DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

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TREATMENT OF GOLD TAILINGS - WAUKARINGA GOLD FIELD A REVIEW OF OPERATIONS AND METALLURGICAL INVESTIGATIONS, 1983-1985

GEOLOGICAL SURVEY

by

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TREATMENT OF GOLD TAILINGS - WAUKARINGA GOLD FIELD A REVIEW OF OPERATIONS AND METALLURGICAL INVESTIGATIONS 1983-1985

ABSTRACT

In the late nineteenth century the two largest mines on the Waukaringa goldfield, Alma and Victoria and Alma Extended, reportedly treated 60 000 and 2 834 tonnes of gold ore respectively. Tailings remaining on site are indicated to contain a minimum 37 710 tonnes grading 2.71 g/t gold and 1 080 tonnes grading 1.94 g/t gold respectively. Some of these tailings were retreated by cyanidation in 1889.

Attempts in 1983-1984 to recover gold from these tailings using a simple gravity concentration plant failed due mainly to the fine particles of gold being contained within goethite and silicates and therefore not conducive for amalgamation.

Cyanidation leach tests in 1983 had demonstrated that between 86% and 92% of the gold could be extracted with high cyanide and lime consumption.

A joint study by SADME and AMDEL in 1983-1985 which involved sampling of the existing plant, magnetic and heavy liquid separation, mercury solubility tests and cyanide leaching has proved that grinding and cyanidation are required to extract most of the gold.

INTRODUCTION

In August 1983, an Agreement was signed between Waukajax Pty Ltd (Leaseholder) and a Western Australian group, Connelly Investments Pty Ltd, for the latter to establish a processing plant on site to recover gold contained in old tailings dumps on the leases.

On completion of tailings treatment and the payment of an agreed sum to Waukajax, the leases were to be transferred to Connelly Investments for future possible underground development.

In 1983, the South Australian Department of Mines and Energy (SADME) carried out an appraisal of these gold tailings (Morris, 1983) and concluded that an indicated minimum of 37 710 tonnes of tailings containing 2.71 g/t gold remained in the main dump and that cyanidation could extract between 86% and 92% of the gold but reagent consumption was high, and gold extraction rates for the coarser tailings were much lower.

From 1984 to 1985 the authors assisted at various times by I.J. Townsend, W.P. Fradd, and B.J. Morris undertook on site sampling.

LOCATION AND TENURE

Waukaringa Goldfield is located 35 km north of Yunta extending eastwards for 1.5 km from the Yunta - Arkaroola unsealed road. Yunta is about 325 km northeast of Adelaide on the Barrier Highway and standard gauge railway to Broken Hill (Fig. 1). Waukaringa township ruins, the most prominent of which is the hotel, are beside the Yunta-Arkaroola road.

Gold tailings are located on section 1012 north out of hundreds, which is Crown Land set aside as a Goldfield Reserve, and part of the Far North Planning Area. The tailings and surrounding mine areas are covered by Mineral Leases (ML) 5098 to 5104 which were granted to TEMAN Pty Ltd on 21 February 1983 and transferred to Waukajax Pty Ltd on 11 May 1983. The bulk of the tailings lie on ML 5102 (Plate 1) with a smaller stockpile on ML 5100. Conditions of the Mineral Leases include:

- the lessee is prohibited from interference with European heritage material found within the tenement.
- liquids or solids discharged from processing not to be allowed to come into contact directly or indirectly with the water storage tank on section 101, Water Reserve.

GEOLOGICAL SETTING

Waukaringa Goldfield is located on the northern margin of the northeasterly trending Waukaringa Syncline. Gold bearing quartz veins occur within Cox Sandstone Member, feldspathic sandstone at the base of Tarcowie Siltstone and immediately overlying Tapley Hill Formation, all sediments belonging to

Umberatana Group of Adelaidean age. The veins which strike northeasterly and dip 30° south are generally conformable with bedding. Gold occurs in sulphide (pyrite, arsenopyrite)-bearing quartz veins up to 2.0 m wide.

Detailed mapping and sampling of the goldfield and accessible underground workings has been completed by SADME (Townsend, in press).

BRIEF HISTORY

Gold was first discovered in 1873 by a shepherd J. Watson. Several mining companies opened mines on the reefs, the largest being Alma and Victoria which produced 60 000 tonnes of ore yielding 1 400 kg of gold bullion. Ore was crushed through a 20 head stamp battery which operated on site between 1887 and 1894. Pyrite concentrate was roasted, then amalgamated in a Denny Pan.

Between 1890 and 1892, Alma Extended mine, approximately 1.5 km west of Alma and Victoria, operated a 20 head battery, roasting furnace and amalgamation plant. Ore was also treated in this battery between 1910 and 1911 and total recorded production is 2 834 tonnes. Brown (1908) reported assays of tailings and slimes as 45 g/t Au and 6.3 g/t Au respectively, vastly different from results reported by Morris (1983).

A detailed history of mining and ore treatment on the Waukaringa goldfield has been compiled by Fradd (in press).

PREVIOUS INVESTIGATIONS

History of gold ore treatment, the geology of the tailings and the following estimate of reserves remaining in the two main dumps associated with old mine workings at Alma and Victoria and Alma Extended are reported by Morris (1983).

Alma and Victoria
37 710 tonnes averaging 2.71 g/t gold.

Alma Extended
1 080 tonnes averaging 1.94 g/t gold.

Based on agitation cyanide leaching tests carried out on four bulk samples, recoverable gold was valued at $1 069 540, at a gold price of $12.9 per gram.

PLANT CONSTRUCTION AND OPERATION

Connelly Investments Pty Ltd commenced site preparation at Waukaringa in February 1984 and in April of the same year, installation of the processing plant began.

A tailings dam was excavated on the western side of Dump C, about 100 m southwest of Victoria shaft. During this work, additional tailings were exposed. Stockpiles of material from various dumps and locations were formed to provide a feed to the mill. Samples of these stockpiles and additional tailings were collected by R. Horn on 27 June 1984 (Table 1), at locations shown on Fig. 3 (modified from Morris, 1983).

TABLE 1

RESULTS OF ASSAYS OF TAILINGS UNCOVERED DURING SITE PREPARATION
(AMDEL Reports AC2217/84 and AC65/85)

Sample No.	Description	g/t Au
A1090/84	Residue from B. Burnie Plant - Pit immediately below shed - sandy and silty sand.	6.4
A1091/84	Tailings in eastern wall of new tailings pond.	3.2
A1092/84	Calcrete & soil base - same location as for A1091/84.	0.3
A1093/84	Western side of access track from shed - clayey material with sulphate minerals.	3.6
A1094/84	Stockpile of material taken from costean 1 - Stockpile on top of old mullock where large shed was.	4.5
A1095/84	Material from the new tailings pond now stockpiled adjacent to the old hearths north of the shed.	1.2
A1096/84	Material from bottom of the new tailings pond now stockpiled in eastern hearths north of the shed.	3.4
A1097/84	Sample of fines from mullock dump west of the shed beside access track.	17.3

A1098/84	Stockpiled mullock for screening. Sample of fines.	7.1
A1099/84	Material from costean and stockpiled below shed. Mainly Dump G material.	2.5
A1100/84	Material from floor of new tailings dam - stockpiled adjacent to Dump F.	4.1
A5200/83	Randomly picked material from southeastern side of main mullock dump below the shack.	0.22
A5201/83	Grab sample from mullock dump west of old Wilfley table shed - western side of track.	0.10

Based on these results:

- grade of tailings may be slightly higher than reported by Morris (1983)
- tonnage is likely to be higher by the addition of the tailings at the dam site.
- samples from the mullock dump on the western side of the shed suggest that screening out of fines and treatment through the plant could add marginally to the reserves.

However the plant as contructed, relied on simple gravity concentration as used to recover relatively coarse free gold from alluvial deposits. There was no provision for fine grinding to release gold from composites nor for recovery of the very fine particles of free gold that had been observed during panning.

Furthermore, there was little testing of tailings before construction, design was based on plants used in Western Australia that had successfully treated gold tailings from old mining operations despite conclusions in Morris (1983) that:

- the complicated treatment history involved amalgamation, cyanide leaching and roast/leach processes
- most gold was unlikely to be recovered by traditional gravity methods
- some form of leaching would probably be necessary.

These findings were supported in an earlier observation by Kinhill Consultants and from work by Dr. Keith Quast (S.A. Institute of Technology) in confidential reports to the leaseholders.

Nevertheless, the plant was completed and commissioned in July - August with further processing trials through September - October 1984 (Plates 2 to 5).

During this period, several major deficiencies became apparent, resulting in blockages, breakdowns and ineffective concentration of a gold product. One of the major early problems was inadequate power, such that not all plant could be operated simultaneously.

The net result of all these problems was that effective production time was reduced to a few hours per day and even then the plant was incapable of producing a suitable gold concentrate.

In addition, contractural agreements between Connelly Investments and Waukajax were not being maintained and the whole operation appeared to be in jeopardy.

Administrative changes resulted in Waukajax Pty Ltd becoming site manager and formal assistance was requested from SADME.

In view of the capital expenditure already exceeding $200 000 and the employment potential in such a remote area, a joint SADME - Australian Mineral Development Laboratories (AMDEL) investigation was approved in November 1984.

INITIAL INVESTIGATION

Strategic samples were collected throughout the operating plant to determine gold content and hence effectiveness of the concentrating process.

Processing circuit, shown diagramatically in Figure 2, consists basically of screening, agitation/mixing, jigging and tabling.

Results of initial sampling during several hours of plant operation (Table 2) indicated that gold was concentrating on the Wilfley Table with values of 39.8 g/t Au in the small amount of Concentrate 1 recovered, down to 7.4 g/t Au in the larger amount of Concentrate 3 (middlings).

TABLE 2

ASSAY RESULTS OF FEED AND PRODUCTS AT WAUKARINGA PROCESSING PLANT (AMDEL Report AC2397/84)

FEED CONSISTING OF DUMPS A & B

Samples	Description	Assay (g/t Au)
A2621/84	Head No 1. ex launder	4.9
A2620/84	Tail No 1. ex pipe	4.2
A2622/84	o/s + 19.05mm (3/4") screen	2.9
A2623/84	o/s + 6.35mm (1/4") screen	3.6
A2625/84	Head No 2. ex launder	4.2
A2624/84	Tail No 2. ex pipe	5.6
A2626/84	o/s + 19.05mm (3/4") screen	2.7
A2627/84	o/s + 6.35mm (1/4") screen	2.9
A2628/84	Concentrate 1.	39.8
A2629/84	Concentrate 2.	10.2
A2630/84	Concentrate 3. (middlings)	7.4

Concentrates consisted predominantly of black sand-sized grains of hematite/goethite. Free gold was rarely visible to the naked eye. However, its presence was revealed by further panning of these concentrates.

Gold values are high in tailings compared to head feed and gold is being lost on the oversize screens.

Based on these results and an estimated through-put of eight tonnes per hour, an approximate mass flow balance was prepared (Table 3). While the plant is achieving some degree of concentration, overall gold recovery in the concentrate is low at 28%, and gold losses to oversize and tailings total 72%.

Clearly, simple gravity concentration was not capable of effectively recovering gold and considerable modification was required.

The next step was to try to recover free gold from the concentrate as inexpensively as possible, in order to establish a cash flow to enable other more radical changes.

TABLE 3

MASS FLOW BALANCE - WAUKARINGA PLANT

8 tph at 4.5 g/t
(36 g/hr. gold)

FEED BIN

Reject o/s

DOUBLE DECK
WET SCREEN

2.5 tph at 2.5 g/t
(6.25 g/hr gold)

JIGS

Reject Tails
5 tph at 4 g/t
(20 g/hr gold)

WILFLEY TABLE

Concentrate

Conc 1 0.5 tph at 20 g/t
Conc 2 (10 g/hr. gold)
Middlings

MOVEMENT OF GOLD THROUGH PLANT

% of gold recovered in concentrate = $\frac{10}{36} \times 100 = 28\%$

% of gold lost to oversize = $\frac{6.25}{36} \times 100 = 17\%$

% of gold lost to tailings = $\frac{20}{36} \times 100 = 55\%$

Amalgamation of the total concentrate was attempted using a small amalgam barrel, in a batch operation, on site. Small charges of concentrate were placed in the barrel with mercury and revolved at low speed with varying contact times. However, this was unsuccessful due to several reasons including mercury sickening, probably caused by the presence of iron oxides and other deleterious contaminants including sulphides, e.g. arsenopyrite.

As a further check on amalgamation three samples of table contentrate and one tailings sample were slurried and passed over the amalgamating plates at Peterborough State Battery.

Results in Table 4 again suggest very low recoveries, particularly as the table concentrate probably contained only 28 to 30% of the total gold available in the original feed.

At this stage, there was no simple way to establish an early cash flow and a full review was proposed to establish an economic treatment.

TABLE 4

ASSAY RESULTS, PETERBOROUGH STATE BATTERY
(AMDEL Reports AC2754/85 and AC3212/85)

		GOLD ASSAYS			Bullion Recovered
Sample	Quantity (Kg)	Head	Tails	Strake	(gm)
Table Conc. A + B (Parcel No.1600)	100	6.0	9.25	62.4	2.0
Table Conc. Dump G. (Caustic Washed) (Parcel No.1601)	675	11.5	9.5	155.0	3.0
Plant Tails (Parcel No.1602)	700	4.1	4.3	18.9	0.1
Table Conc. Dump G. (Cyanide Washed) (Parcel No.1603)	600	12.5	7.3	32.9	1.0

AMDEL INVESTIGATION

Dr. Keith Henley (Chief, Geological Services Section, AMDEL) was briefed to investigate the grain size and distribution of gold throughout the plant.

Initial magnetic separation revealed that 20 to 36% of the total gold in the table concentrate was associated with iron oxides whereas the remainder was probably liberated.

This was confirmed by mercury soluble/insoluble tests which showed that 77% of the gold was mercury-soluble as essentially liberated native gold with the remainder locked in iron oxides (Appendix C - AMDEL Report G6193/85 Part I).

A further sample of the table concentrate was sized and the fractions analysed for mercury soluble/insoluble gold (Appendix C - AMDEL Report G6193/85, Part II). Gold is concentrated in the finest size fraction which has 63% of the gold in 22% of the sample (Table 5). This gold is mostly mercury-soluble and most likely fine liberated gold.

TABLE 5

GOLD DISTRIBUTION IN TABLE CONCENTRATE
Sample A 2628/84
(AMDEL Report G6193/85 Pt. II table 1)

Size Fraction (µm)	Dist. Wt%	Total Gold		Mercury Sol. Gold		Mercury Insol. Gold	
		Assay g/t Au	Dist.%	Assay g/t Au	Dist.% g/t Au	Assay	Dist.%
+420	19.22	10.48	4.9	2.2	1.5	8.28	11.9
-420+300	34.28	10.5	8.7	1.7	2.0	8.84	22.7
-300+210	24.58	38.5	23.0	19.8	17.5	18.7	34.5
-210	21.92	119.2	63.4	100.4	79.0	18.8	30.9
Total Sample	100.00	(41.0) g/t Au	100.0	(28) g/t Au	100.0 g/t Au	(13)	100.0

Size Fraction (µm)	% Mercury Sol. gold in size fraction	Gold Distribution in Sample	
		Mercury Sol.	Mercury Insol.
+420	21	1.0	3.9
-420+300	16	1.4	7.3
-300+210	51	11.7	11.3
-210	84	53.3	10.1
Total Sample	68	67.4	32.6

The proportions of mercury-soluble gold generally increases with finer particle size, reflecting better liberation in the finer fractions.

Gold distribution in the feed material was also examined (Table 6). About 80% of the gold reports in the plus 75 µm fractions, which constitute about 70% of the total weight of sample.

Further tests with heavy liquid and magnetic separations (Appendix C - Table 2, 2A and 3) showed that most of the gold in the plus 75 µm fractions is not liberated but bound up as composite particles with both iron oxides and silicates.

In contrast, most of the gold in the minus 75 µm fraction is liberated native gold which represents only 22% of the total gold in the feed sample.

Fine grinding of the feed material to finer than 75 µm is required to liberate the gold particles from composite grains.

Photomicrographs of gold in composites and as free grains are included as Plates 6-14.

TABLE 6

GOLD DISTRIBUTION in HEAD SAMPLE
A 2631/84, Dumps A+B
(AMDEL Report G6193/85 Pt. I, page 2)

		Gold		
Size Fraction (µm)	Wt%	Assay	Total Gold	Dist^n
	(%)	(ppm)	(%)	
+600	14.14	6.2	0.88	15.12
-600+300	29.76	8.3	2.47	42.44
-300+150	17.23	5.1	0.88	15.12
-150+ 75	8.29	4.1	0.34	5.84
- 75	30.59	4.1	1.25	21.48
	100.00		5.82	100.00

The next step was cyanide leach test on a sample of feed material (dump A), milled to 80% passing 75 µm (Appendix C). Most of the gold was extracted after 6 hours with moderate cyanide and lime consumption (Table 7).

TABLE 7

Cyanidation of Head Sample (Dump A)
(Ground to 80% - 200# (75 μm)
(AMDEL Report G6193/85 Pt. I, page 4)

	Extraction Time (Hrs)			
	2	6	24	48
Gold extraction %	39.7	74.8	84.8	86.1
Cyanide consumption Kg/t	0.6	0.9	1.7	2.7
Lime consumption Kg/t	1.9	2.1	2.0	2.3

In contrast, Morris (1983) reported that contaminants interfered with the leach process, resulting in reduced gold recoveries and extremely high lime consumptions (Table 8).

Further gold recovery tests commissioned to resolve this (Appendix D and Table 9) confirmed that:

- a gravity circuit as constructed at Waukaringa, could not achieve satisfactory gold recovery
- recovery of 75% could be obtained by cyanide leaching from feed consisting of a mixture of dump material, but only after fine grinding to 75 μm.

Reagent consumption was high, particularly of lime which was necessary to achieve the desired alkalinity for the cyanide leach. This may increase cost and requires further investigation.

Owing to the presence of sulphides, the tailings are highly acidic and an acid leach using thiourea may reduce reagent costs.

Investigations to date have confirmed that:

- fine grinding is required to free gold in composite grains.
- leaching is required to recover the gold.

TABLE 8

Agitation Cyanide Leaching Tests
(Ground to 80% - 75 μm)
(After Morris, 1983)

			Composite Samples of each Dump	
A B	C	D		
Head Analysis (g/t Au)	3.86	3.55	2.16	1.65
Calc. Head Analysis	3.4	3.0	2.4	1.6
Residue Analysis	1.73	1.57	0.18	0.11
Extraction %				
17 hrs	47	45	85	89
23 hrs	47	45	86	92
41 hrs	48	45	89	90
48 hrs	50	47	93	94
NaCN Consumption (Kg/t)				
17	0.8	0.9	2.3	2.4
23	1.0	1.1	2.9	2.8
41	1.5	1.4	4.1	3.6
48	1.6	1.5	4.4	3.8
CaO Addition (Kg/t)				
Precondition	1.5	1.0	70	12.5
17	1.7	1.2	71	13.5
23	1.8	1.4	72	14.0
41	2.0	1.4	73	14.4
48	2.5	1.8	74	14.4

TABLE 9

Agitation Cyanide Leach Tests
Composite Sample - dumps A,B,C & D
(AMDEL Report T6068)

Particle Size

	100% - 1 mm	80% - 75 µm	
Head Assay (g/t Au)		4.0	4.0
+ 150 µm		6.6	-
- 150 µm		5.0	-
Cal. head Assay		4.7	4.7
Residue Assay		2.0	1.1
+ 150 µm		5.0	-
- 150 µm		0.8	-
Gold Extraction %			
2 hrs		50.2	64.8
6	55.6	73.5	
24	59.4	72.2	
48	57.1	76.4	
Sodium Cyanide Consumption (Kg/t)			
2	2.2	2.0	
6	3.5	3.0	
24	6.1	5.0	
48	8.5	6.7	
Lime Consumption (Kg/t)			
2	7.3	9.6	
6	10.0	11.7	
24	12.0	13.2	
48	13.5	14.4	

PRELIMINARY COST STUDY

Cyanidation of the tailings milled to 80% minus 75 µm was the only process which yielded reasonable gold recoveries.

On this basis, preliminary estimates of capital and operating costs by AMDEL suggest that the venture may be marginally profitable, provided capital expenditure on plant was kept to an absolute minimum.

The single most expensive operating cost was the consumption of lime which is extremely high compared to most gold treatment plants.

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APPENDIX A

Assay Results of Initial Plant Sampling Program

(AMDEL Report AC2397/85)

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1/16/0 - AC 2397/85

11 December 1984

NATA CERTIFICATE

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A' 83/84

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ANALYSIS

g/tonne

SAMPLE MARK	GOLD Au	
A2620/84	4.2	Tail 1.
A2621/84	4.9	Head 1.
A2622/84	2.9	o/s + 3/4
A2623/84	3.6	o/s + 1/4
A2624/84	5.6	Tail 2
A2625/84	4.2	Head 2
A2626/84	2.7	o/s + 3/4
A2627/84	2.9	o/s + 1/4
A2628/84	39.8	- No. 1 Concentrate
A2629/84	10.2	- No. 2 Concentrate
A2630/84	7.4	- No. 3. Concentrate (middlings).
A2631/84		
+ 25#	6.2	
+ 52#	8.3	
+ 100#	5.1	
+ 200#	4.1	
- 200#	4.1	

METHOD: A7/1 50g charge.

APPENDIX B
Assay Results of Samples Tested at Peterborough State Battery
(AMDEL Reports AC2754/85 and AC3212/85)

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5 February 1985

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ANALYSIS - g/tonne

SAMPLE MARK	GOLD Au	
Koonamore 1	3.60	— Head
2	62.4	— Strake
3	9.25	— Tails

Method: A7/1

N.B. Sample 1 was reassayed = 6.0 g/t

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ANALYSIS - g/tonne

SAMPLE MARK	GOLD Au
Head (1)	6.00
METHOD: A7/1 50g	

— Re-assay of Koonamore 1 (previous page).

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In reply quote:

1/16/0 - AC 3212/85

19 February 1985

NATA CERTIFICATE

WAUKARINGA.

A '16/85

The Director General
S.A. Department of Mines & Energy
P.O. Box 151
EASTWOOD SA 5063

REPORT AC 3212/85

YOUR REFERENCE:	12/03/152 EX-317 .
REPORT COMPRISING:	Cover sheet Page 1
DATE RECEIVED:	6 February 1985

D. Patterson
Chief Chemist
Analytical Chemistry Division

fs

Waukaringa Concentrate.
Peterborough Battery.

ANALYSIS - g/T

		SAMPLE MARK	GOLD Au	
Parcel 1601	WAU 46	A71/85	11.5	- Head
	47	A72/85	9.5	- Tails
	WAU 48	A73/85	155	- Strake table
Parcel 1603	WAU 49	A74/85	12.5	- Head
	50	A75/85	7.3	- Tails
	WAU 51	A76/85	32.9	- Strake table
Parcel 1602	WAU 52	A77/85	4.1	- Head
	53	A78/85	4.3	- Tails
	WAU 54	A79/85	18.9	- Strake table

METHOD: A7/1 50g Charge

APPENDIX C
Gold Mineralogy of Waukaringa Mill Products
Part I - Determination of nature of gold present
Part II - Gold mineralogy and photomicrography
(AMDEL Report G6193/85 Parts I and II)

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7 February 1985

GS 1/16/0

A 6/85

The Director-General
South Australian Department of Mines
and Energy
PO Box 151
EASTWOOD SA 5063

Attention: Mr R. Horn
Mineral Resources

REPORT G 6193/85 - PART I

YOUR REFERENCE: Application 12/03/0132, Ex-282

IDENTIFICATION: A2620/84 - A2631/84

MATERIAL: Mill products

LOCALITY: Waukaringa, South Australia

WORK REQUIRED: Determine nature of gold present

Investigation and Report by: Dr Keith J. Henley

Chief - Geological Services Section: Dr Keith J. Henley

Keith Henley

for Dr William G. Spencer
Manager, Mineral and Materials Sciences Division

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GOLD MINERALOGY OF WAUKARINGA MILL PRODUCTS

1. INTRODUCTION

Various samples of mill products from a gold processing operation at Waukaringa were submitted by the SA Department of Mines and Energy for mineralogical investigation to determine the nature of the gold present. The samples investigated were as follows:

Sample	Sample No.	Au content, ppm
Table concentrate	A2628/84	39.8
Head	A2631/84	5.8

2. TABLE CONCENTRATE (A2628/84)

A polished section (PS33679) of the table concentrate was scanned mineragraphically at a magnification of X100 to detect native gold. The material was found to consist predominantly of goethite and hematite with minor sulphides. Only two grains of native gold were observed - both as inclusions (10 μm and 4 μm) in goethite.

A portion of the table concentrate was separated magnetically using a Frantz isodynamic magnetic separator (current 0.5 Amps, side slope 15°). Portions of the magnetic and non-magnetic products were analysed for Au and examined mineragraphically in polished section (PS 33680). No native gold was observed in the 0.5A magnetic product but 5 grains of native gold were observed in the 0.5A non-magnetic product. These were liberated and ranged from 25 μm to 75 μm in size. Some colour variation was evident, suggesting a variable Ag content of the native gold. The 0.5A magnetic product consisted of ∿98% goethite + hematite and 1-2% pyrrhotite. The 0.5A non-magnetic product contained 5-7% pyrite, 1-3% arsenopyrite, 30-40% goethite and 50-55% silicates; goethite was commonly intergrown with silicates.

The weight and Au distribution between magnetic and non-magnetic products was as follows:

		Au	
Product	Wt. %	Assay, ppm	Distribution, %
0.5 A magnetics	94.2	15.1	19.8
0.5 A non-magnetics	5.8	1000	80.2
TOTAL	100.0	(72)	100.0

These results indicate that about 80% of the Au in the concentrate is not associated with iron oxides. However, the calculated head assay is significantly greater than the 39.8 ppm previously reported and if this value (39.8 ppm) is used as the correct one, the revised distribution becomes as follows:

Product	Wt. %	Au Assay, ppm	Au Distribution, %
0.5 A magnetics	94.2	15.1	35.8
0.5 A non-magnetics	5.8	(443.9)	64.2
TOTAL	100.0	39.8	100.0

Overall, these results indicate that 15.1 ppm Au is associated with iron oxides - representing 20-36% of the total Au in the concentrate, the remainder probably being largely liberated.

A portion of the table concentrate was analysed for mercury-soluble and mercury-insoluble Au to determine the amount of essentially-liberated native gold (mercury-soluble Au) present. Results were as follows:

Au assay, ppm			Distribution, %		
Hg-soluble	Hg-insoluble	TOTAL	Hg-soluble	Hg-insoluble	TOTAL
41.3	12.4	53.7	76.9	23.1	100

The percentage of mercury-soluble gold is comparable to the percentage of Au in the non-magnetic product, indicating that much of the Au is present as essentially-liberated native gold. The mercury-insoluble Au is presumably locked in iron oxides.

3. HEAD SAMPLE (A2631/84)

The head sample was screened and the size fractions assayed for total Au by fire-assay, with the following results:

Size Fraction			Au	
BS mesh	µm	Wt. %	Assay, ppm	Distribution, %
+25	+600	14.14	6.2	15.1
-25 +52	-600 +300	29.76	8.3	42.4
-52 +100	-300 +150	17.23	5.1	15.1
-100 +200	-150 +75	8.29	4.1	5.8
-200	-75	30.59	4.1	21.6
Total		100.00	(5.8)	100.0

Prior to analysis, the +200 mesh size fractions were scanned for native gold using a stereobinocular microscope and two elongate and flattened liberated particles were detected in the -100 +200 mesh fraction. No native gold was detected in the three coarser fractions.

Portions of the +200 mesh fractions were separated at specific gravity 3.3 (to float silicates with locked Au) and the >3.3 sp.gr. product was separated magnetically using a Frantz at 0.5 Amps current and a side slope of 15° (to separate iron oxides with locked Au from liberated native gold and sulphides). The products were analysed for Au by fire-assay and AAS (in the case of the >3.3 sp.gr. 0.5A non-magnetic products which were of low weight). Results are given in Table 1.

In interpreting Table 1 the following points should be noted:

(1) In column 2 ('Spec. Grav. Fraction'), <3.30 refers to the <3.30 sp.gr. material, >3.30 0.50 M refers to the >3.3 sp.gr. 0.5 A magnetic material and >3.30 0.50 N-M refers to the >3.30 sp.gr. 0.5 A non-magnetic material.

(2) Columns 3-5 ('In Size Fractions') give the data for each size fraction - with weight and elemental distributions calculated to 100%. 'Totals for Above Size Fractions' refers to the total +200 mesh material. 'Overall Totals' refers to the weight and element distributions between 'sands' (i.e. +200 mesh material) and 'slimes' (-200 mesh material).

(3) Columns 6-7 ('In Size Fractions') give the data on the basis of the head as 100%.

(4) Columns 8-10 ('Cumulative Data') give the cumulated totals by sp.gr. product. Thus, in the -52+100 mesh size range for example, the <3.3 sp.gr. data refers to the total material with sp.gr. <3.3 in the +100 mesh size fractions.

It can be seen from Table 1 that about 80% of the Au reports in the +200 mesh fractions, which constitutes about 70% of the weight. Of this 80%, 28% reports in the <3.30 sp.gr. product (i.e. locked with silicates) - mainly in the +52 mesh size fractions, 49% reports in the >3.30 sp.gr. 0.5 A magnetic product (i.e. probably locked with iron oxides) - mainly in the -25+100 mesh size fractions, and only 2% reports in the >3.30 sp.gr. 0.5 A non-magnetic product which would contain any liberated native gold. Thus, it may be concluded that at the present size distribution there is minimal liberated native gold in the +200 mesh fractions and even if all the Au in the -200 mesh fraction is present as liberated native gold this still only amounts to 22% of the total Au being in liberated native gold.

The Au assays of the <3.30 sp.gr. products decrease with decreasing particle size whereas the Au assays of the >3.30 sp.gr. 0.5 A magnetics (i.e. iron oxides) remain fairly constant. These features are consistent with progressive liberation of Au-bearing iron oxides from silicates with decreasing particle size.

Polished sections were prepared of the unseparated +200 mesh size fractions, the -200 mesh fraction after removal of the -10 µm fraction by decantation, and of the >3.3 sp.gr. 0.5A non-magnetic products of the -25+52 mesh and -52+100 mesh fractions (PS33647-50, 33678, 33681). The following grains of native gold were observed:

<u>+25 mesh fraction</u>	None
<u>-25+52 mesh fraction</u>	One 10 µm grain of native gold included in porous goethite
<u>-52+100 mesh fraction</u>	None
<u>-100+200 mesh fraction</u>	One 10 x 30 µm grain of native gold intergrown with goethite and silicate
<u>-200 mesh (+10 µm) fraction</u>	One liberated 2 x 10 µm grain of native gold and one liberated 3 µm grain of native gold

<u>-25+52 mesh >3.3 sp.gr. 0.5A non-magnetics</u>

One composite goethite-silicate grain was observed to contain 4 blebs of native gold, 3-5 µm in size and located mainly in the goethite phase.

<u>-52+100 mesh >3.3 sp.gr. 0.5A non-magnetics</u>

No native gold was detected.

<u>Cyanidation</u>

A sample of the head ore was ground to 80% passing 200 mesh (75 µm) and subjected to agitation cyanidation using 0.05% NaCN solution at 45% solids and pH 11. The cyanide and lime consumptions were measured and the Au extraction determined at 2, 6, 24 and 48 hours. The residue assayed 0.8 ppm Au and the calculated head assay was 5.7 ppm Au. Au extraction data, and cyanide and lime consumptions were as follows:

	Extraction Time			
	2 hrs	6 hrs	24 hrs	48 hrs
Au extraction, %	39.7	74.8	84.8	86.1
Cyanide consumption, kg/tonne	0.6	0.9	1.7	2.7
CaO consumption, kg/tonne	1.9	2.1	2.0	2.3

These results show that most of the gold can be extracted by cyanide after 6 hours and that there is than only a small increase to 48 hours. Cyanide and lime consumptions are moderate.

4. DISCUSSION AND RECOMMENDATIONS

It is clear from the present work that the Table Concentrate has a significant proportion (60-80%) of its contained Au present as essentially-liberated native gold which should be amenable to amalgamation. However, it has not been established what the size range of this native gold is and this could be done by sizing a sample of the Table Concentrate and determining mercury-soluble and mercury-insoluble Au on the size fractions.

The Feed Sample contains very little liberated native gold coarser than 200 mesh and gravity recovery of native gold is likely to be low. Much of the Au in the feed is present as native gold inclusions in iron oxides and the cyanidation results indicate that good recoveries of Au can be obtained by standard cyanidation on material ground to 80% passing 200 mesh.

Further work on the Feed Sample could be directed to determining the effect of grinding on liberating native gold for possible gravity recovery. This would involve grinding the material to say 100% passing 52 mesh and 100% ~~plus~~ passing 100 mesh, sizing the ground material, and processing the size fractions in heavy liquids and magnetically similar to the procedure used for the current Feed Sample.

Investigation of the nature of Au losses in the tailings could be undertaken using the same procedure as for the current Feed Sample.

TABLE 1. GOLD DISTRIBUTION IN A 2631/84

SIZE FRACTION (BS MESH)	SR OR MAG FRACTION		IN SIZE FRACTIONS			IN HEAD SAMPLE		CUMMULATIVE DATA		
			WT %	ASSAY (ppm)	Au DISTN. %	WT %	Au DISTN %	WEIGHT %	ASSAY (ppm)	DIST %
+ 25	< 3.30		80.2	3.70	80.7	11.34	16.01	11.34	8.20	16.2
	>3.30	0.50 M	19.7	8.40	19.1	0.78	3.79	2.78	8.40	4.7
		0.50 N-M	0.1	14.0	0.2	0.62	0.05	0.02	14.00	0.0
	TOTAL		100.0	8.65	100.0	14.14	19.84	14.14	8.65	19.6
- 25+ 52	< 3.30		42.1	4.40	22.1	12.53	8.94	23.86	6.44	24.9
	>3.30	0.50 M	57.1	11.20	76.2	16.99	30.88	19.77	10.81	34.67
		0.50 N-M	0.8	17.00	1.7	0.21	0.67	0.27	16.76	0.7
	TOTAL		100.0	8.39	100.0	29.76	40.50	43.89	8.47	60.34
- 52+100	< 3.30		43.1	1.70	14.7	7.42	2.05	31.28	5.32	27.0
	>3.30	0.50 M	56.2	6.80	77.7	8.69	10.69	29.45	9.49	45.3
		0.50 N-M	0.7	55.00	7.6	0.12	1.03	0.38	28.48	1.77
	TOTAL		100.0	4.93	100.0	17.23	13.79	61.12	7.47	74.1
-110+200	< 3.30		57.0	1.80	25.0	4.72	1.38	36.01	4.86	28.3
	>3.30	0.50 M	42.5	6.40	66.4	3.52	3.66	32.98	9.16	49.0
		0.50 N-M	0.5	73.50	8.6	0.04	0.47	0.42	32.72	2.2
	TOTAL		100.0	4.10	100.0	8.29	5.51	69.41	7.07	79.64
TOTALS FOR ABOVE SIZE FRACTIONS	< 3.30		51.9	4.86	35.6					
	>3.30	0.50 M	47.5	9.16	61.6					
		0.50 N-M	0.6	32.72	2.3					
OVERALL TOTALS	SANDS		69.4	7.07	79.6					
	SLIMES		30.6	4.10	20.4					
	TOTAL		100.0	6.16	100.0					

CALCULATED HEAD ASSAY = 6.16 ppm

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28 March 1985

In reply quote: GS 1/16/0

Your Ref: 12.03 and 11.20.2200

Director-General
Department of Mines & Energy
PO Box 151
EASTWOOD SA 5063

Attention: Mr R. Wildy

REPORT G 6193/85 - PART II

YOUR REFERENCE:	Letter dated 15 February 1985, ref. SR/5/6/66/10
IDENTIFICATION:	As given in report
MATERIAL:	Mill products
LOCALITY:	Waukaringa, South Australia
DATE RECEIVED:	19 February 1985
WORK REQUIRED:	Gold mineralogy and photomicrography

Investigation and Report by: Frank Radke

Chief, Geological Services Section: Dr Keith J. Henley

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for Dr William G. Spencer
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GOLD MINERALOGY OF WAUKARINGA MILL PRODUCTS

1. INTRODUCTION

Mill product samples from the Waukaringa gold processing plant which were examined by AMDEL (report G 6193/85) were submitted for further mineralogical analysis to determine the location of gold. The following samples were submitted:

Sample	Sample No.
Table concentrate	A 2628/84
Head	A 2631/84
Head, -200 # fraction	A 2631/84 (-200 #)

It was also requested that photomicrographs (35 mm colour slides) be taken of the gold particles observed in report G 6193/85.

2. TABLE CONCENTRATE SAMPLE A 2628/84

This sample was sized at 36, 52 and 72 mesh (420, 300 and 210 microns) and the four size fractions analyzed for mercury soluble and mercury insoluble gold.

The gold assays and size distribution in the table concentrate are given in Table 1. The gold is concentrated in the finest size fraction which has 63% of the gold in 22% of the sample. This gold is mostly Hg soluble indicating it is most likely fine liberated gold. The proportions of Hg-soluble gold generally increases with finer particle size (this is particularly apparent for the two finer size fractions) reflecting better liberation of finer particle size. The calculated head assay of 41 ppm gold agrees well with the assay of 39.8 ppm reported but not with the calculated head of 72 ppm for the magnetic separation products of report G 6193/85.

3. HEAD SAMPLE A 2631/84

This sample was wet rod-milled to 99% passing 106 microns and then sized by wet screening at 53 and 20 microns. The two coarsest size fractions (+53 and -53+20 microns) were separated in methylene iodide (sp.gr. 3.3). The size fractions and all of the heavy liquid products were analyzed for gold and the heavy products of the +53 and -53+20 micron size fractions were analyzed for Hg-soluble and Hg-insoluble gold. The gold distribution is given in Table 2 which has the same format as Table 1 of report G 6193/85.

The calculated head assay of 4.8 ppm Au is somewhat lower than both the head assay (5.8 ppm) and calculated head assay (6.16 ppm) given for this sample in report G 6193/85. Assays of the size fractions are 6.0 ppm for the +53 micron size fraction which agrees moderately well with the calculated assay of the 7.6 ppm and 7.6 ppm for the -53+20 micron size fraction which is much higher than the calculated head assay of 3.6 ppm. It is considered likely that the major discrepancies of the low calculated assay for the -53+20 micron size fraction and the low head assay are due to a low value for the -53+20 micron heavy products. A calculated assay (16.3 ppm) for the -53+20 micron heavy products using the assay of 7.6 ppm for the size fraction gives the gold distribution in Table 2A. This is considered more accurate and will be used in the discussion below.

From Table 2A it can be seen that approximately 3% of the gold is locked with light gangue in the +20 micron light products, 65% is in the +20 micron heavy products and 32% in the -20 micron slimes. The gold in the heavy products includes liberated gold and gold locked in particles with an average specific gravity above 3.3. Most of the locked gold in the heavy products is probably intergrown with iron oxides or other dense minerals such as pyrite. Mercury soluble and mercury insoluble gold was determined in the heavy products and gave the following results:

Size Fraction	Au Assays (ppm)			Distributions in Product		
	Hg-soluble	Hg-insoluble	Total	Hg-soluble	Hg-insoluble	Total
+53 μm	9.2	4.2	13.4	69	31	100
-53+20 μm	4.2	3.3	7.5	56	44	100
Total Heavy Products	(6.8)	(3.8)	(10.6)	64	36	100

The above data indicate that 64% of the gold in the heavy products is liberated and 36% locked with no surfaces available to amalgamation. The proportion of mercury soluble gold in the heavy products is similar to the proportion of mercury soluble gold in the table concentrate.

4. THE -200 MESH HEAD SAMPLE

This sample was cyclosized after dispersion and the size fractions recombined into -75+20 micron and -20 micron size fractions for further work. The -75+20 micron size fraction was separated in methylene iodide (sp.gr. 3.3). Both the size fractions and the heavy liquid light products (sp.gr. <3.3) were analyzed for total gold and the heavy liquid heavy products (sp.gr. >3.3) were analyzed for Hg-soluble and Hg-insoluble gold.

The gold distribution for this sample is given in Table 3. The calculated assay for the +19 micron size fraction of 3.6 ppm is lower than the actual assay of 4.3 ppm and could reflect a slightly low assay for the heavy products. For the whole sample a calculated head of 3.4 ppm gold was obtained. Approximately 3% of the gold is locked with light gangue in the +19 micron light products, 34% is in the +19 micron heavy products and 63% in the -19 micron slimes. The results for the Hg-soluble and Hg-insoluble gold on the +19 micron heavy products are given below:

Au Assays			Au Distribution		
Hg-soluble	Hg-insoluble	Total	Hg-soluble	Hg-insoluble	Total
5.74	2.7	8.44	68	32	100

This shows that 68% of the gold in the heavy products is essentially liberated and 32% is locked so that it is unavailable to amalgamation. This is similar to the value obtained for the table concentrate (67%) and for the heavy products of the head sample (A 2631/84).

5. PHOTOMICROGRAPHY

35 mm colour slides were taken of gold grains observed in report G 6193/85. The slides are listed in Appendix A with brief captions. The slides are enclosed with this report.

TABLE 1: GOLD DISTRIBUTION IN TABLE CONCENTRATE
(SAMPLE A 2628/84)

Size Fraction µm	Dist. Wt. %	Total Gold		Hg-soluble Gold		Hg-insoluble Gold	
		Assay ppm	Dist. %	Assay ppm	Dist. %	Assay ppm	Dist. %
+420	19.22	10.48	4.9	2.2	1.5	8.28	11.9
-420+300	34.28	10.5	8.7	1.66	2.0	8.84	22.7
-300+210	24.58	38.5	23.0	19.8	17.5	18.70	34.5
-210	21.92	119.2	63.4	100.4	79.0	18.76	30.9
Total Sample	100.00	(41)*	100.0	(28)*	100.0	(13)*	100.0

	% Hg-soluble Gold in Size Fraction	Gold Distribution in Sample	
		Hg-soluble	Hg-insoluble
+420	21	1.0	3.9
-420+300	16	1.4	7.3
-300+210	51	11.7	11.3
-210	84	53.3	10.1
Total Sample	68	67.4	32.6

* Calculated assays in parentheses.

TABLE 2: GOLD DISTRIBUTION IN HEAD SAMPLE (A 2631/84)

Size Fraction µm	Sp.Gr. Fraction	In Size Fractions			In Head Sample	
		Wt. %	Au Assay ppm	Au Dist. %	Wt. %	Au Dist. %
+53	<3.3	46.9	1.0	6.2	11.9	2.5
	>3.3	53.1	13.4	93.8	13.5	37.6
	Total	100.0	(7.6)	100.0	25.4	40.1
-53+20	<3.3	55.2	0.5	7.6	15.2	1.6
	>3.3	44.8	7.5	92.4	12.3	19.2
	Total	100.0	(3.6)	100.0	27.5	20.8
Totals for above Size Fractions	<3.3	51.2	(0.7)	4.1		
	>3.3	48.8	(10.6)	56.8		
Overall Totals	+20 µm	52.9	(5.5)	60.9		
	-20 µm	47.1	4.0	39.1		
Total Sample		100.0	(4.8)	100.0		

TABLE 2A: GOLD DISTRIBUTION IN HEAD SAMPLE (A 2631/84) WITH ADJUSTED ASSAY FOR -53+20 MICRON HEAVY PRODUCT

Size Fraction µm	Sp.Gr. Fraction	In Size Fractions			In Head Sample	
		Wt. %	Au Assay ppm	Au Dist. %	Wt. %	Au Dist. %
+53	<3.3	46.9	1.0	6.2	11.9	2.0
	>3.3	53.1	13.4	93.8	13.5	30.7
	Total	100.0	(7.6)	100.0	25.4	32.7
-53+20	<3.3	55.2	0.5	3.7	15.2	1.3
	>3.3	44.8	(16.3)	96.3	12.3	34.0
	Total	100.0	7.6	100.0	27.5	35.3
Totals for above Size Fractions	<3.3	51.2	(0.7)	3.3		
	>3.3	48.8	(14.8)	64.7		
Overall Totals	+20 µm	52.9	(7.6)	68.0		
	-20 µm	47.1	4.0	32.0		
Total Sample		100.0	(5.9)	100.0		

TABLE 3: GOLD DISTRIBUTION IN -200 MESH HEAD SAMPLE

Size Fraction µm	Sp.Gr. Product	In Head Sample Wt. %	In Head Sample Au Assay ppm	In Head Sample Au Dist. %	In Size Fraction Wt. %	In Size Fraction Au Dist. %
+19	<3.3	21.1	0.4	2.5	60.6	6.8
	>3.3	13.7	8.44	34.1	39.4	93.2
	Total+19	34.8	(3.6)	36.6	100.0	100.0
-19		65.2	3.3	63.4		
Total Sample		100.0	(3.4)	100.0		

APPENDIX A

LIST OF 35 mm COLOUR SLIDES

Slide No.	Sample	Polished Section No.	Scale/Field Length(1) (mm)	Caption
1-2(2)	A 2628/84	33679	0.5	10 μm gold grain in goethite
3-4	"	"	0.2	
5	"	"	2.0	
6	A 2628/84	33679	0.5	4 μm gold grain in goethite
7-8	A 2628/84	33680	1.0	Liberated native gold grains
9-10	"	"	0.5	
11	A 2631/84 -25+52 #	33648	1.0	10 μm gold grain in porous goethite
12		"	0.5	
13-14	"	"	0.2	
15	A 2631/84 -100+200 #	33650	1.0	10 × 30 μm gold grain intergrown with goethite and non-opaque gangue
16		"	0.5	
17-18	"	"	0.2	
19	A 2631/84 -200 #+10 μm	33678	0.5	10 × 2 μm liberated gold particle
20-21		"	0.2	
22-23	A 2631/84 -200 #+10 μm	33678	0.5	3 μm liberated gold particle
24-25		"	0.2	
26	A 2631/84 -25+52 # >3.3 non-mags	33681	1.0	Four small gold inclusions in goethite
27		"	0.5	
28-29	A 2631/84 -25+52 # >3.3 non-mags	33681	0.2	Close up of two inclusions
30-31	A 2631/84 -25+52 # >3.3 non-mags	33681	0.1	Close up of two other inclusions

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(1) This figure is the largest dimension of the colour slide.

(2) Where two slides of the same field are taken the second was taken with less exposure and may show internal details and colour of gold better.

Details of slides listed in Amdel Report G6193/85- Part II

Sample No.	Description	Grade
A2628/84 Dumps A + B	No. 1 Concentrate	39.8 g/t
A2631/84 Dumps A + B	Head Sample	5.82 g/t

APPENDIX D
Gold Recovery Tests
Dumps A + B (combined) and Dump C
(AMDEL Report T6086)

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9 April 1985

OD 1/16/0

The Director General
South Australian Department of
Mines and Energy
PO Box 151
EASTWOOD SA 5063

Attention: Mr R. Wildy

REPORT: T6086

YOUR REFERENCE:	SR/5/6/66/10
MATERIAL:	Gold tailings
LOCALITY:	Waukaringa
WORK REQUIRED:	Gold recovery tests

Investigation and Report by:	P.G. Capps
Manager, Operations Division:	Peter M. Cameron

for Brian S. Hickman,
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GOLD RECOVERY FROM WAUKARINGA TAILINGS

1. INTRODUCTION

Re-treatment of old tailings dumps at Waukaringa in South Australia, was recently commenced. The processing plant consisted of screening, with rejection of oversize and agglomerate particles, followed by jigging and tabling.

Overall recovery of gold and final concentrate grades have been extremely poor, and the operation has not been profitable. AMDEL was requested by the S.A.D.M.E. to undertake a programme of work to investigate the potential for improving gold recovery from the existing plant. The metallurgical test programme was specified by the Department, and the results are contained in this Report. Some mineralogical work, which has reported separately by the Geological Services Section (Report G6193), was also carried out.

2. MATERIAL EXAMINED

Samples containing material from Dumps A plus B (combined) and Dump C were submitted by SADME for testwork to investigate impact crushing, tabling and amalgamation. Head samples of each Dump sample were assayed for gold, and results are shown below:

Dump A + B	4.3 g/t Au
Dump C	2.6 g/t Au

Further samples, representing Dumps A, B, C and G, were subsequently provided for direct cyanide leach tests.

3. PROCEDURE AND RESULTS

3.1 Impact Crushing

Sizing analyses of 'as received' samples (Dumps A + B, Dump C) were obtained by first screening off plus 38 mm agglomerates, then subsampling the undersize for dry screening down to 2.8 mm. The minus 2.8 mm fractions were wet screened down to 38 µm.

The non-sized balances of each sample were then crushed in a small impact crusher fitted with four striker bars. Subsamples were riffled from the crushed products and sized down to 38 µm (as for the 'as received' material).

Results of the sizings of 'as received' and crushed samples are shown in Table 1. Moisture contents are also shown.

Both Dumps A + B and Dump C samples contained almost 30% by weight of agglomerates in the plus 25 mm size range. Impact crushing reduced the top size to only 4% plus 2.8 mm, and significantly increased the weight proportions in the 1.1 to 2.8 mm range. The proportions of minus 38 µm fines were also increased, from 14% to 24% for Dumps A + B, and 17% to 40% for Dump C.

3.2 Desliming

The impact crushed samples (Section 3.1) were each slurried at 25% solids and deslimed through a 25 mm diameter cyclone fitted with a 9 mm vortex finder. A 6.5 mm spigot was used for Dumps A + B sample, and a 9 mm spigot for Dump C.

Underflow fractions were collected, dried and sampled for gold assay. Overflow fractions were subsampled during the desliming operation and assayed for gold also.

Weight proportions and assays of the cyclone products are shown in Table 2. The finer sizings of Dump C material in Table 1 are again reflected in the high weight proportion of 53% of the Dump C sample that reported to cyclone overflow, and containing 43% of the gold. For Dumps A + B material, only 27% by weight and 22% of the gold was contained in the cyclone overflow.

There was only a slight degree of gold concentration into the cyclone underflow products for both samples (3.3 to 3.5 g/t for Dumps A + B, 2.8 to 3.4 g/t for Dump C).

3.3 Table Concentration

Deslimed samples (cyclone underflow products, Section 3.2) were concentrated on a laboratory size Wilfley shaking table, using a 12.5 mm stroke at 400 strokes per minute.

Product weights and assays are shown in Table 3.

Tabling of Dump C material allowed a reasonably small weight proportion to be separated as concentrate and middling #1 (7.2% and 6.6% respectively). However, only a slight concentration of gold was achieved, with gold grades of 7 g/t and 5.6 g/t for the concentrate and middlings #1 products respectively. These relate to a calculated feed grade of 4 g/t Au and a tailings grade of 3.8 g/t Au. The concentrate and first middling grades account for only 12.6% and 9.2% of the gold present in the table feed.

It was not possible to collect a low weight concentrate from the Dump A + C sample, for which most of the material remained essentially unseparated on the table. There was not a clear delineation of a heavy mineral band apparent. Thus the concentrate product for Dump A + B contained 39.5% by weight and 42.3% of the gold in the table feed, at a grade of 6.0 g/t Au. As for Dump C, the concentration of gold to concentrate was very poor, at a concentrate grade of 6.0 g/t Au compared with a calculated head of 5.6 g/t Au and a tailings grade of 5.4 g/t Au.

It should be noted that the above gold recoveries relate only to table feed, and do not make allowances for gold reporting to the cyclone overflow products (Section 3.2). For Dumps A + B and Dump C respectively, only 77.9% and 57.5% of the gold reported to table feed, giving overall gold recoveries of 33.0% and 7.2% respectively to table concentrates.

3.4 Amalgamation of Table Concentrates

Samples of table concentrates (Section 3.3) were amalgamated with mercury in rolling bottles for a period of 2 hours. After that time, the mercury and residues were separated by panning, and assayed for gold.

Results are shown in Table 4. Two amalgamation tests were carried out using Dump A + B material, one for which twelve ceramic balls of 12.5 mm diameter were present in the slurry to examine the effect of cleaning the particle surfaces. The other test was carried out without the ceramic balls, as was the test on Dump C material.

Only 11% of the gold in Dump A + B table concentrate (ie 3.6% of total gold in the original Dump A + B sample) was amalgamated. This figure was not improved by the presence of ceramic balls in the amalgamation slurry.

Gold recovery to amalgam from Dump C table concentrate was higher at 38% . This represents an overall recovery of 2.7% of the gold present in the original Dump C sample.

3.5 Direct Cyanide Leaching

Individual samples representing Dumps A, B, C and G were blended in the approximate ratio of the dump tonnages, as below:

Dump	Weight(%)
A	11
B	12
C	43
G	34

The blended composite was dried at 60°C and impact crushed to break up agglomerates. One sample was riffled out and stage crushed and ground to 100% minus 1 mm, and a further portion was stage ground to 80% minus 75 μm.

Both samples (100% minus 1mm and 80% minus 75 μm) were cyanide leached in agitated beakers, using solutions of 0.05% NaCN and pH 11 (controlled by CaO addition) at 35% solids. Aeration was provided continuously to the slurries throughout the leach period of 48 hours.

Solution samples were taken after 2, 6, 24 and 48 hours leaching for titration to determine the NaCN concentration and for gold assay. Reagents (NaCN, CaO) were added as required after each titration to re-establish the leaching conditions as described above. At the completion of the 48 hour leach period, residues were washed, dried, weighed and assayed for gold. Plus and minus 150 μm fractions of coarse head and residue samples were also assayed for gold.

Some previous work carried out at AMDEL had indicated extremely high reagent consumptions for Dump C material. In order to reduce the effect of soluble material in the composite sample, the 100% minus 1 mm sample was soaked overnight in water, and the sample filtered and washed before cyanide leaching. The water had acquired a deep red/brown colour overnight. A similar washing effect was achieved for the fine sample by grinding to the required size of 80% minus 75 μm in water, then filtering and washing the ground product.

Results are shown in Table 5. Calculated head assays and gold extractions were calculated from residue and solution assays.

Extractions of 57% and 76% of the gold in the coarse and fine samples respectively were achieved after 48 hours leaching, although only minor dissolution occurred after the initial 6 hours. The lower gold extraction from the coarse sample is further reflected in the assays of plus 150 μm and minus 150 μm residue samples. Although the minus 150 μm residue fraction contained only 0.8 g/t Au, the plus 150 μm fraction (36.3% by weight) contained 5 g/t Au.

Cyanide consumptions were high at 3.5 kg/t for the coarse sample and 3.0 kg/t for the fine sample after 6 hours, and increasing to 8.5 kg/t and 6.7 kg/t respectively over 48 hours. Lime consumptions were also high at 10 kg/t and 11.7 kg/t after 6 hours for coarse and fine samples respectively.

4. DISCUSSION

As expected, impact crushing was effective in size reducing agglomerated material from the dumps. The use of crushers in the Waukaringa circuit would allow an additional 30 to 40% of the material from Dumps A, B and C to be processed.

However, the minus 38 μm contents were significantly increased by crushing. During desliming of crushed products, 22% of the gold in the Dumps A + B sample and 43% of the gold in the Dump C samples was rejected into the cyclone overflow products.

Table concentration was not successful in producing high grade gold concentrates from either sample (6 g/t Au Dump A + B, 7 g/t Au Dump C). Recoveries were also low (42% Dump A + B, 13% Dump C), and the results indicate that the majority of the gold in the table feed samples is not present in liberated form.

The poor liberation indicated by the tabling results is further demonstrated by the results achieved during amalgamation of the table concentrates. Only 11% of the gold in Dump A + B table concentrates (4.6% of gold in table feed, 3.6% of gold in total sample) and 38% of the gold in Dump C table concentrate (4.8% of gold in table feed, 2.7% of gold in total sample) amalgamated with mercury and can therefore be considered as essentially liberated gold.

Direct cyanide leaching achieved approximately 75% recovery of gold from a ground composite sample representing Dumps A, B, C and G and containing approximately 4 g/t Au. Leaching of unground material achieved only 60% extraction, with minimal gold dissolution from the plus 150 μm size fraction. Grinding of the dump material will be necessary for successful treatment by cyanidation.

Both cyanide and lime consumptions were high. Lime consumption continued steadily throughout the leach period, and would have been associated with reactions between the cyanide and the acid species. Lower cyanide consumption and possibly higher gold recovery (due to easier maintenance of the correct cyanide solution strength) would probably be achieved by pre-agitation of the slurries with air and lime addition prior to the addition of cyanide.

5. CONCLUSIONS AND RECOMMENDATIONS

1. Insufficient gold is liberated in the present plant feed to achieve satisfactory gold recovery in a gravity circuit.

2. Direct cyanide leaching of ground feed achieved 75% recovery of gold from a composite sample representing Dumps A, B, C and G.

3. Direct cyanide leaching of unground feed did not provide adequate gold extraction (60%).

4. Reagent consumptions during cyanide leaching were high. Cyanide consumption could be reduced by pre-conditioning the slurry with lime and air addition.

5. In view of the relative recoveries achieved by gravity concentration and cyanide leaching it is recommended that further tests be carried out on representative dump samples to optimise gold recovery by direct cyanide leaching.

TABLE 1 : SIZING OF 'AS RECEIVED' AND 'IMPACT CRUSHED' SAMPLES

	Dumps A + B		Dump C	
	As Received	Impact Crushed	As Received	Impact Crushed
Moisture (%)	2.7	2.2	24.5	23.6
Weight (%) *				
+ 25 mm	28.7	-	28.6	-
-25 + 12.7	3.7	-	10.2	-
-12.7 + 2.8	7.6	4.0	26.0	4.4
- 2.8 + 1.7	0.1	2.6	0.5	5.1
- 1.7 + 1.1	0.2	2.0	0.5	5.4
- 1.1 + 0.5	5.1	8.6	1.2	13.8
- 0.5 + 0.355	9.4	10.7	0.7	1.1
- 0.355+ 0.212	11.3	14.2	1.2	2.3
- 0.212+ 0.106	10.3	16.5	2.4	4.6
- 0.106+ 0.053	7.0	12.6	7.4	14.9
- 0.053+ 0.038	2.7	4.4	3.9	8.1
- 0.038	13.9	24.4	17.4	40.3
	100.0	100.0	100.0	100.0

* Sizing weights on dry weight basis

TABLE 2 : DESLIMING OF IMPACT CRUSHED SAMPLES

	Dumps A + B	Dump C
Cyclone Underflow		
Weight (%)	73.1	46.7
Au Assay (g/t)	3.5	3.4
Au Distribution (%)	77.9	57.5
Cyclone Overflow		
Weight (%)	26.9	53.3
Au Assay (g/t)	2.7	2.2
Au Distribution (%)	22.1	42.5
Calculated Head Au (g/t)	3.3	2.8

TABLE 3 : TABLE CONCENTRATION

	Weight	Au	
	(%)	(g/t)	Dist.(%)
Dumps A + B			
Con	39.5	6.0	42.3
Mid	37.0	5.3	35.0
Tail	23.5	5.4	22.7
Total	100.0	5.6	100.0
Dump C			
Con	7.2	7.0	12.6
Mid #1	6.6	5.6	9.2
Mid #2	16.5	3.0	12.3
Tail	69.7	3.8	65.9
Total	100.0	4.0	100.0

TABLE 4 : AMALGAMATION OF TABLE CONCENTRATES

	Residue Assay Au (g/t)	Calc.Head Au (g/t)	Recovery to Amalgam (%)
Dumps A + B	3.8	4.3	10.9
Dumps A + B (ceramic balls present during amalgamation)	3.7	4.1	10.6
Dump C	3.0	4.7	38.0

TABLE 5 : DIRECT CYANIDE LEACHING - DUMPS A,B,C,G

		Particle size	
		100% -1 mm	80% - 75 μm
Head Assay, Au (g/t)		4.0	4.0
+ 150 μm, Au (g/t)		6.6	-
- 150 μm, Au (g/t)		5.0	-
Calc. Head Assay, Au (g/t)		4.7	4.7
Residue Assay, Au (g/t)		2.0	1.1
+ 150 μm, Au (g/t)		5.0	-
- 150 μm, Au (g/t)		0.8	-
Au Extraction (%)	2 hour	50.2	64.8
	6	55.6	73.5
	24	59.4	72.2
	48	57.1	76.4
NaCN Consumption (kg/t)	2 hour	2.2	2.0
	6	3.5	3.0
	24	6.1	5.0
	48	8.5	6.7
CaO Consumption (kg/t)	2 hour	7.3	9.6
	6	10.0	11.7
	24	12.0	13.2
	48	13.5	14.4

APPENDIX E
Preliminary Cost Study for Cyanidation Plant at Waukaringa
(AMDEL Report T6116 OD)

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28/6/85

OD 1/16/0

The Director
South Australian Department of
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PO Box 151
EASTWOOD SA 5063

Attention: Mr R. Wildy

REPORT: T6116 OD

YOUR REFERENCE: 11-20-2200

WORK REQUIRED: Preliminary Cost Study for Cyanidation plant at Waukaringa

Investigation and Report by: W.M. Walker

Manager, Operations Division: Peter M. Cameron

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PRELIMINARY COST STUDY FOR CYANIDATION PLANT AT WAUKARINGA

1. INTRODUCTION

Re-treatment of old tailings slumps at Waukaringa in South Australia, was recently commenced. The processing plant consists of screening followed by jigging and tabling. Overall recovery of gold and final concentrate grades have been extremely poor.

In work undertaken by AMDEL for the SADME, it was demonstrated that the gold in the dumps was not liberated and that gravity concentration was therefore unable to achieve satisfactory gold recovery (Report G6193, 7 February, 1985, describes the mineralogical study and Report T6086, 9 April 1985, describes gravity separation and cyanidation testwork).

Direct cyanidation did not give high recoveries but cyanidation of the tailings ground to 80% minus 75 µm gave recoveries of 75% of the gold and this method was recommended for further optimisation tests.

As a result of this recommendation, SADME requested AMDEL to make preliminary estimates of capital and operating costs for such a cyanidation plant.

2. ADDITIONAL TESTWORK

In the previous direct cyanidation testwork (Report T6086), the sample was soaked overnight in water and the sample filtered and washed before cyanide leaching. This treatment resulted in much of the acid being removed prior to cyanidation and gave 6 hour reagent consumptions of 11.6 kg/t lime and 3.0 kg/t cyanide for a gold recovery of 76%. While acid removal is obviously desirable, it was felt that such a system would not be possible at Waukaringa for a number of reasons.. Large quantities of water are unlikely to be available; the acid removed would require neutralisation before disposal; the additional capital cost of thickness or filters would be high.

A further test was made therefore, in which the sample (ground to 80% minus 75 µm) was conditioned with lime prior to cyanidation. The sample was agitated with lime at pH 10.5 overnight, and then cyanided at 0.05% NaCN and pH 11. The results of this test were:

		Consumptions	
Time h	Gold Extraction %	NaCN kg/t	Lime kg/t
2	69	1.5	83.8
6	73	2.2	83.9
24	71	4.1	85.2
48	71	4.0	87.1

The head sample contained 4.2 g/t Au.

From these results, the preliminary cost estimate was based on a plant with pre-conditioning of the tailings with lime, a minimum of 6 h leaching, a gold extraction of 70%, a cyanide consumption of 4 kg/t and a lime consumption of 85 kg/t.

3. PLANT DESIGN

For the purpose of this preliminary capital cost estimate, pretreatment with lime has been assumed, the lime being conveniently added to the ball mill. The plant will be operated on a batch basis on either 1, 2 or 3 shifts per day to give treatment rates of 50, 100 or 150 t/d. Since the tailings dumps contain as estimated 40,000 t at 4 g/t Au, the required plant life will be approximately 1 to 3 years depending on how many shifts per day are utilised.

A schematic flowsheet of the plant is shown in Figure 1.

At the start of a shift, slurry will be pumped from a 70 kl leach vessel through a carbon retrieval screen to the tailings dam. As carbon is recovered from the screen deck, a portion will be split out for stripping and regeneration while the balance will be held in intermediate storage ready for recycling to the next batch. Emptying the vessel will take approximately one hour.

When the vessel is empty, new ore will be fed to the ball mill at a rate of 7 t/h. It has been assumed that the feed recovered from the tailings dumps will be 80% minus 3 mm and that for grinding to 80% minus 75 µm, the Work Index will be 15 kWh/t. (ie total power required = 110 kW). Open circuit grinding has been assumed but the mill should be fitted with a 0.85 mm trommel to screen out organics and gross oversize. Lime at a rate of 85 kg lime/t feed will be fed into the ball mill with the feed. The mill discharge along with cyanide pellets will be pumped to the leach vessel. Also fed to the leach tank will be activated carbon recovered from the tailings screen plus the make-up quantity of new or regenerated carbon to replace that removed for gold recovery.

The leach tanks will be 70 kl capacity air agitated pachucas, thus utilising air for both agitation and the maintenance of the dissolved oxygen level in solution necessary for efficient gold dissolution. The 50 tpd option will require 1 tank and will give about 16 h leaching period. The 100 tpd option will require 2 tanks and will also give 16 h leaching while the 150 tpd option will require 2 tanks thus giving about 8 h leaching.

Carbon will be fed to the leach tank at a rate equivalent to about 30 g/l ie a 70 kl batch will require 2.1 t carbon. Assuming a loading of about 1500 g/t gold on the carbon, about 0.1 t carbon will require to be split from each batch for gold recovery, the remaining 2 t carbon going to a storage tank until recycled (with 0.1 t fresh carbon make-up) to the next batch. An agitated carbon storage tank of 5 kl capacity with a 2.3 kW agitator has been assumed. The loaded carbon can be stored in 200 l drums until sufficient quantity is available for batch recovery of the contained gold by elution and electrowinning. It has been assumed that elution will be made on 1 tonne batches contained in an epoxy-lined 2 kl vessel. Initially, a 5% HCl solution will be percolated through the carbon for about 8 h to remove any build-up of calcium picked up from the process water. After a brief water wash, the gold can then be eluted by circulating a hot solution containing 1 to 2 % NaOH and 0.2 to 0.5 % NaCN at 95°C for about 60 h. The eluant,in turn will be pumped to the electrolytic cells where the gold will be deposited on the steel wool cathode, the barren eluant being returned to the elution vessel. On the completion of elution, the stripped carbon will be discharged to 200 l drums and stored until required as make-up to the leach vessels.

It is normal to regenerate stripped carbon every 5 cycles or so by heating in a steam atmosphere to about 800°C. For this small tonnage operation, it has been assumed that sufficient new carbon will be purchased to permit 5 cycles of operation without the need to install a regeneration kiln. On completion of the project, arrangements must be made for the resale of the carbon inventory and its contained gold (about 50-100 g/t after prolonged elution).

4. CAPITAL AND OPERATING COST ESTIMATES

Preliminary cost estimates have been prepared based on equipment costs contained in AMDEL files. Purchased equipment costs as shown, for the 150TPD plant, in Table 1 are $580,000. For the 100 TPD and 50 TPD options, these costs reduce to $520,000 and $430,000 respectively. The major components are in the cost of the ball mill and lime storage silo.

Fixed capital investment estimates for the 3 plant sizes are given in Table 2. These are in the range $1 x 10^6 ± 20%.

The operating costs over the life of the plant are given in Table 3. These costs are $1.23 x 10^6 excluding any labour costs, administrative or selling costs, or financial expenses. It should be noted that lime costs for this plant are very high compared with most gold treatment plants.

5. PROFITABILITY

The recovered gold is worth (at US $316 per ounce):

$$40{,}000 \times 4 \times 0.7 \times 15 = \$1.68 \times 10^6$$

Costs in recovering this gold are:

Operating cost (Table 3)	$1.23 x 10^6
Capital cost (Table 2)	$1.00 x 10^6 ± 20%
less, salvage value	$0.25 x 10^6
Total	$2.0 x 10^6 ± 10%

On this basis, the venture is not profitable. Obviously, since the plant will only have a short life, it may be possible to reduce significantly the required capital investment by using used equipment. For example, used ball mills of about the required size are frequently advertised for about $60,000. In this way, the venture may become marginally profitable.

TABLE 1: PURCHASED FIXED EQUIPMENT (150 TPD Plant)

Item	Cost, $	
1. Feed bin, feeder and Conveyor 7 m^3 bin, 10 TPH	20,000	
2. Ball Mill 2.4 m x 1.8 m x 110 kW	250,000	2
3. Lime silo, 150 T capacity	150,000	50,000 !
4. Lime feeder, Conveyor 600 kg/h	10,000	
5. Cyanide bin & feeder, 30 kg/h	2,000	
6. Mill discharge sump and pump 4/3 Warman, 12 kW	8,000	
7. Pachuca, 2 x 70 kl	40,000	
8. Air Compressor, 400 m^2/h	17,000	
9. Carbon sump and pump 4/3 Warman, 12 kW	8,000	
10. Carbon retrieval screen, 0.6 x 0.8 m sieve bend	5,500	
11. Carbon tank and agitator, 5 kl, 2.3 kW	8,500	
12. Tailings sump and pump, 4/3 Warman, 12 kW	8,000	
13. Carbon recycle pump, diaphragm	4,000	
14. Elution vessel, epoxy-lined 2 kl	5,000	
15. Acid tank and agitator, epoxy-lined 2 kl, 2.3 kW	9,000	
16. Eluate tank and agitator 5 kl, 2.3 kW	8,500	
17. Acid pump, 5 kl/h	5,500	
18. Eluate pump, 5 kl/h	5,500	
19. Steam generator 100 kg/h	7,500	
20. Eluate heat exchanger, 2 m^2	1,500	
21. Electrowinning cell and power source	6,500	
TOTAL	$580,000	

TABLE 2: CAPITAL INVESTMENT

Item	Cost, $\$10^3$ 50 TPD	100 TPD	150 TPD
1. Delivered equipment (1.1 x Purchased)	473	572	638
2. Installation, 10% of (1)	47	57	64
3. Foundations and steelwork, 15% of (1)	71	86	96
4. Pipework, 5% of (1)	23	28	32
5. Painting, 5% of (1)	23	28	32
6. Instrumentation plus laboratory equipment	10	10	10
7. Buildings, storage shed, lab caravan etc	15	15	15
8. Fencing	10	10	10
9. Tailings dam, incl. fence,	20	20	20
10. Plant design, drawings	30	30	30
11. Supervision during construction	30	30	30
12. DIRECT COSTS	752	886	977
13. Contingency, 15% of (12)	113	134	148
14. FIXED CAPITAL, 12 + 13	865	1020	1125

NOTE: No cost has been allowed for electrical generator, materials handling equipment or provision of water supply, all of which are already available.

TABLE 3: OPERATING COSTS (over life of plant)

	Item	Cost, \$ 10^3
1.	Reagents	
	NaCN, 4 kg/t at \$1800/t	288
	Lime, 85 kg/t at \$200/t	680
	Carbon, 15 t at \$3000	45
	HCl, 10t at \$300	3
	NaOH, 1 t at \$800	1
2.	Fuel, loader, 40,000 l at \$0.6	24
	Steam, 25,000 l at \$0.6	15
3.	Water, 50,000 kl at \$0.5	25
4.	Electricity, 1,000,000 kWh at \$0.1	100
5.	Maintenance, 5% of Fixed Capital	50
	TOTAL, excluding labour costs, administrative or selling costs, or financial expenses	\$1,231

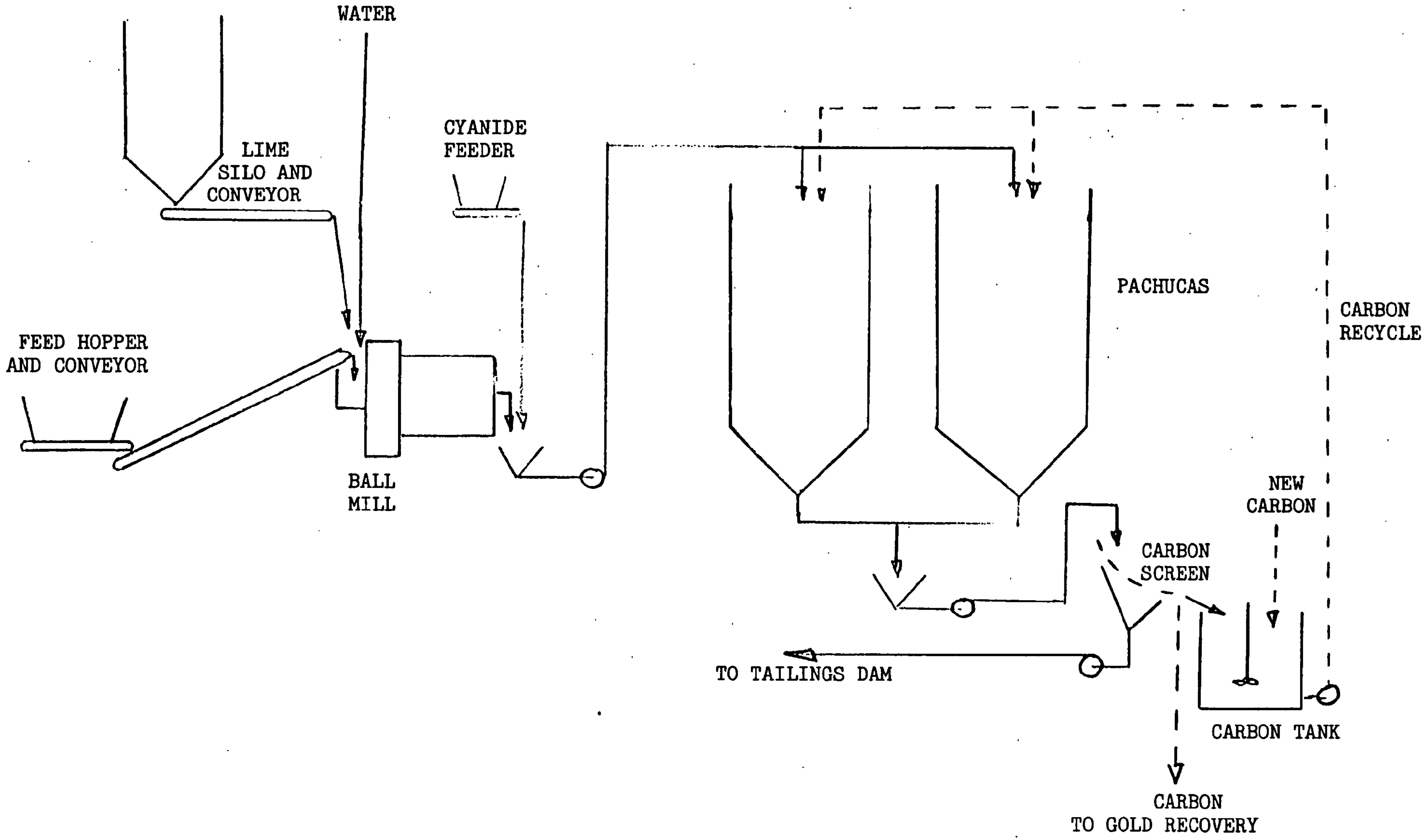


FIGURE 1 : SCHEMATIC FLOWSHEET

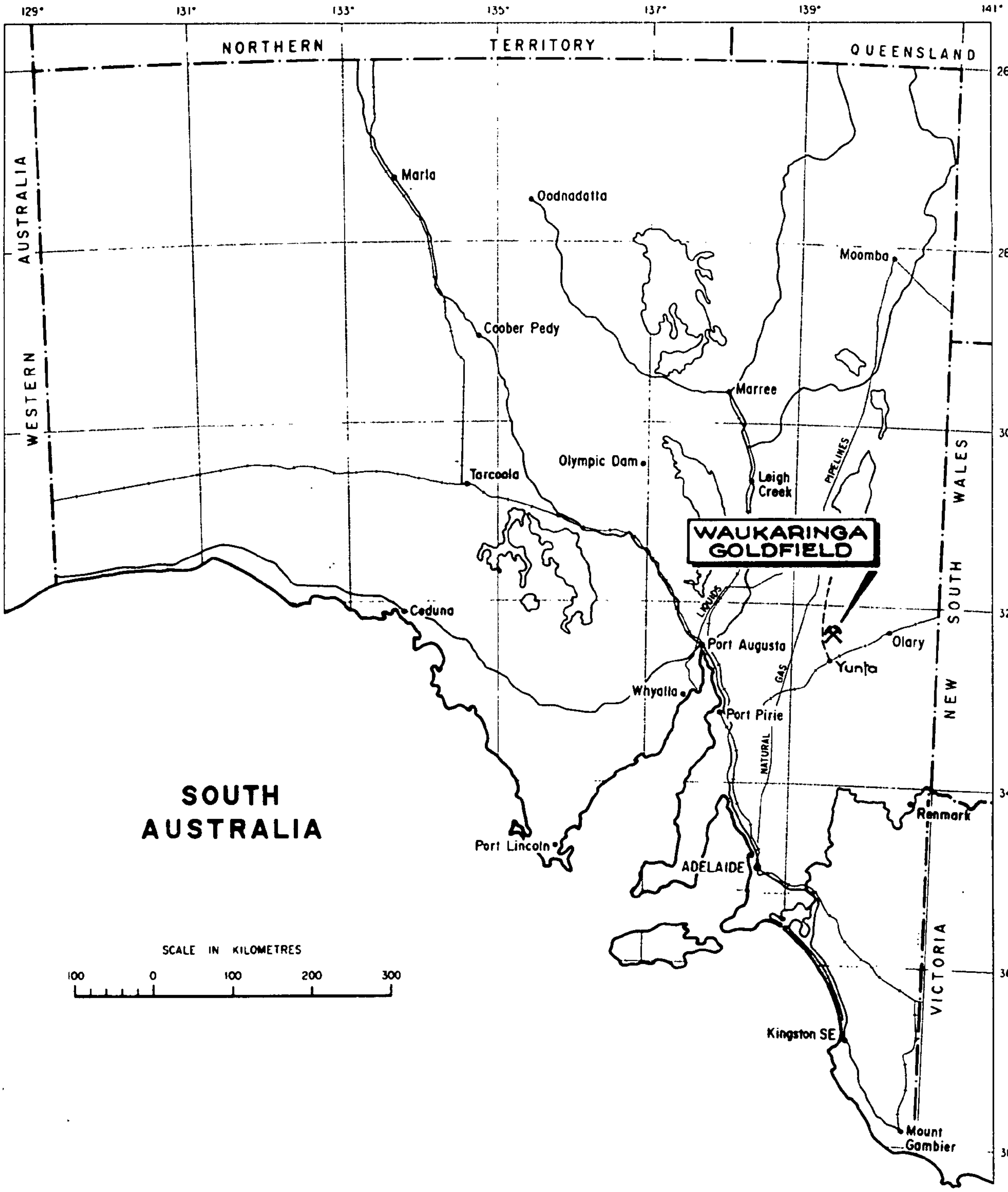


Figure......1

DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

WAUKARINGA GOLDFIELD

LOCALITY PLAN

COMPILED		3·12·85
I.J.T.	C.D.O.	DATE
DRAWN	SCALE	
M.R.		
DATE	PLAN NUMBER	
Sept. '84	S 17927	
CHECKED		

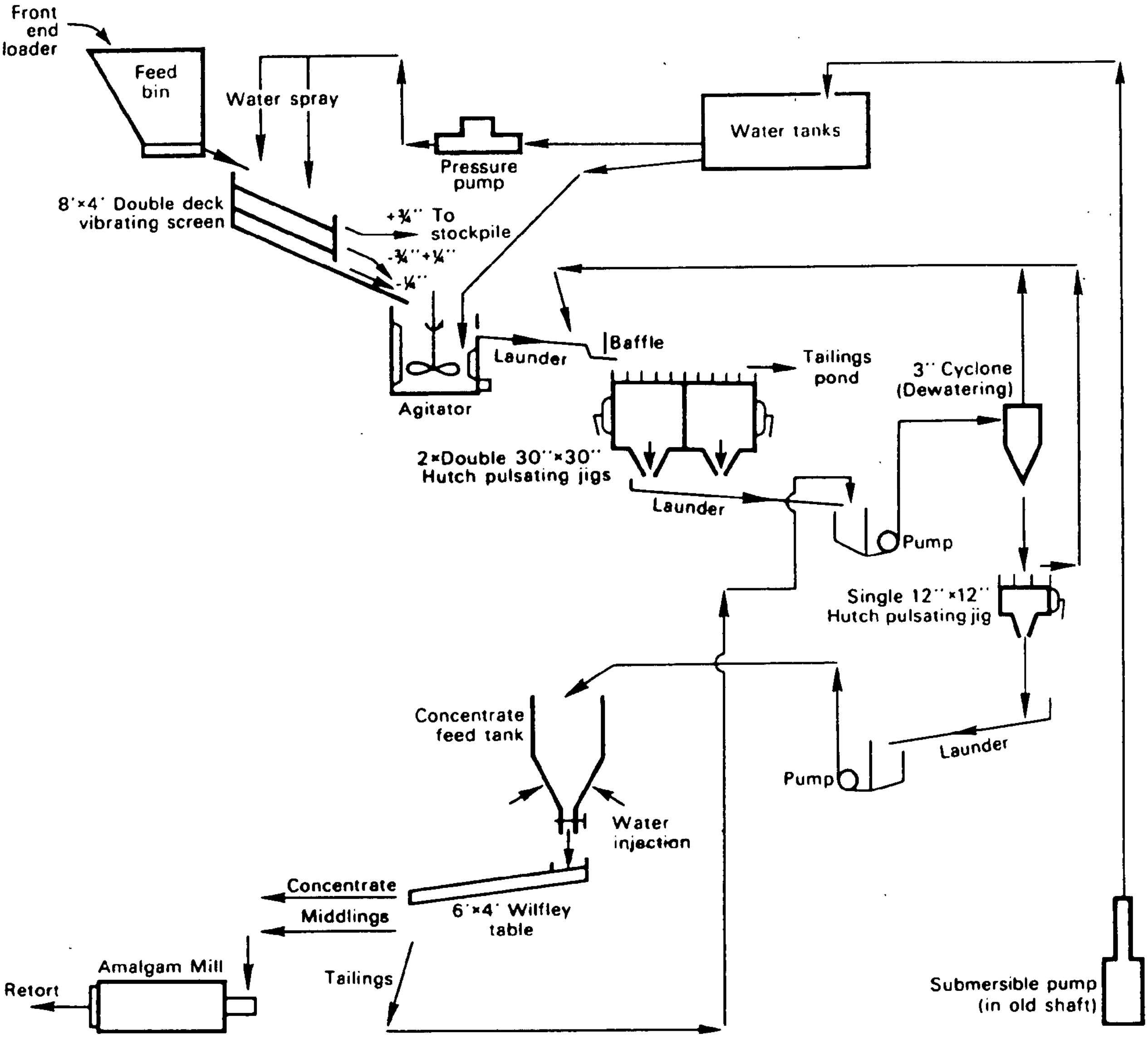


Figure..... 2

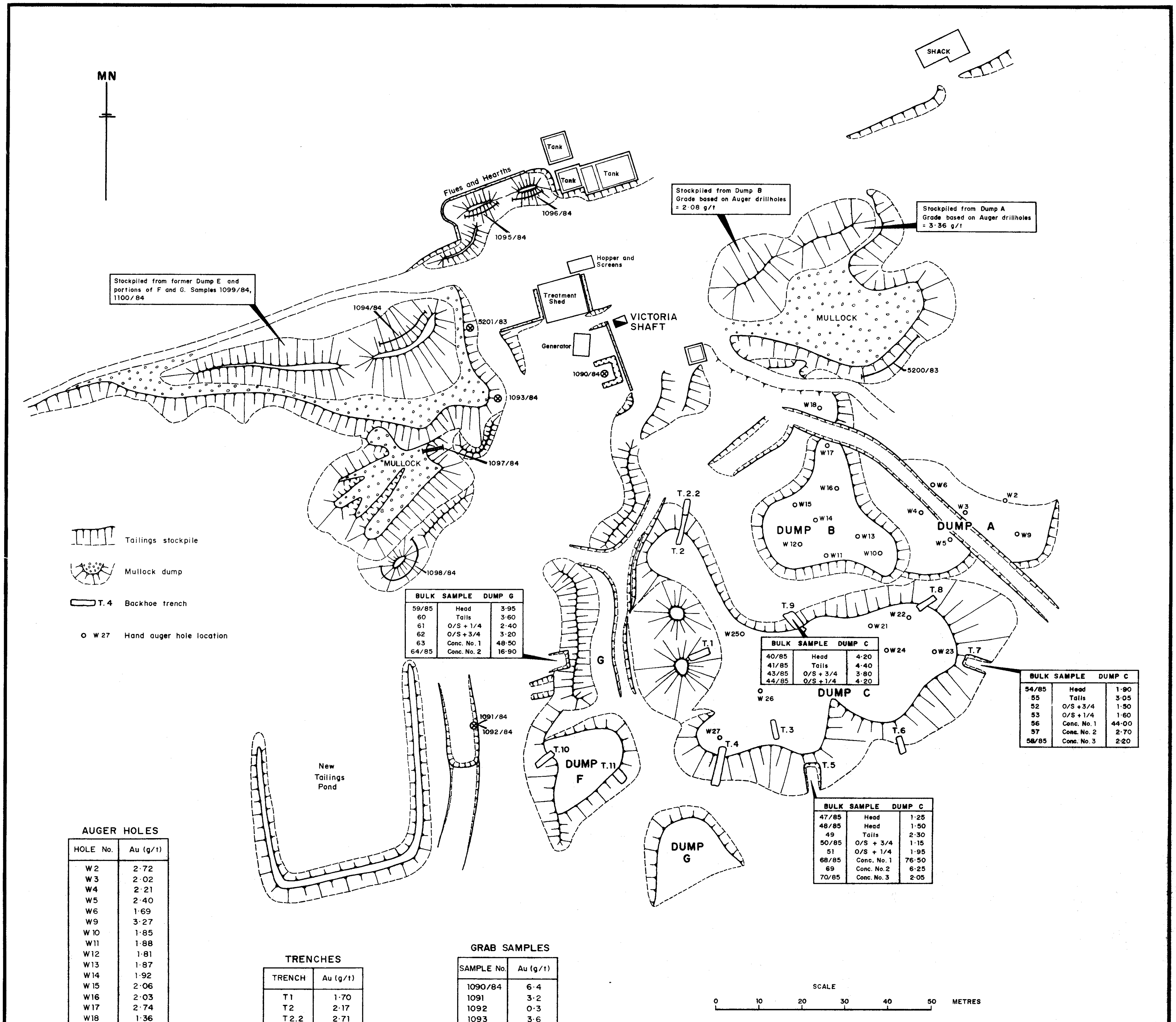
DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

WAUKARINGA GOLDFIELD

GOLD CONCENTRATION PLANT

SCHEMATIC FLOW DIAGRAM

COMPILED I. J. T.	5·12·86	C D O DATE
DRAWN P.A.	SCALE —	
DATE Nov. '84	PLAN NUMBER S 17930	
CHECKED		



BULK SAMPLE DUMP G		
59/85	Head	3·95
60	Tails	3·60
61	O/S + 1/4	2·40
62	O/S + 3/4	3·20
63	Conc. No. 1	48·50
64/85	Conc. No. 2	16·90

BULK SAMPLE DUMP C		
40/85	Head	4·20
41/85	Tails	4·40
43/85	O/S + 3/4	3·80
44/85	O/S + 1/4	4·20

BULK SAMPLE DUMP C		
54/85	Head	1·90
55	Tails	3·05
52	O/S + 3/4	1·50
53	O/S + 1/4	1·60
56	Conc. No. 1	44·00
57	Conc. No. 2	2·70
58/85	Conc. No. 3	2·20

BULK SAMPLE DUMP C		
47/85	Head	1·25
48/85	Head	1·50
49	Tails	2·30
50/85	O/S + 3/4	1·15
51	O/S + 1/4	1·95
68/85	Conc. No. 1	76·50
69	Conc. No. 2	6·25
70/85	Conc. No. 3	2·05

AUGER HOLES

HOLE No.	Au (g/t)
W2	2·72
W3	2·02
W4	2·21
W5	2·40
W6	1·69
W9	3·27
W10	1·85
W11	1·88
W12	1·81
W13	1·87
W14	1·92
W15	2·06
W16	2·03
W17	2·74
W18	1·36
W19	1·92
W20	2·52
W21	1·63
W22	1·85
W23	1·40
W24	1·64
W25	2·06
W26	2·12
W27	1·30

TRENCHES

TRENCH	Au (g/t)
T1	1·70
T2	2·17
T2.2	2·71
T3	N/S
T4	1·17
T5	0·78
T6	1·71
T7	1·74
T8	1·89
T9	2·05
T10	3·83
T11	5·00

GRAB SAMPLES

SAMPLE No.	Au (g/t)
1090/84	6·4
1091	3·2
1092	0·3
1093	3·6
1094	4·5
1095/84	1·2
1096	3·4
1097	17·3
1098	7·1
1099	2·5
1100/84	4·1
5200/83	0·22
5201/83	0·10

Fig. 3

DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

WAUKARINGA GOLDFIELD—GOLD TAILINGS

TAILINGS STOCKPILES MODIFIED AFTER MORRIS, 1983
BULK SAMPLING AND ADDITIONAL DUMP SAMPLES

COMPILED R. H.	C.D.O. 7·2·86 DATE
DRAWN M. B.	SCALE 1 : 500
DATE Jan '86	PLAN NUMBER 86 – 2
CHECKED	

6669

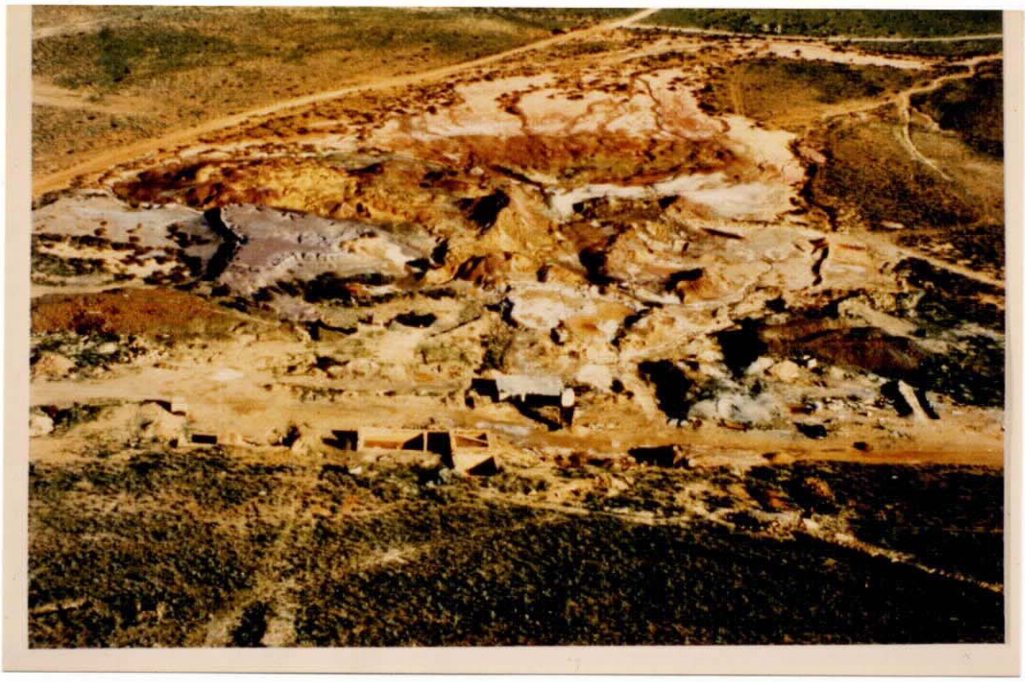


Plate 1. View to south-southeast of tailings stockpiles at Waukaringa. Note flue to chimney stack in lower right corner, 1983.
(photo No. 34080)

Plate 2. Plant shed with hopper and screens.
(photo No. 34490)

Plate 3. Agitator unit fed with material from the screens behind.
(photo No. 34491)

Plate 4. Two Double (0.76m x 0.76m) Hutch Pulsating Jigs.
(photo No. 34492)

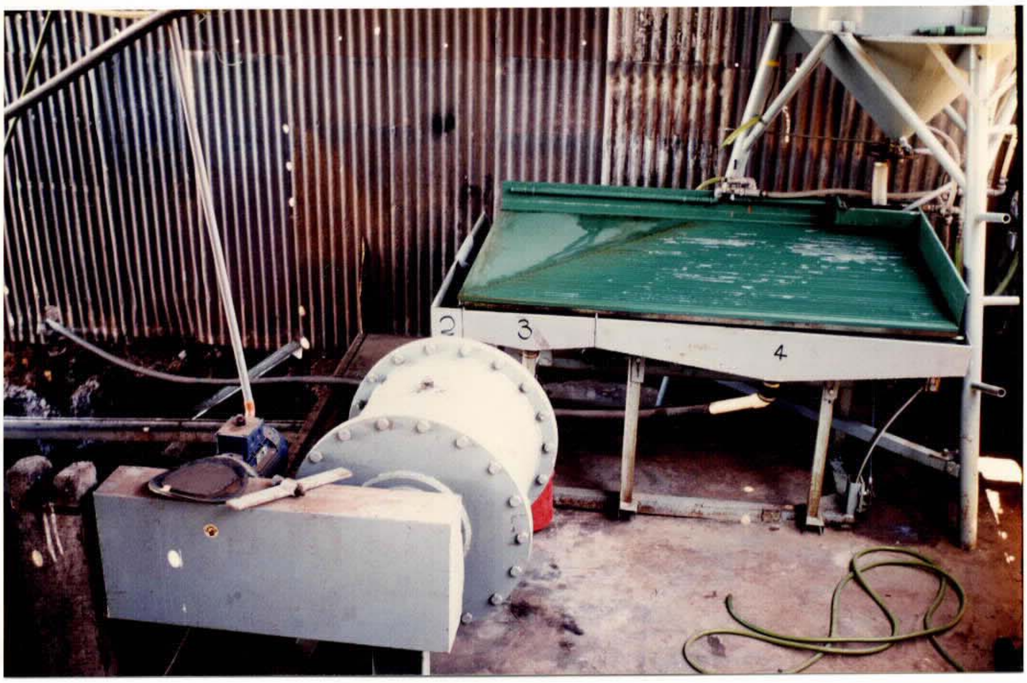


Plate 5. Wilfley Table and Amalgam Barrel.
(photo No. 34493)

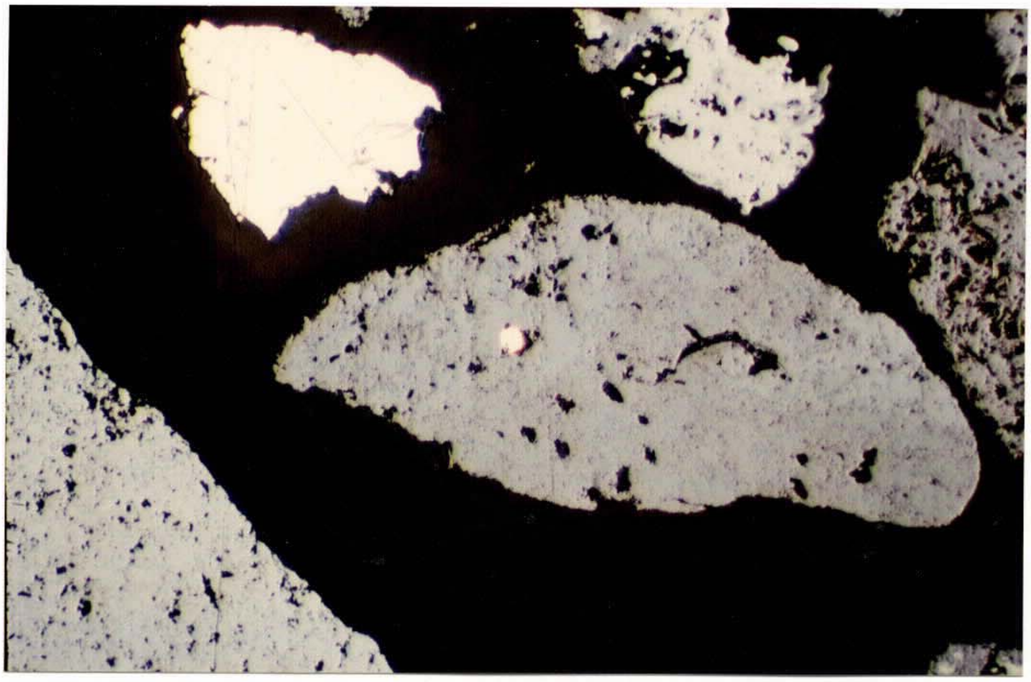


Plate 6. 10 m gold grain in goethite. Field length 0.5mm
Polished Sect. 33679 Sample A 2628/84.
(photo No. 34494)

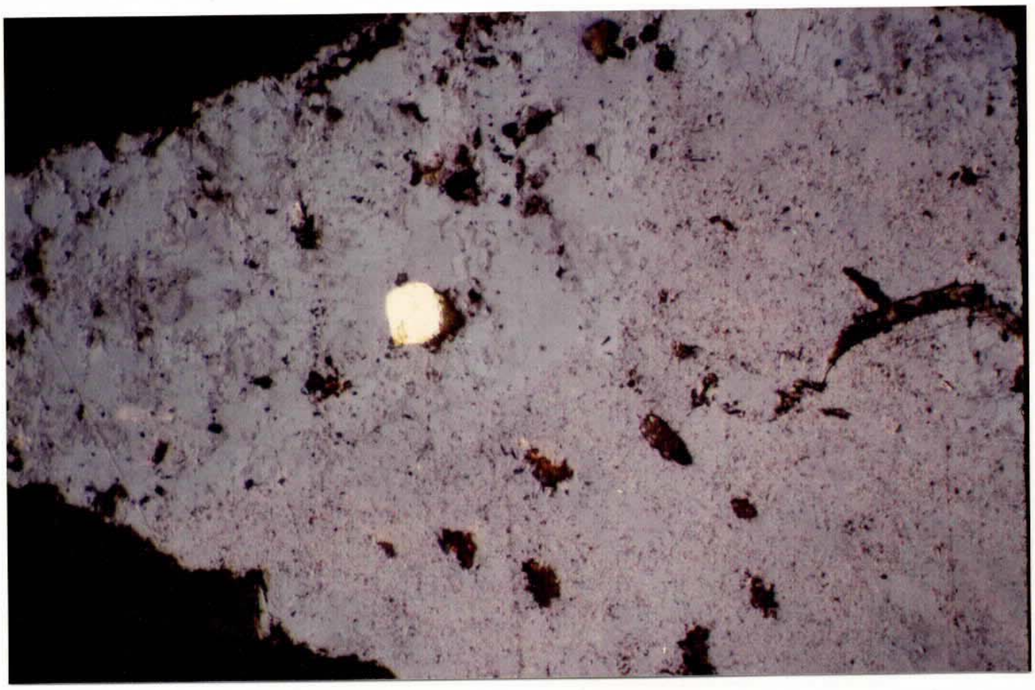


Plate 7. 10 m gold grain in goethite. Field length 0.5mm
Polished Sect. 33679 Sample A 2628/84.
(photo No.34496)

Plate 8. 4 m gold grain in goethite. Field length 0.5mm
Polished Sect. 33679 Sample A 2628/84.
(photo No. 34496)

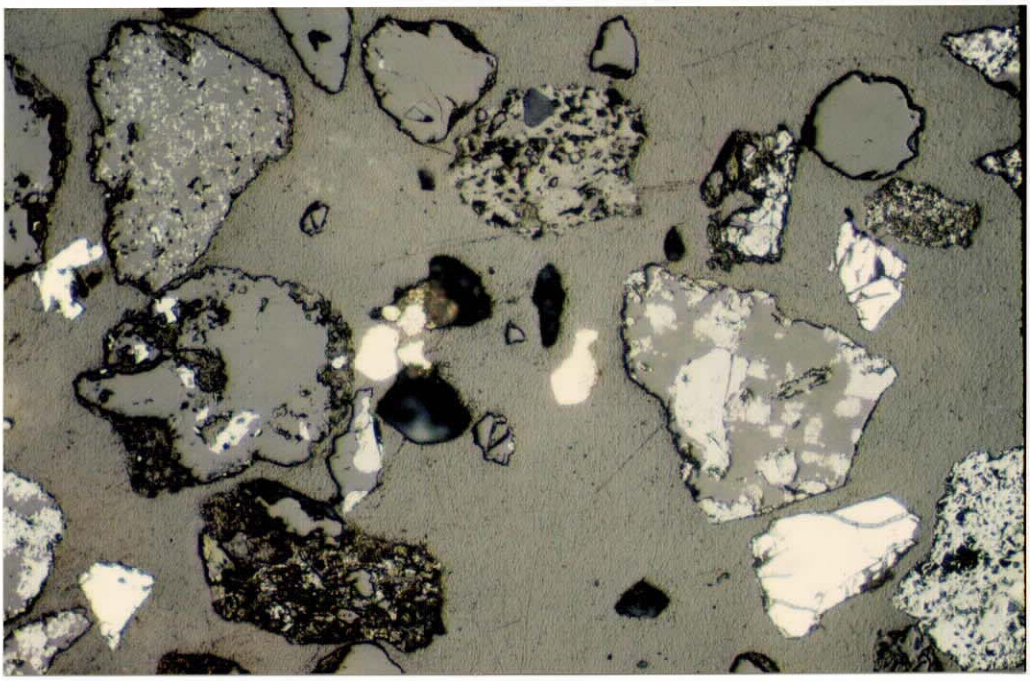


Plate 9. Liberated native gold grains. Field length 0.5mm
Polished Sect. 33680 Sample A 2628/84.
(photo No. 34498)

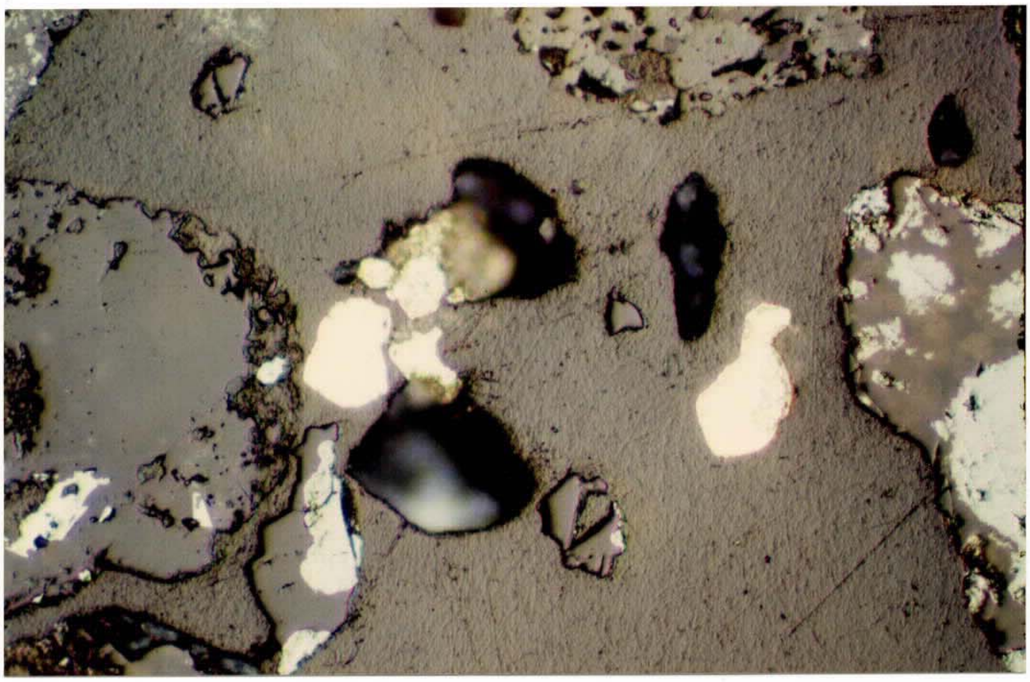


Plate 10. Liberated native gold grains. Field length 0.5mm
Polished Sect. 33680 Sample A 2628/84.
(photo No. 34498)

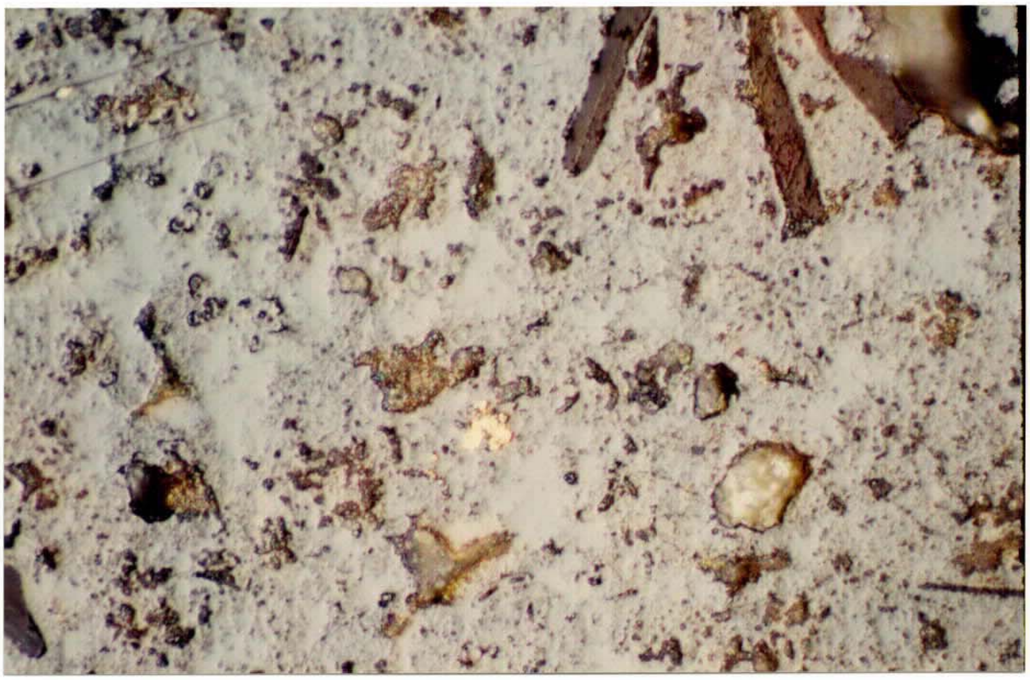


Plate 11. 10 m gold grain in porous goethite. Field length 0.2mm Polished Sect. 33648 Sample A 2531/84 - 25 + 52#.
(photo No. 34499)

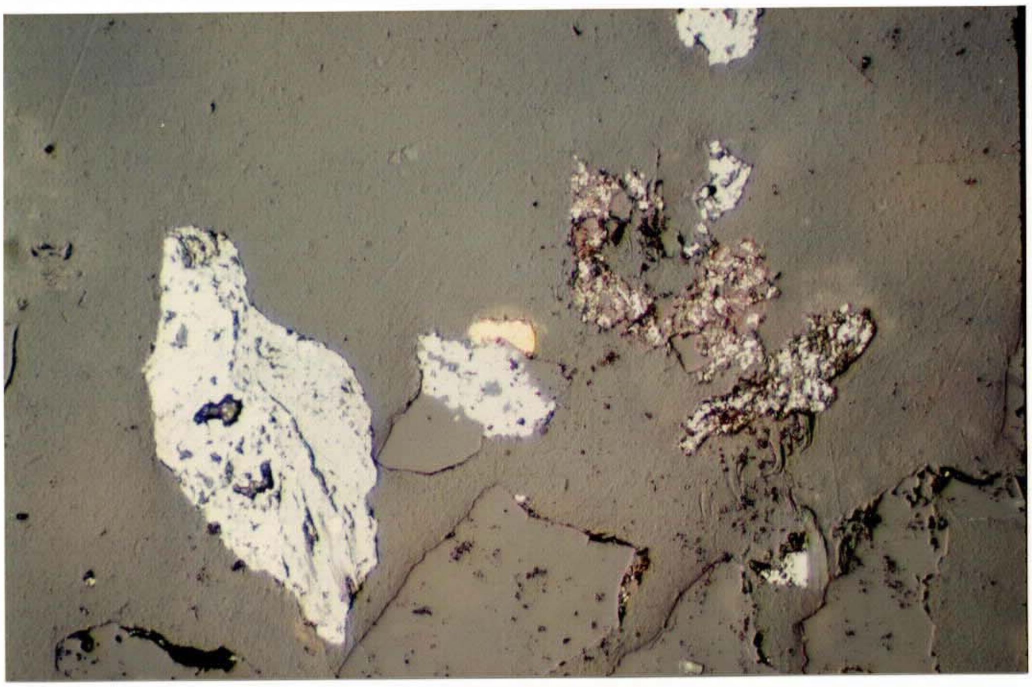


Plate 12. 10 x 30 m gold grain intergrown with goethite and non opaque gangue. Field length 0.5mm. Polished Sect. 33650 Sample A 2631/84 - 100 + 200#.
(photo No. 34500)

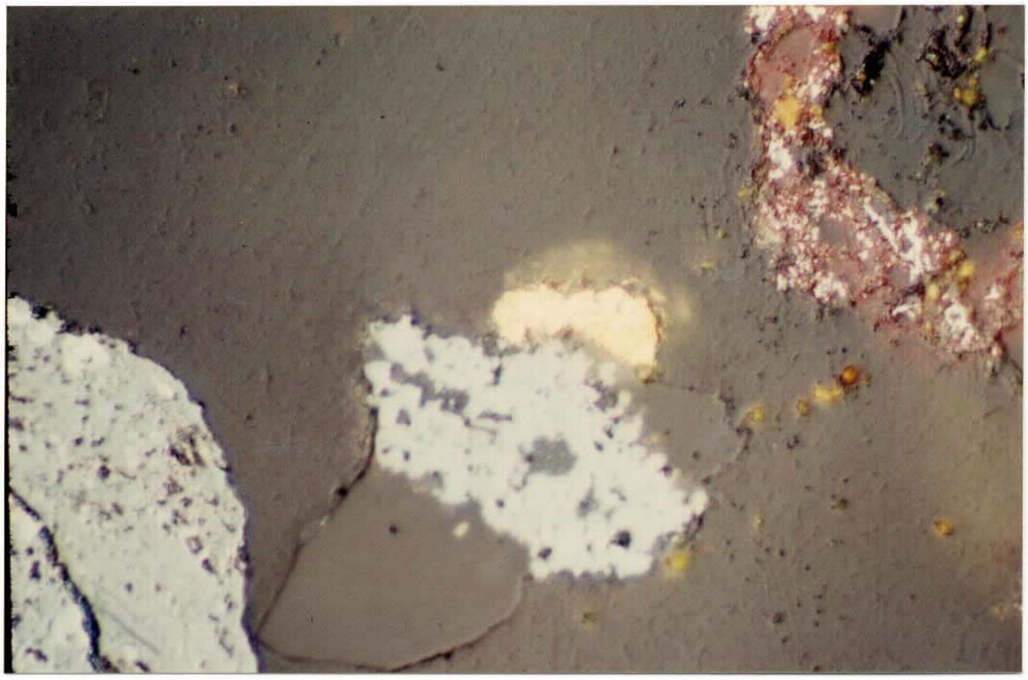


Plate 13. 10 x 30 m gold grain intergrown with goethite and non opaque gangue. Field Length 0.2mm Polished Sect. 33650 Sample A 2631/84 - 100 + 200#. (photo No. 34501)

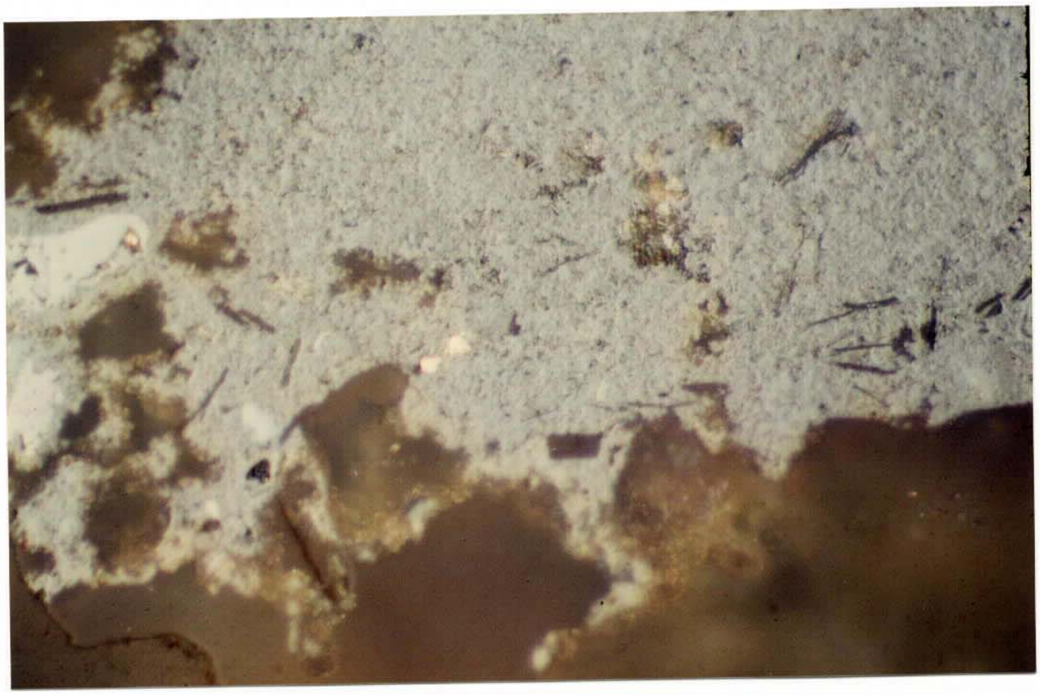


Plate 14. Close up of two gold inclusions. Field length 0.2mm. Polished sect. 33681 Sample A 2631/84 - 25 + 52# 3.3 non-mags. (photo No. 34502)