page
1.	Preliminary study of gold extraction from underground sulphide ore	1
1.1	Basic characteristic of the ore	2
1.2	Heavy Media Separation	4
1.3	Cyanidation	10
1.4	Flotation	12
1.5	Gravity Separation - Tabling	18
1.6	Combination of flotation and tabling of flotation tailings	23
1.7	Combination of flotation and cyanidation of flotation tailings	24
1.8	Upgrading of the Flotation Concentrate	25
1.9	Discussion	28
2.	Preliminary study of gold extraction from open pit oxidized ore	31
2.1	Basic characteristic of the ore	32
2.1.a	Sample collection and composition	32
2.1.b	Sample preparation and analyses	32
2.2	Heavy Media Separation	37
2.3	Gravity Separation - Tabling	42
2.4	Cyanidation	46
2.4.a	Direct cyanidation of the ore	46
2.4.b	Cyanidation of gravity separation tailings	49
2.5	Discussion	51

D000870840

G 19[illegible]026

3. Conclusions 53

3.1 Sulphide - Underground Ore 54

3.2 Oxide - Open Pit Ore 55

3.3 Combined (Sulphide, Oxide) Processing 56

GS 1986 026

Part 1

Preliminary study of gold extraction from underground sulphide ore

D000870850

Introduction

The Mineral Resources Development Laboratory was requested for assistance to carry out preliminary studies of gold extraction from gold ores from Ironclad Mine, Cargo, NSW.

The leaseholders, Messers I. R. and W. J. R. Finlayson hold G.M.L. 5828 over portions G.L. 137 and G.L. 163, parish Cargo, county Ashburnham; map reference Si55-8 8631 11. The lease lies northwest of the village of Cargo, which is located between Orange and Canowindra, about 360 km west of Sydney.

Mining of the lode predates year 1875 and altogether 172 kg of gold was produced from the mine. The present operations are to the south of the old Ironclad workings, the largest of the underground lode mines in the Cargo Goldfields.

The leaseholders over the past 2-3 years worked on refurbishing the shafts in order to regain access to the old workings and the project has reached the stage where decisions will have to be taken about establishing an ore processing plant.

The request for assistance was submitted by the regional mining engineer in Orange; samples of the ore from the mine were supplied by the Geological Survey of NSW, the Lachlan-Metallic Section.

GC 1986 926

1.1 Basic characteristics of the ore

Two samples of sulphide ore from underground workings of Ironclad Mine were supplied by the Lachlan-Metallic Minerals Section of the Geological Survey of NSW. The channel samples were taken from the western face of the drive on the 300 feet level, before the underground workings were flooded. The samples were marked "Lower" and "Upper", referring to the position where from the face the sample was taken. Since the underground workings are flooded and inaccessible for sampling, these samples were the only recent ore samples available. After consultation with the Lachlan-Metallic Minerals Section personnel it was decided to prepare a "run of mine" ore composite sample for the metallurgical investigation by mixing the two samples on the basis of 1:1 ratio by weight.

The samples were separately crushed in two stages of jaw crushing and in the third stage crushing using a rolls crusher; the ore samples were reduced to -3 mm particle size. Then the individual samples were quartered, until representative sample of the each ore sample was obtained. From the remainder of the samples, equivalent portions by weight were taken and homogenised in a tumble mill. Another sample from the homogenised ore was quartered for analytical purposes.

The three quartered samples were then chemically analysed (Table 1) and parts of the samples subjected to X-ray diffraction (powdered specimens) for determination of the main mineral constituents. Other parts of the samples were microscopically examined.

Table 1

Sample	Assays						
	g/t		%				
	Au	Ag	Cu	Pb	Zn	As	Sb
U/G "Upper"	13.9	3.8	0.125	0.004	0.004	0.045	<0.002
U/G "Lower"	22.6	7.8	0.34	0.004	0.002	0.057	<0.002
U/G Composite	16.7	4.4	0.17	0.004	0.003	0.056	<0.002

The XRD revealed a substantial presence of pyrite, pyrhotite as the main metallic minerals and quartz as the main gangue mineral. Microscopic examination corroborated the XRD finds and in addition a small content of a mineral resembling chalcopyrite was detected. No free gold was found by the microscopic examination.

Since the natural form in which gold is occurring in the ore is of the most crucial importance for a gold extraction process, further tests were carried out to ascertain the preliminary conclusion that the gold is closely associated with the sulphide minerals. Consequently a sample of the composited ore was sieved into particle size fractions (the +600 micron fraction was rejected) and the individual fractions were assayed for contents of gold and other metals (Table 2), in order to obtain an indication whether, in which particle size range and with which other metals the gold tends to associate.

D000870860

Table 2

Particle Size microns	Assays						
	Au g/t	Ag g/t	Cu %	Pb %	Zn %	As %	Sb %
-600 +425	19.0	8.0	0.54	0.007	0.005	0.049	<0.002
-425 +300	16.2	8.6	0.36	0.004	0.003	0.061	<0.002
-300 +150	16.1	7.6	0.30	0.003	0.002	0.068	<0.002
-150 +75	22.0	6.0	0.25	0.003	0.002	0.059	<0.002
-75	18.1	5.5	0.23	0.004	0.003	0.069	<0.002

Silver and copper contents tend to decrease with the reduction in particle size which would indicate that silver is probably associated with copper bearing minerals. The gold contents do not indicate any particular trend of concentration with increase or decrease in particle size or any association with other metals. The fact that the content of gold in the finest particle size fraction (0.75 microns) is not higher than in the other fractions could be interpreted as the absence of liberated free gold.

1.2 Heavy Media Separation

A substantial sample of the composited underground ore was dry sieved to the following particle size fractions: -600 +425 microns, -425 +300 microns, -300 +150 microns, -150 +75 microns. The individual fractions were then wet sieved at the appropriate

GS 1986 026

sieve aperture (425, 300, 150, 75 microns) and air dried. The dry fractions were then subjected to a heavy media separation (HMS) in the heavy liquid "Tetrabromoethane" of 2.94 S.G.

The HMS separations of the coarse fractions (+300 microns) were conducted in beakers, the HMS of the fine fractions (-300 microns) were conducted in a separating funnel. All the separations of the fractions were carried out in small portions, maintaining solids:liquid ratio at approximately 1:10, in an effort to facilitate as good as practicable separations. The products of the HMS were vacuum filtered, several times washed on the filters by ethyl-alcohol, dried, weighed, samples prepared for assaying of metal contents, powdered specimens analyzed by XRD for mineral compositions and the individual HMS products microscopically examined.

Portions of the "sink" (heavier than S.G. 2.94) fractions were then subjected to HMS in Clerici's solution (Thallium Formate + Thallium Malonate) at S.G. 4.65. The separations were conducted at the elevated temperature of 50°C, using a separating funnel enveloped in a water jacket through which water at 52°C was pumped in a closed circuit with a Thermomix and reservoir, in order to maintain the temperature of the Clerici's solution at constant 50°C. The HMS products were vacuum filtered at elevated temperatures >50°C, washed several times with hot water, dried, weighed and the similar sample preparation and analyzing procedures as described before were applied.

The -75 micron fraction was not separated by HMS because of the inherent difficulties to separate fine particles in heavy liquids, especially at higher S.G.

The results of the HMS are presented in the Tables 3, 4, 5 and 6.

D000870870

Table 3

HMS -600 +425 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	60.6	28.8	96.3
Float S.G. 2.9	39.4	1.7	3.7
Feed to S.G. 2.9	100.0	18.1	100.0
--- . ---			
Sink S.G. 4.6	90.5	30.8	96.8
Float S.G. 4.6	9.5	9.8	3.2
Feed to S.G. 4.6	100.0	28.8	100.0
--- . ---			
Float S.G. 2.9	39.4	1.7	3.7
Float S.G. 4.6	5.7	9.8	3.1
Sink S.G. 4.6	54.9	30.8	93.2
Feed to HMS	100.0	18.1	100.0

D000870880

6 . . 16025

Table 4

HMS -425 +300 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	62.6	34.1	47.1
Float S.G. 2.9	37.4	1.7	2.9
Feed to S.G. 2.9	100.0	22.0	100.0
---	---	---	---
Sink S.G. 4.6	94.2	35.8	98.9
Float S.G. 4.6	5.8	6.7	1.1
Feed to S.G. 4.6	100.0	34.1	100.0
---	---	---	---
Float S.G. 2.9	37.4	1.7	2.9
Float S.G. 4.6	3.6	6.7	1.1
Sink S.G. 4.6	59.0	35.8	96.0
Feed to HMS	100.0	22.0	100.0

Table 5

HMS -300 +150 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	63.0	24.7	96.8
Float S.G. 2.9	37.0	1.4	3.2
Feed to S.G. 2.9	100.0	16.1	100.0
--- . ---			
Sink S.G. 4.6	93.5	26.0	98.3
Float S.G. 4.6	6.5	6.5	1.7
Feed to S.G. 4.6	100.0	24.7	100.0
--- . ---			
Float S.G. 2.9	37.0	1.4	3.2
Float S.G. 4.6	4.1	6.5	1.7
Sink S.G. 4.6	58.9	26.0	95.1
Feed to HMS	100.0	16.1	100.0

Table 6

HMS -150 +75 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	58.3	26.4	94.9
Float S.G. 2.9	41.7	2.0	5.1
Feed to S.G. 2.9	100.0	16.2	100.0
--------- . ---------			
Sink S.G. 4.6	96.5	27.0	98.8
Float S.G. 4.6	3.5	9.1	1.2
Feed to S.G. 4.6	100.0	26.4	100.0
--------- . ---------			
Float S.G. 2.9	41.7	2	5.2
Float S.G. 4.6	2.0	9.1	1.1
Sink S.G. 4.6	56.3	27.0	93.7
Feed to HMS	100.0	16.2	100.0

The results of the heavy media separations support the preliminary conclusions that the gold is closely associated with the sulphide minerals. There are also indications that the gold is finely disseminated within the sulphide minerals. This appears to be the limiting factor in an effort to increase concentration of gold in the concentrates of heavy minerals. Although the content of sulphide minerals in this composited sample of ore may be higher than the content of the minerals in a truly representative "run of mine" ore sample, there is still a distinct probability that the gold content within the sulphide minerals will vary very little and consequently the concentration of gold in sulphide concentrates (even in pure sulphides) will remain relatively low. For this reason it became apparent that separation of gold bearing sulphide minerals could yield reasonable gold recoveries but gold grades of the sulphide concentrates will be low. The logical solution would be to extract gold from the sulphide minerals on a selective basis. Consequently some other methods involving this principle had to be tested.

1.3 Cyanidation

The initial cyanidation test was conducted on a portion of the -75 micron ore fraction. The cyanide solution of 0.2% NaCN was used; solids:liquid ratio 1:2.5. The cyanidation was conducted in a beaker, the immersed mechanical stirrer maintained the solids in suspension and through the vortex created by the slurry in circular motion a continuous aeration of the slurry was accomplished; the impeller of the stirrer dispersed the air adequately. The cyanidation time was 24 hours. Alkalinity of the slurry was kept at pH 9-10 by addition of lime.

The slurry after the cyanidation was vacuum filtered, the solids washed on the filter by water. The filter cake was dried and assayed for gold content.

D000870890

The feed into the cyanidation assayed 19.0 g/t Au; the cyanidation tailings assayed 13.3 g/t Au. The corresponding calculated recovery of gold was 30% only.

Although the pregnant liquor was not assayed, it was noticed that the liquor had a blueish tinge; consequently a hypothesis was assumed that a preferential leaching of copper could take place, consuming cyanide and leaving gold undissolved in the solids.

On this assumption the further cyanidation tests were conducted with increased quantity of cyanide solution and a higher concentrations of cyanide in the solutions.

The composite sample was sieved to the following particle size fractions: -425 +300 microns, -300 +150 microns, -150 +75 microns. All of the fractions were cyanided at solids:liquid ratio of 1:6, concentration of NaCN in the solution 0.3%, cyanidation time 24 hours. The cyanidations were conducted in beakers. Solids in the slurries were kept in suspension by mechanical stirrers, air was supplied by a blower and dispersion of the air was achieved by the impellers of the stirrers; pH was maintained in the 9-10.5 range by lime.

Table 7

The results of particle size fractions cyanidations

Particle Size Fraction	Assays Au g/t		Au Recovery %
	Feed	Tailings	
-425 +300 microns	22.0	7.00	68.2
-300 +150 microns	16.1	5.10	68.4
-150 +75 microns	16.2	6.05	62.7

D000870900

The results confirmed that the cyanidation is not sufficiently effective to secure adequate recovery of gold. Consequently a sample of -75 micron ore was roasted and the roasted sample cyanided. Roasting of sulphide minerals containing gold should improve access of cyanide solution to the finely disseminated gold particles improving the effectiveness of cyanidation.

A sample of -75 micron composite ore, assaying 19.0 g/t Au was roasted at 700°C for two hours. The reduction of the mass of the original sample, caused by the burning of the sulphur content, was 15% by weight. Then the sample was cyanided in 0.2% NaCN solution at solids:liquid ratio of 1:9 for 24 hours at pH 10-11, using a beaker, stirrer, blower arrangement. The cyanidation tailings assayed 0.49 g/t Au, which means that the gold recovery was 97.4%. The results indicate, that using a combination of roasting followed by cyanidation of the calcined ore, it is technically feasible to achieve satisfactory gold recoveries. However the other considerations, such as: the investment required for this type of a processing plant and the necessary environmental safeguards even for a small scale production plant, would make the intended exploitation of the deposit uneconomic. The same would probably apply if the ore could be transported to the nearest major pyrometallurgical plant.

1.4 Flotation

The next possible processing method tested, in the effort to achieve a satisfactory recovery of gold at an acceptable grade of the gold containing product, was flotation.

The initial orientation flotation test was conducted at the following conditions: A sample of ore was ground in a ball mill at the density of 68% solids. The ground slurry was diluted to 30% density, pH adjusted to pH 5 by sulphuric acid, conditioned

D000870910

for 2 minutes with the addition of 200 g/t of Sodium Isobutyl Xanthate - SIBX (collector) and prior to the flotation Cresylic Acid (frother) was added; after 3 minutes of the flotation further addition of Cresylic Acid was required. During 5 minutes of flotation time the Concentrate 1 was collected. Then further 100 g/t of SIBX was added, 1 minute of conditioning applied and Cresylic Acid added. The second stage of the flotation was conducted for further 5 minutes, collecting the Concentrate 2, until the froth ceased to be mineralized.

The flotation products: Concentrate 1, Concentrate 2, Tailings were filtered, dried, weighed, samples for assaying prepared and assayed for gold and sulphur contents.

Table 8

Results of Flotation 1

Product	Weight %	Assays		Distribution %	
		Au g/t	S%	Au	S
Conc. 1	37.0	31	50.9	62.1	87.2
Conc. 2	5.5	52	30.6	15.5	7.8
Tailings	57.5	7.2	1.9	22.4	5.0
Feed	100.0	18.5	21.6	100.0	100.0

The results indicate that in the flotation products the sulphur/gold grades and the sulphur/gold recoveries (distribution) do not necessarily coincide. While the sulphur recovery into the concentrates is 95%, the gold recovery is only

D000870920

77.6%. The flotation regime was aimed, in principle, at the recovery of sulphide (iron) minerals and it was anticipated that the gold will follow; the actual results indicate that probably in the process a considerable part of the gold content was liberated (some probably only partially) and the flotation regime was not suitable for recovery of the liberated gold or midling particles. Consequently the flotation regime was revised with the view to enhance collection of gold and midling particles.

The grinding conditions were retained; the slurry was diluted to 30% density, pH adjusted to pH 5 by Sulphuric Acid. Then the slurry was conditioned for 2 minutes with the addition of 200 g/t of SIBX, then 200 g/t of Aero Promoter 404 (Cyanamid) was added, followed by 2 minutes conditioning and then Aerofloat 242 added as a frother/promoter. The flotation was conducted for 7 minutes. After this time froth began to thin, consequently further 100 g/t of Aero Promoter 404 and Aerofloat 242 were added and the flotation continued for 8 more minutes. The total flotation time was 15 minutes.

Table 9

Results of Flotation 2

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	41.6	31.8	85.0
Tailings	58.4	4.0	15.0
Feed	100.0	15.6	100.0

Although the recovery of gold improved, the losses of gold in the tailings remained unacceptable. The flotation was repeated using

the identical flotation regime, except the second addition of Aero Promoter 404 was omitted.

Table 10

Results of Flotation 2

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	40.8	36.7	83.5
Tailings	59.2	5.0	16.5
Feed	100.0	17.9	100.0

The tailings were then subjected to wet sieving and the resulting individual particle size fractions were assayed for gold contents.

GS 1986 02 8

Table 11

Particle size analyses of Flotation Tailings 3

Particle Size microns	Weight %	Au Assay g/t	Au Distribution %
+150	3.71	45	33.3
-150 +106	8.80	9.5	16.7
-106 +75	17.15	3.5	12.0
-75 +45	19.48	3.8	14.7
-45	50.87	2.3	23.3
Feed (Calc.)	100.0	5.0	100.0

The -106 micron fractions were analyzed for sulphur contents in order to establish in what form the gold is present in the fine fractions of the tailings. The results of the sulphur analyses:

-106 +75 microns 0.47% S
-75 +45 microns 0.30% S
-45 microns 0.36% S

The results indicate that the gold present in these fractions is most probably not associated with sulphide minerals, it is free gold which did not respond to the flotation treatment. However 50% of the lost gold in the tailings was in the coarser particles (>106 microns) accounting for only 12% of the tailings by weight. The gold grade also drops dramatically in the -106 micron fractions. These observations indicate a need for finer grinding, e.g. 100% to -106 microns, to facilitate an improved gold recovery by the flotation. However in the subsequent

flotation test the grinding conditions were retained while other variables (e.g. introduction of an activator) were tested.

In an effort to improve the gold (in particular the suspected free gold) recovery another flotation test was conducted with the addition of copper sulphate as an activator. The ground and diluted slurry was conditioned for 2 minutes with 500 g/t of $CuSO_4$ and towards the end of the conditioning H_2SO_4 was added to adjust pH to 5; then 200 g/t of SIBX was added and the slurry conditioned for another 2 minutes; prior to the 3rd 2 minutes conditioning 200 g/t of Aero Promoter 404 was added and finally the Aerofloat 242. After 7 minutes of flotation, Aerofloat 242 was added again in order to maintain an adequate froth column throughout the entire 15 minutes of the flotation. The results of the flotation are presented in the Table 12.

Table 12

Results of Flotation 4

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	43.1	32.8	90.7
Tailings	56.9	2.55	9.3
Feed	100.0	15.6	100.0

The recovery of gold improved to 90.7%, but it became obvious that modifications of a flotation process would improve gold recovery only marginally. Consequently another flotation test was conducted, identical with the flotation 3, in order to produce sufficient quantities of flotation concentrates and flotation tailings for further testing.

Table 13

Results of Flotation 5

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	43.0	32.2	87.3
Tailings	57.0	3.5	12.7
Feed	100.0	15.9	100.0

1.5 Gravity Separation - Tabling

The evaluation of the flotation tests indicated that fine grinding liberates a considerable proportion of gold and part of the liberated gold is lost in the flotation tailings. Theoretically at least a part of this liberated gold should be recoverable by gravity separation methods. On the basis of this premise, gravity separation was tested.

The Mozley Laboratory Mineral separator (a variation to a super-panner) was used to assess effectiveness of gravity separation as a treatment method for the gold ore. Although the separator act in a similar way as the hand panning of gold, the results indicated a positive response of the ore to gravity treatment (e.g. concentrating tables, spirals, Reichert cones etc.).

The initial test was conducted using a sample of sieved ore to -106 microns and the flat (fine) deck of the separator.

Table 14

Results of Tabling

Ore, -106 microns

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	31.4	48.5	84.1
Midlings	12.9	11.5	8.2
Tailings	55.7	2.5	7.7
Feed	100.0	18.1	100.0

The results are comparable with the results achieved by flotation, consequently a more thorough testing was conducted.

A substantial sample of the composite ore was crushed to -1 mm size and sieved to +106 micron (89.8% by weight) and -106 micron fractions (10.2%). The individual fractions were then tabled using the Mozley Lab. Mineral Separator with the flat (fine) deck for -106 micron fraction and the "V" (coarse) deck for the +106 micron fraction.

Table 15

Results of Tabling

Ore, -106 microns, Flat deck

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	26.8	41.9	79.4
Midlings	8.9	10.8	6.9
Tailings	64.3	3.0	13.7
Feed	100.0	14.2	100.0

The results confirmed the relatively good response of the -106 micron fraction of the ore to the tabling.

Table 16

Results of Tabling

Ore, -1 mm +106 microns, "V" deck

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	24.3	30.2	34.5
Midlings	7.3	30.4	10.5
Tailings	68.4	17.1	55.0
Feed	100.0	21.2	100.0

The tabling results of the coarse fraction (-1 mm +106 microns) were disappointing; it was understood that the testing was not conducted at the optimum conditions (compare HMS, Table 3) and there is some room for improvement of the gold recovery. However the desired 90+% gold recovery can not be achieved even at the most optimum conditions. The only solution would be to grind the ore to -106 microns and subject the slurry to fine gravity separation. However, it has to be remembered that the fine deck of the Mozley separator is a laboratory superpanner, which is very efficient at separation of fine particles (2-3 micron gold particles were detected in concentrates) and in this respect superior to the most of the equipment used by the mineral processing industry for gravity separation of fine particles. If gravity separation would be used as the sole processing method for the ore, it would result in unacceptably low gold recoveries.

Since the gravity separation of fine ore proved to be relatively effective (recovering free gold), an attempt was made to separate the remaining gold from the flotation tailings. The tailings from the Flotation 5 were wet sieved at 125 microns and the two resulting particle size fractions (+125 microns, -125 microns) were subjected to gravity separation using Mozley Lab. Separator with the "V" deck for the +125 micron fraction and with the flat deck for the -125 micron fraction. The +125 micron fraction accounted for 5.3%, the -125 micron fraction for 94.7% by weight of the tailings. The grade of the flotation tailings was 3.5 g/t Au. The results of the tablings are presented in Table 17 and 18.

D000870930

Table 17

Results of Tabling

Flotation Tailings 5, +125 microns

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	4.2	239	88
Tailings	95.8	1.42	12
Feed	100.0	11.3	100.0

Table 18

Results of Tabling

Flotation Tailings 5, -125 microns

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	1.2	202	79
Tailings	98.8	0.66	21
Feed	100.0	3.1	100.0

The results indicate that the remaining gold was recovered quite effectively and the grades of the concentrates are the proof that the substantial proportion of the gold is free gold.

D000870940

1.6 Combination of flotation and tabling of flotation tailings

The following illustration (Table 19) of the effectiveness of the combined treatment is based on the results of the Flotation 5 and the subsequent tabling of the flotation tailings.

Table 19

Product	Weight %	Au Assay g/t	Au Distribution %
Flot. Concentrate	43.01	32.2	87.30
Table Conc. +125 u	0.13	239	1.91
Table Con. -125 u	0.65	202	8.31
Combined Conc.	43.79	35.3	97.52
Tailings +125 u	2.91	1.42	0.26
Tailings -125 u	53.30	0.66	2.22
Combined Tailings	56.21	0.70	2.48
Orig. Feed	100.00	15.9	100.00

The overall recovery of gold was 97.5% into the concentrates and the grade of the combined concentrates is 35.3 g/t Au. However, in a production plant, the losses in fine particle size ranges have to be considered and as a result the 97.5% gold recovery is probably the upper limit.

1.7 Combination of flotation and cyanidation of flotation tailings

The tabling of the flotation tailings proved that the gold contained in the flotation tailings is prevalently the free gold which did not respond to the flotation treatment. Theoretically the free gold should be amenable to cyanidation, therefore the cyanidation of flotation tailings was tested.

A sample of the flotation 4 tailings was cyanided at the following conditions; solids:liquid ratio of 1:3; 0.2% NaCN concentration in the solution; cyanidation time - 24 hours; pH 10-11 maintained by lime; using a beaker, stirrer, blower arrangement. The slurry after the cyanidation was filtered, the solids washed, dried and analyzed for gold content. The following results were obtained:

Feed to cyanidation (Flotation Tailings 4)	- 2.55 g/t Au
Cyanidation tailings	- 0.06 g/t Au
Gold recovery (from the Flotation Tailings 4)	97.6%

The results of the combined treatment - flotation followed by cyanidation of the flotation tailings, are illustrated in Table 20.

D000870950

Table 20

Product	Weight %	Au Assay g/t	Au Distribution %
Flot. Concentrate	43.1	32.8	90.70
Flot. Tailings (Cyanid. Feed)	56.9	2.55	9.30
Cyanid. Solution	-	-	9.08
Cyanid. Tailings	56.9	0.06	0.22
Feed (to Flot.)	100.00	15.6	100.00

The total recovery of gold was 99.78%. The results confirm that by using the flotation treatment followed by cyanidation of the flotation tailings a very good recovery can be achieved.

1.8 Upgrading of the Flotation Concentrate

The rougher flotation sulphide concentrates have relatively low gold contents. Although an introduction of a cleaning circuit should improve the gold grade in the cleaner concentrates, the improvement will be only marginal and it is possible that marketing of these concentrates could encounter some problems. Consequently the possibility of upgrading the concentrates and the eventuality of extraction of the gold from the concentrates in metallic form were investigated.

The samples of the flotation concentrates were roasted at 650°C and the upgrading of the roasted concentrates due to the burning of the sulphur content was evaluated. The results are presented in the Table 21.

D000870960

Table 21

Roasting Time Minutes	Weight % after Roasting	Grade of Concentrate Au g/t after roasting
0	100	34.6
20	90	38.2
40	86	43.0
80	73.4	44.4
120	71.0	48.5

The 120 minutes roasting of the flotation concentrate increased the gold grade by 40%, from 34.6 g/t Au to 48.5 g/t Au, while the original weight of the concentrate was reduced by 29%.

The roasted concentrates were subjected to cyanidation in order to assess the effect of roasting on cyanidation and the possibility of gold extraction from the concentrate.

The samples of the concentrates were cyanided maintaining the conditions of the cyanidations as follows: the solid:liquid ratio 1:10; 0.1% concentration of NaCN in the solution; pH 10-10.5 maintained by lime; cyanidation time 17.5 hours; beaker, stirrer, blower arrangement was used. The results of the tests are presented in Table 22.

Table 22

Roasting Time Minutes	Feed Grade Au g/t	Tailings Grade Au g/t	Au Recovery %
0	34.6	33.3	3.8
20	38.2	28.6	25.2
40	43.0	35.7	17.0
80	44.4	3.25	92.7
120	48.5	3.05	93.7

The direct cyanidation of the flotation concentrates yielded very poor recovery of gold. It is assumed that the flotation reagents adsorbed on the surfaces of the sulphide particles inhibit cyanidation. An extensive roasting appears to be a prerequisite to a reasonable recovery of gold by cyanidation from the flotation concentrates.

The pregnant liquor from the cyanidation of the flotation concentrate roasted for 120 minutes was analyzed for the contents of Au, Cu and CN (as KCN) with the following results: Au 2.13 mg/L (milligrammes per litre), Cu 187.5 mg/L, KCN 25 mg/L. Although the recovery of gold into the pregnant solution was quite satisfactory, the remaining low concentration of cyanide and mainly the high concentration of copper in the pregnant solution indicate a high degree of copper leaching which could cause considerable production difficulties, e.g. high consumption of cyanide, lime, activated carbon or zinc dust and probably low grade bullion. This alternative, if considered practicable, will require much more thorough research and the outlined findings should be considered as a preliminary hypothesis.

outlined findings should be considered as a preliminary hypothesis.

If we consider the best flotation recovery 90.7% Au (Table 12) into the flotation concentrate which is then subjected to roasting and cyanidation at the recovery of gold of 93.7% (Table 22), then the overall gold recovery will be 85%.

If we consider the flotation followed by cyanidation of the flotation tailings and then roasting of the flotation concentrate followed by (separate) cyanidation of the roasted flotation concentrate, the overall gold recovery could reach about 94%.

1.9 Discussion

The gold contained in the underground ore is finely disseminated within the sulphide minerals which are intimately intergrown with the gangue minerals. In order to recover the gold from the ore (at acceptable recovery and grade) a fine grinding is required which would liberate the gold containing sulphide minerals from the gangue minerals and allow subsequent separation of the gold containing minerals from gangue. However, during the comminution process a part of the fine metallic gold in the sulphide minerals matrix becomes also liberated; this creates a mineral processing problem due to the different properties of the target minerals (metallic gold, sulphide minerals) and consequently the different responses to the mineral processing methods considered suitable for this type of ore, e.g.:

1. The gravity separation is effectively separating the liberated sulphide minerals of coarser particle sizes and the fine free metallic gold particles from the gangue minerals; but fine particles of the sulphide minerals and the very fine free gold particles are lost to the tailings, because the effectiveness of gravity separation processes in these particle size and specific gravity ranges is low or nil.

D000870970

2. The cyanidation is effective where the gold is exposed to the action of cyanide solutions (e.g. fine free gold, gold on the surface or close to the surface of sulphide minerals' particles), but the gold locked within the sulphide minerals matrix does not dissolve and is lost in the cyanidation tailings.

   The results of the cyanidation can be improved by roasting and by ultrafine grinding (e.g. 100% - 20 u) of the ore prior to cyanidation; however both processes are costly (installation and operations) affecting substantially production costs and in the case of a small volume production plant (low tonnage throughput) the cost can become prohibitive.

   About 70% gold recovery can be expected when the ground ore is cyanided using a high cyanide concentration in the liquor; 97.6% gold recovery was achieved when the ore was ground to 100% -75 microns, then roasted and cyanided; about 90% gold recovery is anticipated when the ore is ground very fine (e.g. 100% -20 microns) and then cyanided. There are also indications that relatively high cyanide consumptions can be expected.

3. The flotation is reasonably effective recovering finely ground sulphide minerals, but very fine particles of the sulphide minerals and the free gold particles respond poorly to the flotation process causing the bulk of the gold losses in the flotation tailings. The gold values of the flotation concentrate are relatively low at about 32-36 g/t.

   These mineral processing methods were considered the most promising methods for this type of ore. However, as the abreviated description of the positive and the negative aspects of the testing of each of the methods reveals, none

of the methods is suitable on its own as the sole processing method for the ore if a high standard gold recovery is required. When the recovery of gold is considered as one of the most important measures for evaluation of the particular processing methods, the best results were achieved by flotation. The flotation separates >95% of sulphide minerals (Table 8) with associated gold from gangue and the recovery of gold in excess of 90% can be achieved. The gold lost in the flotation tailings is mostly fine free gold which can be most effectively recovered by cyanidation of the tailings at the gold recovery of 97.6%. The combined gold recovery of the flotation followed by cyanidation of the flotation tailings was 99.7%. The flotation accounted for 90.7% and the cyanidation for 9.08% of the gold recovery.

The gravity separation of the fine free gold from the flotation tailings was less efficient, recovering 80.4% of the gold from the flotation tailings; the combined gold recovery of the flotation followed by gravity separation of the flotation tailings was 97.5%; the flotation accounted for 87.3% and the gravity separation for 10.22 of the gold recovery.

926

Part 2

Preliminary study of gold extraction from open pit oxidized ore

D000870980

2.1 Basic characteristics of the ore

2.1.a Sample collection and composition

Sixteen samples of oxidized ore from the Ironclad open pit were supplied by the Lachlan-Metallic Minerals Section of the Geological Survey of NSW. The original samples were marked: G85/18-23 and G85/26-35. On the basis of the location, where from the samples were collected, the samples were allocated into three groups of samples.

Western Part, Open Pit:	G85/018, 019, 020, 021
Middle Part, Open Pit:	G85/022, 023, 024, 033, 034, 035
Eastern Part, Open Pit:	G85/027, 028, 029, 030, 031, 032

The samples were composited into the group by quartering out the same weight sample from each of the samples. The composited and homogenized group samples were then composited into a representative "Open Pit Ore" sample using the following formula.

Western Part O/P	composite sample	20% by weight
Middle Part O/P	" "	40% by weight
Eastern Part O/P	" "	40% by weight
Open Pit Ore	composite sample	100% by weight

The resulting "Open Pit Ore" sample was again homogenized and used for all investigations described in this report. The compositing of the samples was agreed upon in consultations with the personnel of the Lachlan-Metallic Minerals Section of Geological Survey of NSW.

2.1.b Sample preparation and analyses

All the individual samples were crushed in two stages to -4 mm, using jaw crushers. The samples were then quartered to

approximately 1,500 grammes. These samples were quartered further to approximately 200 grammes, then pulverized in the Braun disc pulverizer and further pulverized in the Siebtechnique mill pulverizer for chemical analyses.

The chemical analyses are presented in the Table 1.

Table 1

Chemical Analyses of Samples of ore from Open Pit

Sample No.	Chemical Analyses ppm						
	Au	Ag	Cu	Pb	Zn	As	Bi
G85/018	0.64	0.6	410	30	40	120	<20
G85/019	2.55	1.3	300	15	46	80	<20
G85/020	0.28	<0.2	410	8	28	60	<20
G85/021	2.15	2.0	550	60	39	<20	<20
G85/022	0.62	<0.2	560	18	58	<20	<20
G85/023	1.80	<0.2	470	8	100	40	<20
G85/026	16.5	2.3	370	10	95	100	<20
G85/028	0.33	0.4	260	8	55	45	<20
G85/029	0.15	<0.2	360	8	90	25	<20
G85/030	0.30	<0.2	400	8	180	<20	<20
G85/031	2.18	0.7	510	50	520	100	<20
G85/032	1.18	1.2	270	30	30	100	<20
G85/033	12.1	1.8	1270	34	100	85	<20
G85/034	0.41	<0.2	630	<5	85	<20	<20
G85/035	0.07	2.3	1090	8	80	<20	<20

D000870990

From the remaining samples, 1,200 g of each sample was used for compositing the three group samples. Each of the composited group samples were homogenized in a tumble mill for 2 hours. Parts of the composited and homogenized group samples were quartered out, prepared for chemical analyses and analyzed.

Table 2

Chemical Analyses of the Group Samples

Sample	Chemical Analyses						
	Au g/t	Ag g/t	Cu %	Pb %	Zn %	As %	Sb %
Western O/P	1.05	0.1	0.05	0.004	0.002	0.017	<0.002
Middle O/P	4.0	2.9	0.08	0.004	0.008	0.016	<0.002
Eastern O/P	3.50	0.7	0.04	0.004	0.008	0.016	<0.002

From the remainder of the composited group samples the following quantities were used to prepare the "Open Pit Ore" sample:

Western O/P	3 kg
Middle O/P	6 kg
Eastern O/P	6 kg
	15 kg

The "Open Pit Ore" sample was homogenized (tumble mill, 2 hours) and the homogenized sample crushed using a rolls crusher with the

1.5 mm gap between the rolls. This sample was then used for the metallurgical testing.

The ore was analyzed by XRD of powdered specimens for major mineralogical constituents and also ore specimens were microscopically examined. The main minerals determined in the ore were: Quartz, Feldspar, Hematite and small quantities of iron sulphides were also detected. The gold is present mainly as free gold in gangue minerals and the surfaces of gold particles are heavily tarnished by iron oxides. The size of gold particles ranges from approximately 20 microns long (resembling wire) particles to <2 micron diameter flat particles and possibly also sub-micron size particles which could not be positively identified.

A substantial sample of the ore was sieved to the particle size fractions as presented in Table 3, and each of the fractions was analyzed for gold content.

Table 3

Size fraction microns	Au Analyses g/t
-600 +425	3.0
-425 +300	3.95
-300 +150	7.82
-150 +75	4.55
-75	5.15

The gold is relatively evenly spread throughout the particle size fractions, though there is a modest gold content increase in favour of the finer particle size fractions.

2.2 Heavy Media Separation

The remainder of the particle size fraction samples were de-slimed, air dried and subjected to a heavy media separation at S.G. 2.94 using tetrabromoethane as the heavy liquid.

The coarse fractions (+300 microns) were separated in beakers, the separations of the fine fractions (-300 microns) were conducted in a separating funnel. The fractions were separated in small portions in order to maintain the solids:liquid ratio of approximately 1:10. The products of the HMS were vacuum filtered, several times washed on the filters by ethyl-alcohol, air dried, weighed, samples for analyses prepared, assayed for gold content, analyzed by XRD, microscopically examined and part of each of the "Sink" fractions were separated further by HMS at S.G. 4.65 using Clerici's solution as the heavy liquid.

The separations in the Clerici's solution were conducted at 50°C, in a separating funnel enveloped in water jacket and connected to a constant temperature equipment. The HMS products were vacuum filtered at >50°C, thoroughly washed with hot water, dried, weighed, samples prepared and analyzed as described above.

The -75 micron fraction was not separated by HMS. The results of the Heavy Media Separations are presented in the Tables 4, 5, 6 and 7.

D000871000

Table 4

HMS -600 +425 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	8.5	31.4	88.7
Float S.G. 2.9	91.5	0.4	11.3
Feed to S.G. 2.9	100.0	3.0	100.0

--- . ---

Sink S.G. 4.6	4.9	341	53.1
Float S.G. 4.6	95.1	15.5	46.9
Feed to S.G. 4.6	100.0	31.4	100.0

--- . ---

Float S.G. 2.9	91.5	0.4	11.3
Float S.G. 4.6	8.1	15.5	41.6
Sink S.G. 4.6	0.4	341	47.1
Feed to HMS	100.0	3.0	100.0

D000871010

G 1986 026

Table 5

HMS -425 +300 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	9.9	35.6	89.3
Float S.G. 2.9	90.1	0.5	10.7
Feed to S.G. 2.9	100.0	3.9	100.0
---	---	---	---
Sink S.G. 4.6	4.6	460	58.9
Float S.G. 4.6	95.4	15.3	41.1
Feed to S.G. 4.6	100.0	35.6	100.0
---	---	---	---
Float S.G. 2.9	90.1	0.5	10.9
Float S.G. 4.6	9.5	15.3	37.1
Sink S.G. 4.6	0.4	460	52.0
Feed to HMS	100.0	3.9	100.0

Table 6

HMS -300 +150 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	9.5	56.9	69.2
Float S.G. 2.9	90.5	2.6	30.8
Feed to S.G. 2.9	100.0	7.8	100.0

--- . ---

Sink S.G. 4.6	20.1	256	90.2
Float S.G. 4.6	79.9	7	9.8
Feed to S.G. 4.6	100.0	56.9	100.0

--- . ---

Float S.G. 2.9	90.5	2.6	30.8
Float S.G. 4.6	7.6	7.0	6.8
Sink S.G. 4.6	1.9	256	62.4
Feed to HMS	100.0	7.8	100.0

G. 1900626

Table 7

HMS -150 +75 micron fraction

Product	Weight %	Au Assay g/t	Au Distribution %
Sink S.G. 2.9	7.5	44.0	71.1
Float S.G. 2.9	92.5	1.4	28.9
Feed to S.G. 2.9	100.0	4.6	100.0
---	---	---	---
Sink S.G. 4.6	28.0	143	90.8
Float S.G. 4.6	72.0	5.6	9.2
Feed to S.G. 4.6	100.0	44.0	100.0
---	---	---	---
Float S.G. 2.9	92.5	1.4	29.2
Float S.G. 4.6	5.4	5.6	6.6
Sink S.G. 4.6	2.1	143	64.2
Feed to HMS	100.0	4.6	100.0

The recovery of gold increases along with the decreasing particle size, while the grade of the final concentrate (Sink S.G. 4.6) decreases. The coarser the fraction, the higher the losses of gold in the midlings (Float S.G. 4.6). The overall recovery of gold is in the 47-64% range, which is an indication that gravity separation may be used, most probably as a supplementary method.

2.3 Gravity Separation - Tabling

The sample for the initial test of tabling was pulverized and sieved at 106 microns. The +106 micron fraction was rejected, the -106 micron fraction was used as the feed to the flat deck of the Mozley Laboratory Mineral Separator. The tabling was conducted in two stages; in the first stage three products were produced: concentrate, midlings, tailings. The tailings were the final product. The midlings were used as the feed for the second stage of the tabling and two products were produced: the concentrate from the 2nd stage was added to the concentrate from the 1st stage and the tailings from the 2nd stage were considered as the midlings. All the final products were filtered, dried, weighed, samples for analyses prepared acid then analysed. The results of the tabling are presented in the Table 8.

D000871020

Table 8

Tabling of ore, - 106 microns

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	2.2	164	59.7
Midlings	9.2	15.8	23.7
Tailings	88.6	1.15	16.6
Feed	100.0	6.1	100.0

The results of the tabling were encouraging, though the prospect of the fine grinding of the ore to 100% - 106 microns was not considered as an attractive proposition. Consequently further tabling tests were conducted using as the feed for the tabling a substantial sample of ore crushed to 100% -1 millimetre. The sample was again sieved at 106 microns to the -1 mm +106 micron fraction and the -106 micron fraction. The +106 micron fraction represented 84% by weight of the original sample and was used as the feed for "V" deck of the Mozley Separator. The -106 micron fraction represented 16% of the sample and was tabled on the flat deck of the separator. The original split line between the concentrates and the midlings was moved further towards the midlings.

The results of the tabling are presented in the Tables 9, 10 and 11.

D000871030

Table 9

Results of Tabling

Ore, -106 microns

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	4.3	110	77.1
Midlings	9.6	4.5	6.9
Tailings	86.1	1.15	16.0
Feed	100.0	6.2	100.0

Table 10

Results of Tabling

Ore, -1 mm +106 microns

Product	Weight %	Au Assay g/t	Au Distribution %
Concentrate	1.3	289	77.9
Midlings	4.5	7.3	6.7
Tailings	94.2	0.80	15.4
Feed	100.0	4.9	100.0

Table 11

Combined Results of Tabling

Ore, -1 mm +106 micron and -106 micron fractions

Product	Weight %	Au Assay g/t	Au Distribution %
Comb. Concentrate	1.8	220	77.8
Comb. Midlings	5.3	6.5	6.8
Comb. Tailings	92.9	0.85	15.1
Comb. Feed	100.0	5.1	100.0

The results demonstrate that the gravity separation can be used to produce good grade concentrates at a reasonable recovery of gold. The gold recovery to midlings was reduced by shifting the split line between concentrates and midlings further towards midlings (compare Table 8), though the losses of gold into the tailings remained relatively high. The combined tailings and midlings contained gold content of 1.16 g/t Au in the 98.2% mass of the original feed accounting for 22.2% of gold. Although the loss of gold could be marginally reduced by further treatment of midlings, the loss of gold at approximately 20% can be expected. Since the lost gold is mainly represented by ultrafine free gold (does not respond to gravity separation) and fine gold disseminated in gangue, cyanidation was considered as a prospective method to yield higher recovery of gold.

2.4 Cyanidation

2.4.a Direct Cyanidation of the Ore

The initial cyanidation test was conducted using as the feed a -75 micron sample of the ore, assaying 5.15 g/t Au. The solids:liquid ratio was 1:2.5; 0.2% NaCN concentration of cyanide solution. The cyanidation was conducted in a beaker, the immersed mechanical stirrer maintained the solids in suspension and through the vortex created by the slurry in a circular motion, aeration was achieved; pH was maintained by lime at 9.5-10.0; cyanidation time was 24 hours.

The slurry after the cyanidation was vacuum filtered, the filter cake washed by water on the filter, dried and assayed for gold content. The cyanidation tailings contained 0.44 g/t Au and the corresponding gold recovery was calculated at 91.5%.

It was assumed that the results can be improved if the optimum cyanidation conditions are determined and applied. For the next series of cyanidation tests a substantial sample of O/P ore was pulverized to 100% -150 microns and using a rotary splitter, divided into six samples of almost equal weight (and presumably of gold content). One of the samples was assayed for gold content, the remaining five samples were cyanided at identical conditions, except cyanidation time. The purpose of the testing was to determine the minimum cyanidation time at which an acceptable gold recovery can be achieved.

The following cyanidation conditions were applied:

solids:liquid ratio - 1:5
cyanide concentration 0.2% NaCN in solution
pH 10-10.5 (by lime)
cyanidations conducted using the beaker, stirrer, blower arrangements
feed assay: 5.25 g/t Au

D000871040

cyanidation times: 24 hours; 17 hours; 8 hours; 6 hours; 3 hours

All the cyanidation tailings were vacuum filtered, washed by water on the filters, dried and assayed for gold contents. The results are presented in the Table 12.

Table 12

Determination of cyanidation time

Cyanidation Time Hours	Au Assay g/t in Cyanidation Tailings	Au Recovery %
24	0.44	91.6
17	0.47	91.0
8	0.44	91.6
6	0.53	89.9
3	1.57	70.1

The feed assay: 5.25 g/t Au

The results indicate that 8 hours cyanidation time is sufficient to secure a reasonable recovery of gold and extension of cyanidation time beyond 8 hours does not improve gold recovery.

The next series of cyanidation test was conducted with the aim of determining the effect of cyanide concentration in the cyanide solution on gold recovery. The ore was pulverized to 100% -106

D000871050

microns; a feed sample for assaying was quartered out and the rest of the ore sample was divided into 12 equivalent samples using a rotary splitter. The samples were then cyanided in duplicates, maintaining identical cyanidation conditions, except cyanide concentration in the cyanide solutions.

The following cyanidation conditions were applied:

Solids:liquid ratio - 1:5
pH 10-10.5 (by lime)
cyanidations conducted using beaker, stirrer, blower arrangements
feed assay: 4.96 g/t Au
cyanidation time: 7.5 hours
cyanide concentrations: 0.2%, 0.1%, 0.05%, 0.02%, 0.01%

All the cyanidation tailings were vacuum filtered, washed by water on the filters, dried and assayed for gold contents. The assays of each pair of the samples cyanided at the same concentration of the cyanide solution were averaged. The results are presented in Table 13.

D000871060

Table 13

Effect of concentration of cyanide in solution

Cyanidation Concentration % NaCN	Au Assay g/t in Cyanidation Tailings	Au Recovery %
0.2	0.22	95.6
0.1	0.16	96.8
0.05	0.24	95.2
0.02	0.15	97.0
0.01	0.55	88.9

The feed assay: 4.96 g/t Au

The 0.02% concentration of NaCN is adequate for achieving of a reasonable gold recovery, also indicating an acceptable consumption of cyanide.

2.4.b Cyanidation of gravity separation tailings

Although by the direct cyanidation of "run of mine" ore, good gold recoveries can be achieved, providing the ore is finely ground, the necessity for the fine grinding could possibly be avoided if the coarse midling particles containing high concentration of gold and the coarse free gold are eliminated by gravity separation prior to cyanidation. This assumption was tested using as the feed to cyanidation, the "float" fractions at S.G. 2.94 from the HMS.

D000871070

The -425 +300 micron, -300 +150 micron, -150 +75 micron fractions of -2.94 S.G. from the HMS were subjected to cyanidation under the conditions as follows:

solids:liquid ratio - 1:5
pH 10-11 (by lime)
cyanidation conducted using beaker, stirrer, blower arrangements
cyanide concentration: 0.3% NaCN
cyanidation time: 24 hours

The cyanidation tailings were vacuum filtered, washed by water on the filters, dried, pulverized and assayed for gold contents. The results are presented in Table 14.

Table 14

Cyanidation of HMS (2.9 S.G.) "float" fractions

Float fraction microns	Au Assay g/t prior cyanidation	Au Assay g/t after cyanidation	Au Recovery %
-425 +300	0.47	0.12	74.5
-300 +150	2.65	0.16	94.0
-150 +75	1.42	0.14	91.4

The tests confirmed that the tailings can be successfully cyanided without prior fine grinding. On this basis a further cyanidation test was carried out using as the feed the tailings from the coarse tabling (-1 mm +106 microns). The tailings were

GS 1986 028

pulverized to 100% -300 microns and subjected to cyanidation at the following conditions:

solids:liquid ratio - 1:3
pH 10-11 (by lime)
cyanidation conducted using beaker, stirrer, blower arrangement
cyanide concentration: 0.02% NaCN
cyanidation time: 8 hours
feed assay: 0.80 g/t Au
cyanidation tailings assay: 0.23 g/t Au
gold recovery by cyanidation: 71.3%

If these results are applied, then the combined recovery of gold (tabling followed by cyanidation) would be 95.6%. However it is feasible to improve the cyanidation recovery (see Table 14) by applying more suitable cyanidation conditions and the overall recovery could be increased to 97-98% of gold without fine grinding.

2.5 Discussion

The gold contained in the open pit oxidized ore is in principle free gold finely disseminated in the gangue minerals, prevalently in hematite, though presence of some iron sulphide minerals in the concentrates was also detected. Although relatively coarse particles of free gold were found, the substantial majority of gold in the ore is represented by fine and very fine gold particles.

The coarse free gold particles and gold rich midlings can be readily separated by gravity separation at good gold grades of concentrates and modest gold recoveries. However, the fine and very fine gold especially when locked within gangue minerals, responds poorly to gravity separation and a considerable quantity of the gold reports to the tailings.

D000871080

GS 19

The gold contained in the open pit ore is amenable to cyanidation and it appears that the finer the ore is ground, the better the gold recovery by cyanidation. This could lead to a conclusion that a fine grinding is a prerequisite to an acceptable recovery of gold by cyanidation process. However, there are some other aspects which would indicate, that fine grinding reduces the particle sizes of the "coarse" free gold which is then amenable to an accelerated dissolution in a cyanide liquor. In other words, the coarse free gold requires a long time to dissolve in a cyanide solution and when the adequate time is not available, the undissolved gold is reporting to the cyanidation tailings, where it is lost.

Alternatively, if the "coarse" gold is removed from the slurry prior to cyanidation, these losses of gold can be eliminated without fine grinding of the ore, using a relatively short cyanidation time and low concentration cyanide solutions (Table 14).

This would suggest a processing along the similar pattern as follows: a relatively coarse grinding (e.g. 100% -300 microns) followed by a simple gravity separation (e.g. Spiral or Reichert Cone) aimed only at recovering of coarse gold rich midlings and mainly free ("coarse") gold into a small quantity of gold rich concentrate. The remainder of the slurry, containing prevalently the fine and very fine gold, is subjected to a relatively short cyanidation (probably 7-8 hours). The cyanidation should be able to recover a substantial proportion of the remaining free fine gold and the gold still in association with gangue minerals. The combination of the methods should result in a satisfactory gold recovery exceeding 96%, from the open pit oxidized ore.

GC 1986 026

Part 3

Conclusions

D000871090

3.1 Sulphide - Underground Ore

(a) The gold is finely disseminated in the sulphide minerals; the sulphide minerals are intimately intergrown with gangue minerals.

(b) Using a flotation process for the separation of sulphide minerals from the ore ground to e.g. 100% -106 microns, gold recoveries in excess of 90% can be achieved; a concentrate grade of approximately 35 g/t Au can be expected.

(c) Using gravity separation of finely ground ore (e.g. 100% -106 micron) as the processing method, gold recoveries of up to 85% may be achieved; a concentrate grade of approximately 35-40 g/t Au can be expected.

(d) Direct cyanidation of the ore would result only in approximately 70% gold recovery.

Roasting of finely ground ore followed by cyanidation of the roasted ore would increase the efficiency of cyanidation; gold recoveries >97% can be achieved. However, both these processes (fine grinding and roasting) would be costly to install and operate.

(e) The gold lost in the flotation tailings is mostly "free" gold. This gold can be recovered by cyanidation of the flotation tailings. In the Laboratory trials, flotation followed by cyanidation of flotation tailings, the combined recovery of gold was 99.7%; the gold in the flotation concentrate being approximately 35 g/t Au.

(f) A significant part of the gold lost in the flotation tailings can also be recovered by gravity separations suitable for fine particle size slurry (e.g. Tables, Reichert Cone). In the Laboratory trials, flotation

followed by gravity separation of flotation tailings, the combined recovery of gold was 97.5%; the gold content in the combined concentrate can be expected at approximately 38 g/t Au.

(g) The sulphide concentrate (flotation, gravity separation) grades can be increased from 35-40 g/t Au to >50 g/t Au by roasting (de-sulphurizing) of the concentrates.

(h) The roasted concentrates can be cyanided with reasonable recoveries, though high reagent (cyanide, lime) consumptions and high contents of other heavy metals in pregnant liquors have to be expected.

3.2 Oxide - Open Pit Ore

(a) The gold in the ore is finely disseminated in gangue minerals (mainly in hematite).

(b) The gold is amenable to direct cyanidation. Recoveries of gold in excess of 97% can be achieved, when cyanidation is applied after fine grinding (100% -106 microns).

(c) Using gravity separation as the processing method, gold recoveries of approximately 80% can be achieved; the concentrate grades in excess of 200 g/t Au can be expected.

(d) When coarsely ground ore (e.g. 100% -300 microns) is subjected to a simple gravity separation, aimed at separating only coarse gold-rich midlings and free gold into a high grade gold concentrate (e.g. >220 g/t Au) without particular attention to gold recoveries, and the tailings of this separation then cyanided, combined gold recoveries of >96% can be expected.

3.3 Combined (Sulphide, Oxide Ore) Processing

Although there can be numerous variations and combinations of the processing methods which were tested as alternative treatments of the Ironclad Mine ores, the combination which yielded the best gold recovery consisted of flotation followed by cyanidation of the flotation tailings for Ironclad underground (sulphide) ore; and gravity separation of high grade gold concentrate followed by cyanidation of the gravity separation tailings for Ironclad open pit (oxide) ore.

An outline of a possible flowsheet is presented in the Figure 1.

According to the flowsheet both ores have to be crushed separately and stored in separate fine ore bins. Grinding facilities also would have to be separate (e.g. grinding U/G ore to 100% -106 micron in a ball mill, grinding O/P ore to 100% -300 microns in a rod mill) both mills operating in closed circuits with classification. The grinding is followed by the flotation of the U/G ore and the gravity separation of the O/P ore. The tailings from the flotation and the gravity separation continue for cyanidation. Cyanidation of the flotation tailings and the gravity separation tailings could be conducted in a joint cyanidation plant.

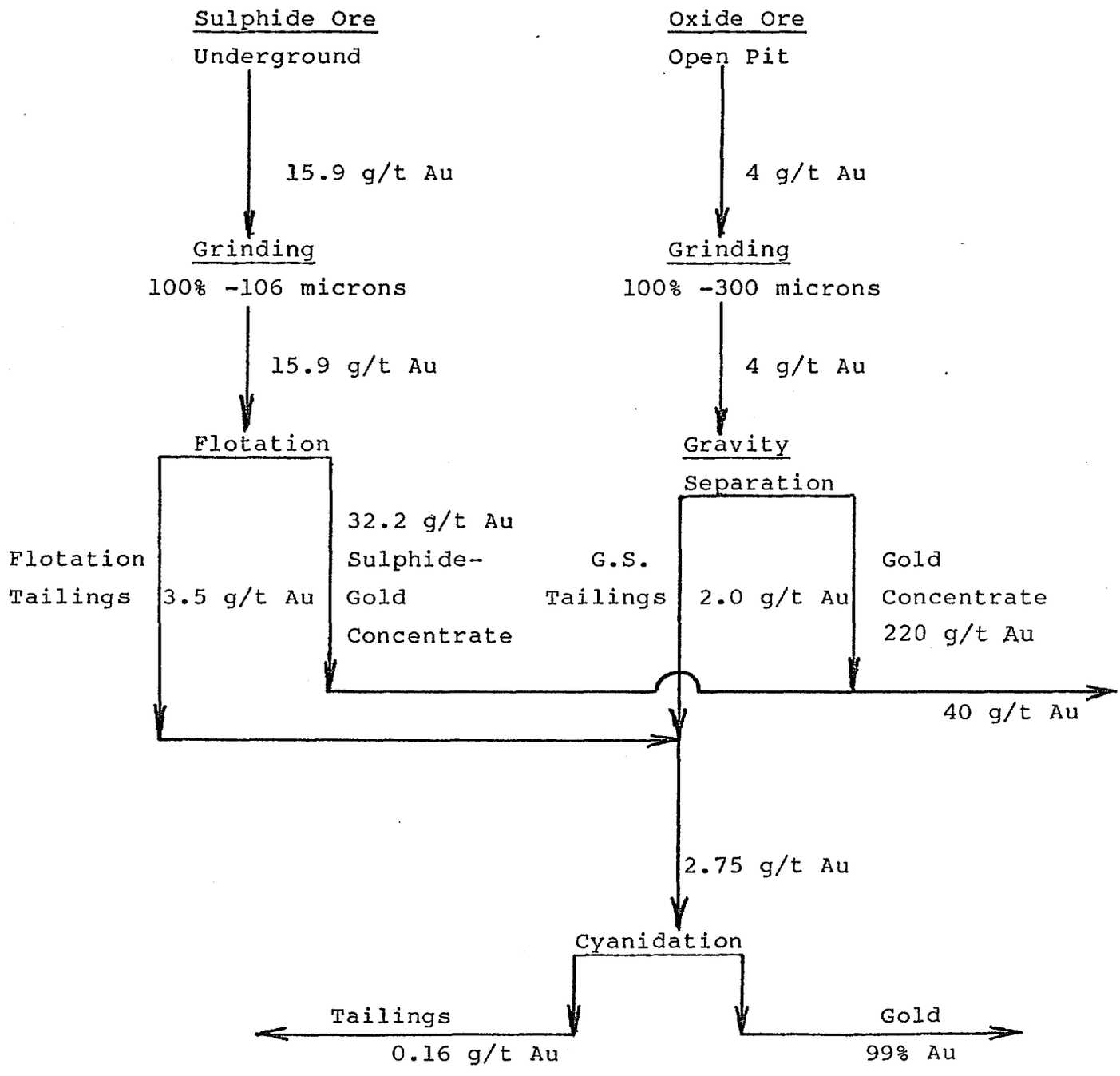
The flotation concentrate and the gravity separation concentrate can be combined and handled as one product. The inclusion of the gravity separation concentrate would upgrade the flotation concentrate e.g. from 32.8 g/t Au to 40.3 g/t Au, when equal quantities of O/P and U/G ore are processed.

However, regardless of the above technical findings, economic considerations will be decisive in the selection of treatment methods and the direction of further investigations.

Figure 1

Combined treatment of underground and open pit ores

from Ironclad gold mine



D000871100