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DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

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APPRAISAL OF GOLD TAILINGS-DELORAINE GOLDFIELD

GEOLOGICAL SURVEY

By

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DME No.: 604/80

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DEPARTMENT OF MINES AND ENERGY
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APPRAISAL OF GOLD TAILINGS-DELORAINE GOLDFIELD

ABSTRACT

Between 1912 and 1939, 44 900 tonnes of auriferous ore with a grade of 18.6 g/t gold were treated on the Deloraine Goldfield, 6.4 km north-northeast of Kersbrook, and the tailings stored on site. There have been several attempts to treat the tailings in the past but an indicated 35 400 tonnes containing 3.62 g/t gold and 0.23% copper remain. Cyanidation tests indicate that 101 600 g (±42 400 g) of the 128 000 g (±52 100 g)of contained gold can be recovered economically by simple cyanidation of the tailings, without further treatment. The goldfield lies within the water catchment area of the South Para Reservoir and cyanidation will not be approved at this site.

INTRODUCTION

The main mines and ore treatment areas of the Deloraine Goldfield are located on Pt. Sections 2, 1548 and 1602, hundred Para Wirra county Adelaide, about 6.4 km north-northeast of Kersbrook and 1.6 km east of the Kersbrook to Williamstown road (Figs. 1 and 2). Most of the gold ore raised from the mines between 1909 and 1940 was treated on the field, consequently sizeable tailings dumps remain (Plate 1).

In March 1980, Deloraine Gold Mines N.L. pegged Mineral Claims 1228 and 1229, with the view to extracting the last of the gold from the tailings by cyanidation. The claimholders submitted a bulk sample of the tailings from several hand auger holes to Australian Mineral Development Laboratories (AMDEL) to determine the feasibility of treating the tailings by cyanidation. Under the assistance to industry scheme, the S. Aust. Dept. of Mines and Energy undertook to determine the quantity and grade of the auriferous tailings.

Plate 1: View looking south, scattered tailings in foreground, main tailings dump in background, creek on left and pine forest on right (Sept. 1980) (Slide No. 15837).

In September 1980, the dump was surveyed by stadia theodolite and 16 hand auger holes, up to 5.5 m deep, were drilled by the author and S.J. Ewen, M.W. Flintoft and B.W. Atterton (Field Assistants). Fifty samples, collected over one metre intervals, and twenty-five duplicates were analysed by AMDEL for gold by fire assay, and copper by atomic absorption spectroscopy.

The geology and the mine workings are described in S. Aust. Dept. of Mines Mining Review Nos. 10-73 and summarised in Robertson (1976).

TENURE

On 7th March 1980, Mineral Claims 1228 and 1229, of 19.44ha and 11.53 ha respectively, were registered by R.B. Scott, B.V. White and C.P.T. Woods. On 22nd September 1980, the claim holders were granted Exploration Licence 729 covering approximately 28 km^2 surrounding the claims under the name of Deloraine Gold Mines N.L. for a term of 6 months (Fig. 1).

The Mineral Claims cover part of sections 1602 and 1548, hundred of Para Wirra, which is freehold land owned by Deloraine Gold Mines N.L. and part of section 2, hundred of Para Wirra, which is freehold land owned by D.R. and T.L. Wilkin. The tailings dumps and main mine workings lie on section 1602 straddling a tributary of Malcolm Creek which drains northwards into the South Para Reservoir, 5 km to the north (Fig. 2). An application for Mineral Leases over the two claims was lodged on June 1980. The claims are within the District Council of Gumeracha, part of the outer Metropolitan Planning Area and within the South Para Reservoir catchment area.

The Exploration Licence gives the operators the right to explore within the defined area. Should the operators wish to commence mining outside Mineral Claims 1228 and 1229 then other mineral claims must be pegged.

GEOLOGICAL SETTING

The Deloraine Mines are located in Burra Group sediments of Adelaidean Age that dip steeply eastward and strike approximately north-south (Robertson, 1976). On ADELAIDE (Thomson, 1969) country rock comprises of an interbedded sequence of siltstone, shale and quartzite, probably Saddleworth Formation, which are faulted immediately to the west against feldspathic quartzite and siltstone of Undalya Quartzite (Fig. 1).

The workings extend intermittently for about 5 km and the gold occurs in quartz veins, 30 cms to 1.5 m wide, infilling fissures and crush zones that parallel the bedding of the country rock with trend north-south and easterly dip of between 40^{o} and 65^{o}. Associated minerals include covellite, chalcocite, native copper and limonite plus minor pyrite and chalcopyrite below about 90 m. Hydrothermal alteration, shown by sericitised wall rocks, around the quartz veins has been described by Ward (1913). Secondary copper minerals indicate that supergene enrichment has taken place and the gold grade becomes low and irregular below about 90 m.

GEOLOGY OF TAILINGS DUMP

The tailings consist of interbedded (about 1 cm thick) fine silty sand and silty clay (Plate 2) containing mostly quartz and muscovite with minor amounts of pyrite, chalcopyrite, covellite, chalcocite, malachite, goethite, hematite, rutile, siderite, dolomite, barite and native gold.

The dump is divided into three groups (Fig. 3):-

- south, main tailings dump about 6 000 m^2 in area and up to 5.5 m deep (plate 3).

Plate 2: Interbedded fine silty sand and silty clay in eastern face of main tailings dump (Sept. 1980) (Slide No. 15838).

Plate 3: View looking southwest of main tailings dump (Sept. 1980) (Slide No. 15389)

- central, about 10 000 m^2 in area of scattered tailings up to 1.7 m deep.
- north, slimes dump about 770 m^2 in area and 1.7 m deep.

HISTORY OF MINING AND TREATMENT

S. Aust. Dept. of Mines and Energy records that about 44 900 tonnes of ore yielded 829 000 g (26 650 oz.) of gold after treatment by private batteries on the goldfield between 1912 and 1939. The present tailings dumps were produced during this period. A further 114 000 g (3 670 oz) of gold were recovered from 4 500 tonnes of ore treated at the Mount Torrens Government Battery between 1909 and 1940. Production is summarised in Table 1 from yearly production figures in Appendix A.

TABLE 1

Summary of Production - Deloraine Goldfield

Treatment Site	Ore (tonnes)	Gold Recovered (g)	Gold Recovered (oz)
Deloraine Batteries	44 900	829 000	26 650
Mount Torrens Battery	4 500	114 000	3 670
TOTAL	49 400	943 000	30 320

The following five private stamp batteries operated on the Deloraine Goldfield between 1912 and 1939:-

1912-1920(?)	-	5 head
1915-1920(?)	-	5 head
1932 - ?	-	2 head
1933 - ?	-	4 head
1937-1939	-	10 head

In general, the tailings were passed over a concentrating table. The concentrates, containing up to 402.4 g/t (12.94 oz/t) gold and 11.47% copper, were sent to Port Kembla (N.S.W.) for treatment. The tailings containing 3-5.5 dwts (4.6-8.5 gms) gold per tonne and 1.5% copper were stockpiled at Deloraine (Ward, 1913).

The high gold content has encouraged several attempts to retreat the tailings. However, the S. Aust. Dept. of Mines and Energy records as listed hereafter do not document these attempts in detail.

Ward (1913) -	2 000 tons (2 032 tonnes) of tailings stacked, containing 3-5.5 dwts (4.6-8.5gms) gold per tonne and 1.5% copper.
Pearson (1924) -	Company erecting cyanide plant to treat an estimated 15 000 tons (15 240 tonnes) of tailings containing 3-5 dwts (4.6 - 7.7 gms) gold per tonne. Plant expected to treat 40 tonne per day. No record of plant being completed.
Winton (1930a) -	Tailings stockpiled but copper content too high for cyanidation.
Winton (1930b) -	Plant being erected to treat tailings by flotation.
Winton (1933) -	Flotation plant nearing completion.
Ritchie (1936) -	Flotation plant working.
Armstrong (1936) -	Cyanidation plant erected to test tailings. Five parcels of one tonne treated - no results reported.
Ritchie (1937) -	Flotation plant completed but no regular production since completion.

Abell and Gartrell (1937) - Tailings contain 3.1 dwts (4.8 gms) gold per tonne and 0.29% copper. Flotation recovers 2.7 dwts (4.2 gms) gold per tonne; straking up to 1.5 dwts (2.3 gms) gold per tonne; and amalgamation up to 2.2 dwts (3.4 gms) per tonne. Best recoveries obtained with fine grinding. Presence of copper made cyanidation unsuitable due to high consumption of cyanide.

Blaskett and Gartrell (1940a) - Tailings contain 1.9 dwts (2.9 gms) gold per tonne and 0.29% Copper. Grinding and straking tests give only 50% recovery.

Blaskett and Gartrell (1940b) - Tailings contain 5.3 dwts (8.2 gms) gold per tonne. Straking and centrifigal concentration tests give recoveries of 50%. Flotation tests give 70% recovery. Gold is fine (less than 5 microns) and mainly contained in pyrite and iron oxides. Any treatment should include grinding.

McEwin (1940) - Deloraine mines closed August 1940, plant erected to treat tailings operated for one month and closed in January 1941. No mention of type of treatment plant or quantity of tailings treated.

AMDEL APPRAISAL OF TAILINGS

Deloraine Gold Mines N.L. submitted a bulk sample of the tailings obtained from several hand auger holes, to AMDEL to determine suitability for gold extraction. The auger samples were taken from the main tailings dump, the scattered tailings dumps and the slimes dump.

AMDEL carried out the following:-

- ore-dressing mineralogy to assess the feasibility of gravity separation.
- cyanide leaching tests, with and without acid pre-leaching, with and without further grinding.
- cost estimate of leaching plant.

Conclusions from their detailed report, included as Appendix B are:

- the tailings contain 2.5 dwts (3.9 g) per tonne gold and 0.21% copper and consist of quartz and muscovite with minor amounts of magnesian siderite, dolomite, barite, pyrite, chalcopyrite, covellite, chalcocite, malachite, goethite, hematite, rutile and native gold.
- about 70% of the tailings are - 150um and about 30% are -53 um particle size.
- native gold is mainly less than 25 um in diameter and occurs as inclusions in, or intergrowths with pyrite, chalcopyrite and goethite. Some may also be locked in silicate.
- the distribution and small particle size of the gold plus the fineness of the tailings indicates that gravity concentration would give low (less than 50%) gold recovery.

- cyanidation before grinding gives about 79% gold recovery and after grinding about 94% gold recovery Acid leaching prior to cyanidation increases gold recovery to 86% for tailings and 96% for tailings after grinding and reduces the cyanide consumption from about 3.2 kg/t to 1.0 kg/t.
- the feasibility of economic extraction of gold by

  (a) simple cyanidation

  (b) cyanidation plus solution recycling to minimise environmental pollution

and (c) cyanidation after grinding, have been examined and the results are as follows:

Process	(a) Basic	(b) Solution Recycle	(c) Grinding
	$	$	$
Total Capital investment	300 600	535 000	524 000
Total manufacturing costs:			
a) on site	565 700	827 300	802 300
b) with ore transport	597 200	858 800	833 800
value of gold extracted*	849 600	849 600	1 008 900
Profit before tax:			
a) on site	283 900	22 300	206 600
b) with ore transport	252 400	-9 200	175 100

*(Value of Au taken as $17.70/g ($550/oz)

RESERVES

Reserves in table 2, were calculated from cross sections (Fig. 4) and the gold and copper content were determined using weighted averages. These reserves are classified as indicated.

Reserve and grade calculations are detailed in Appendix D based on the plan of tailings dump on figure 3 and cross sections on figure 4. Geological logs of hand auger holes with gold and copper assays are presented in Appendix C.

The main tailings dump appears to be from the last major period of production between 1937 and 1939, during which the tailings were passed over concentrating tables before dumping. This probably explains the significantly lower grade of this portion of the dump (Table 2).

COST ESTIMATE

AMDEL calculated a cost estimate of gold extraction for 15 000t of tailings with an average grade of 3.9 g/t gold (Appendix B). Table 3 presents cost estimates with AMDEL calculations modified to take into account the larger tonnage of tailings determined from the hand auger drilling reported herein. Cost estimates are based on tailings being transported to Palmer, about 45 km east of Deloraine for treatment. As in the AMDEL report, allowance for the building of a tailings dam, should this be required, is not included in cost estimates.

TABLE 3

COST ESTIMATES (in dollars)

Process

	Basic cyanidation	Cyanidation +soln. recycling	Grinding before cyanidation
MAIN TAILINGS			
Total manufacturing costs including ore transport	979 100	1409 600	1 367 800
Value of gold extracted*	934 100	934 100	1 098 200
Profit before tax	-45 000	-475 500	-269 600
SCATTERED TAILINGS			
Total manufacturing costs including ore transport	342 300	492 800	478 200
Value of gold extracted	824 100	824 100	968 800
Profit before tax	481 800	331 300	490 600
SLIMES			
Total manufacturing costs including ore transport	87 600	126 100	122 300
Value of gold extracted	202 600	202 600	239 500
Profit before tax	115 000	76 500	117 200
TOTAL TAILINGS AND SLIMES			
Total manufacturing costs including ore transport	1 409 000	2 028 500	1 968 300
Value of gold extracted	1 960 800	1 960 800	2 306 400
Profit before tax	551 800	- 67 700	338 100

*(Based on gold price of $19.30/g ($600/oz))

CONCLUSIONS

- A total of 35 400 tonnes of tailings, containing 3.62 g/t gold and 0.23% copper are indicated on the surface at the Deloraine Goldfield.
- The tailings are suitable for gold extraction by cyanidation.
- The tailings contain about 128 000 g (±52 100 g) of gold.
- Simple cyanidation is expected to recover 101 600 g (±42 400 g) economically.
- The tailings are located within the water catchment area of the South Para Reservoir and cyanidation on site will not be approved.

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APPENDIX A
Production

1. Ore Treated on Field
2. Deloraine Ore Treated At Mount Torrens Battery
3. Total Production of Deloraine Goldfield.

TABLE 2
Reserve Estimates

	Volume (m^3)	Tonnage	Cu (%)	Cu Content (+)	Au (g/t)	Au Content (g)	Recoverable Au at 80% (g)	[2]Value ($)	Recoverable Au at 94% (g)	[2]Value ($)
Main Tailings	14 200	24 600	0.23	57	2.46 (±1.70)	60 500 (±41 800)	48 400 (±33 400)	934 100 (±44 600)	56 900 (±39 300)	1 098 200 (±758 500)
Scattered Tailings	5 000	8 600	0.21	18	6.21 (±1.30)	53 400 (±11 200)	42 700 (±9 000)	824 100 (±173 700)	50 200 (±10 500)	968 900 (±202 700)
Slimes[3]	1 300	2 200	0.26	6	5.99	13 200	10 500	202 600	12 400	239 300
TOTAL	20 500	35 400	0.23	81	3.62	128 000 (±52 100)	101 600 (±42 400)	1 960 800 (±818 300)	119 500 (±49 800)	2 306 400 (±961 200)

[1]Density of 1.73 is assumed
[2]Based on gold price of $19.3/g ($600/oz)
[3]Insufficient data to calculate range for slimes, no variation assumed.

1) ORE TREATED ON FIELD

Mining Review	Period	Ore (tonnes)	Gold Recovered (g)	(oz)
17	6mths to Dec.1912	772.2	17 313	556.7
18	" " June 1913	914.4	15 827	508.9
19	" " Dec. 1913	1 011.9	26 954	866.7
20	" " June 1914	1 071.9	27 253	876.3
21	" " Dec.1914	1 168.4	23 014	740.0
22	" " June 1915	1 264.9	17 696	569.0
23	" " Dec. 1915	2 275.8	37 165	1195.0
24	" " June 1916	2 279.9	39 932	1284.0
25	" " Dec. 1916	2 319.5	43 944	1413.0
26	" " June 1917	2 564.4	52 310	1682.0
27	" " Dec. 1917	2 602.0	58 436	1879.0
28	" " June 1918	2 685.3	53 554	1722.0
29	" " Dec. 1918	2 771.3	43 229	1390.0
30	" " June 1919	2 698.5	47 303	1521.0
31	" " Dec. 1919	2 211.8	33 122	1065.0
32	" " June 1920	767.1	18 722	602.0
33	" " Dec. 1920	238.8	5 847	188.0
65	12mths to March 1937	4 673.6	62 915	2023.0
67	" " March 1938	4 470.4	109 130	3509
			7 029*	226.0*
69	" " March 1939	3 722.6	44 100	1418.0
			10 574*	340.0*
70	March to June 1939	692.9	12 057	387.7
71	June to Dec. 1939	1 724.2	20 246	651.0
		44901.8	1 182*	651.0*
		approx 44 900	828 854	26 651.3
		approx.	829 000	26650

(*Concentrates from tailings treated at Pt. Kembla)

2) DELORAINE ORE TREATED AT MOUNT TORRENS BATTERY
(from Mining Review 73)

Mine	Ore(tonnes)	Gold Recovered (g)	(oz)
Birthday Gift	688.8	9 557	307.3
New Deloraine	1 878.2	67 751	2 178.5
Deloraine North(Blocks)	1 272.9	20 728	666.5
Deloraine Queen	31.1	196	6.3
Prairie Deloraine(South)	217.6	8 864	285.0
Uraparinga(Clark's Find)	57.9	1 306	42.0
Pearce's Find	18.0	190	6.1
Sheoak Ridge	19.8	600	19.3
Easter Gift	300.2	5 007	161
TOTAL	4 484.5	114 199	3 672.0
approx.	4 500	appr 114 000	3 670

3) TOTAL PRODUCTION OF DELORAINE GOLDFIELD

	Ore(tonne)	Gold Recovered (g)	(oz)
Treated On Field	44 900	829 000	26 650
Treated at Mount Torrens	4 500	114 000	3 670
TOTAL	49 400	943 000	30 320

APPENDIX B

AMDEL Report No. 1343, Feasibility of Extracting Gold From Deloraine Tailing Dumps by P. Capps and K.J. Henley

R.B. 80/75
DME 394/77.

1/1/246

June, 1980

S.A. DEPARTMENT OF MINES AND ENERGY

AMDEL Report

No. 1343

FEASIBILITY OF EXTRACTING GOLD FROM DELORAINE TAILING DUMPS

by

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C O N T E N T S

SUMMARY

Background

Following discussions between S.A.D.M.E., AMDEL and Mr C. Woods of Chetwood (Australia) Pty Limited, S.A.D.M.E. authorized AMDEL to undertake a preliminary evaluation of the economic feasibility of setting up a small plant to extract gold from gold tailing dumps at Deloraine, near Williamstown, South Australia.

Summary of Work Done

A composite sample of tailing was supplied by Mr C. Woods. Ore-dressing mineralogy was done on the sample to assess the feasibility of gravity concentration. Cyanide leaching tests, with and without acid pre-leaching to remove cyanicides, and with and without further grinding, were done to assess the gold extraction and cyanide and lime consumptions.

On the basis of these tests, various alternative flowsheets were costed and their economics assessed.

Conclusions

(1) The tailing contains 3.9 ppm Au and 0.21% Cu and consists predominantly of quartz and muscovite with minor to trace amounts of magnesian siderite, dolomite, barite, pyrite, chalcopyrite, covellite, chalcocite, malachite, goethite, hematite, rutile and native gold. About 70% of the tailing is -150 μm and about 30% is -53 μm in particle size.

(2) Relatively little of the native gold is liberated, most observable native gold occurring as inclusions in, or intergrowths with, pyrite, chalcopyrite and goethite. However, some native gold may be locked in silicates. The grain size of the native gold is mainly less than 25 μm. The colour of the native gold indicates a silver content of less than 10%.

(3) The distribution of gold with specific gravity and particle size indicates that gravity concentration would give low (probably less than 50%) recoveries of gold and, although finer grinding would improve gold liberation, recoveries by gravity concentration would be unlikely to increase significantly because of the less efficient separation at the finer particle sizes.

(4) Cyanidation of the tailing gives high extraction of gold - about 79% on the as-received material and 94% on material ground to 78% -150 μm. Acid-leaching prior to cyanidation increases the gold extraction slightly (to 86% on as-received material and 96% on material ground to 77% -150 μm) and reduces the cyanide consumption from about 3.2 kg/t to 1.0 kg/t, presumably by removing cyanicides such as malachite.

(5) The feasibility of economic extraction of gold by (a) simple cyanidation, (b) cyanidation plus solution recycle (to minimize environmental pollution),

and (c) cyanidation after grinding have been examined, considering both treatment on site and treatment after transport to the Sedan/Cambrai area. Results are as follows:

Process	Basic $	Solution Recycle $	Grinding $
Total capital investment	300 600	535 000	524 000
Total manufacturing costs:			
On site	565 700	827 300	802 300
With ore transport	597 200	858 800	833 800
Value of Au extracted*	849 600	849 600	1 008 900
Profit before tax:			
On site	283 900	22 300	206 600
With ore transport	252 400	-9 200	175 100

*Value of Au taken as $17 700/kg ($550/oz.)

It is emphasized that these are preliminary cost estimates only and no allowance has been made for building a tailings dam, should this prove necessary.

Recommendations

This investigation has been of a preliminary nature only and before proceeding to plant operation it is recommended that further metallurgical studies be undertaken to determine:

(a) the required leaching time for optimum gold extraction,

(b) the adsorption characteristics of various carbons,

(c) the conditions necessary for satisfactory oxidation of the cyanide ion, and

(d) the settling characteristics of the tailings.

1. INTRODUCTION

Following discussions on assessment of gold tailing dumps at Deloraine near Williamstown with Messrs R. Wildy and G. Drew of the S.A. Department of Mines and Energy (S.A.D.M.E.), Mr C. Woods of Chetwood (Australia) Pty Limited, and Dr K.J. Henley of AMDEL in February 1980, a proposal was submitted by AMDEL to S.A.D.M.E. to evaluate the economic feasibility of setting up a plant to extract gold from the dumps.

Previous work on various samples from the dumps had indicated that the gold content was 3 to 5 ppm and the particle size distribution was about 95% minus 0.42 mm (36 mesh B.S.S.) with a variable proportion (40% to 90%) of minus 0.075 mm material. Only a small proportion of the gold appeared to be recoverable by gravity treatment but cyanidation had given extractions of about 80%; secondary copper minerals (~0.3% Cu) were present and caused high cyanide consumption. The dumps are estimated to contain about 15,000 tonnes of tailings.

The proposed programme of investigation was as follows:

(1) Obtain a representative composite sample of the dumps. It was understood that Mr Woods had auger drilled and sampled the dumps and a bulk sample had been deposited at AMDEL.

(2) Carry out a gold mineralogy/liberation study of the composite to determine how the gold occurs and assess the possibilities of gravity concentration.

(3) Carry out cyanidation testing of the composite to determine gold extraction, cyanide consumption and lime consumption. Testing would be carried out on material (a) as received, (b) after grinding, and (c) after leaching out the secondary copper minerals.

(4) On the basis of the results of (2) and (3), carry out a preliminary paper feasibility study to indicate likely costs for various alternative processes (e.g., cyanidation alone, cyanidation plus gravity concentration, gravity concentration alone).

If the results of (4) appeared promising it would then be necessary to consider pilot-scale extraction tests to prove up the process, but the latter was not included in the initial project.

Approval for the project was given on 10 May 1980 and this report gives the results of the project.

2. MATERIAL EXAMINED

The tailing sample investigated* was a thoroughly mixed composite of all material provided by Mr C. Woods (other than that previously removed for analysis) and was stated by Mr Woods to be reasonably representative of the dumps. AMDEL was not involved in the sampling programme.

The sample assayed 3.9 ppm gold and 0.21% copper and had the following size distribution:

Size Fraction (μm)	Wt, %
+500	2.7
-500+355	11.0
-355+300	5.1
-300+250	5.5
-250+150	6.0
-150+105	21.0
-105+53	18.0
-53+10	26.7
-10	4.0
Total	100.0

* The results presented in this report apply only to the sample submitted by the Client and described in the text. No guarantee either express or implied is given as to the applicability of the results to other samples.

3. ORE-DRESSING MINERALOGY

3.1 Procedure

The purpose of the ore-dressing mineralogical examination was to establish the feasibility of recovering gold from the tailing by gravity concentration.

A 1 kg aliquot of the tailing sample was wet screened and sized by sedimentation. Two separate riffled portions of each +10 μm size fraction were separated in heavy liquids of specific gravity 2.96 and 4.2, the +105 μm fractions statically and the -105+10 μm fractions centrifugally. The products from one portion were pulverized and analysed for gold and the 2.96-4.2 sp.gr. and >4.2 sp.gr. products of the other portion were prepared as polished sections (PS 22541-22544) and examined mineragraphically. Selected <2.96 and and 2.96-4.2 sp.gr. products were analysed by X-ray diffraction to identify the main minerals present.

3.2 Distribution of Gold with Specific Gravity and Particle Size

The distribution of gold with specific gravity and particle size is given in Table 1. In interpreting Table 1 it may be assumed that in commercial gravity concentration (e.g. jigging, tabling) >4.2 sp.gr. material would report into the concentrate, <2.96 sp.gr. material would report into the tailing and 2.96-4.2 sp.gr. material would report into the middling (or partly into the concentrate and partly into the tailing).

The results indicate that with decreasing particle size there is a progressive increase in the proportion of gold within each size which reports in the >4.2 sp.gr. product, from 8.2% in the +150 μm fraction to 68.2% in the -53+10 μm fraction, suggesting progressive liberation of gold-bearing minerals from light gangue. However, overall in the +10 μm material, which constitutes 96% of the weight of the sample, only 35.5% of the gold reports in the >4.2 sp.gr. products. Clearly, therefore, gravity concentration without further fine grinding would not give high recoveries of gold, particularly in view of the fine particle size of the material (where gravity concentration processes are less efficient than at coarse sizes).

3.3 Mineralogy of the Separation Products

The <2.96 sp.gr. products consist predominantly of quartz and muscovite with minor to trace proportions of feldspar (probably albite or microcline), ?calcite and ?dolomite.

The 2.96-4.2 sp.gr. products consist predominantly of magnesian siderite, chalcopyrite and goethite with minor to trace proportions of dolomite, malachite, muscovite and quartz. No attempt was made to locate native gold in these products but it may be assumed to be locked in one or more of the above minerals.

The >4.2 sp.gr. products consist predominantly of pyrite, barite and chalcopyrite with traces of covellite/chalcocite, goethite, hematite and rutile. The polished sections of the >4.2 sp.gr. products were scanned carefully at a magnification of ×100 to detect any native gold present. The following is a summary of the nature of the native gold observed:

Size Fraction (μm)	Native Gold: Size (μm)	Native Gold: Shape and Occurrence	
+210	12	Angular inclusion in pyrite	
	40	Rounded grain intergrown with chalcopyrite	same particle
	5	Elongate grain intergrown with pyrite	same particle
	2×60	Narrow fracture infilling in pyrite	
-210+105	5	Elongate inclusion in goethite after pyrite in composite pyrite/chalcopyrite/goethite particle	
	12	Rounded inclusion in pyrite	
-105+53	3×12 3×12	Elongate irregular inclusions in goethite	same particle
-300+10	3×10 3×20 3×25	Elongate irregular inclusions in goethite	same particle
	25	Equant liberated particle	

It can be seen from this tabulation that the only liberated native gold observed was in the -300+10 μm fraction and that the bulk of the native gold occurred as angular or rounded grains in pyrite or, less commonly, in chalcopyrite and goethite. One example was observed in native gold infilling a fracture in pyrite. The largest grain of native gold was about 40 μm in size but most of the grains were less than 25 μm in size. The native gold was a uniform yellow colour in polished section, suggesting a silver content of less than about 10%.

The presence of secondary copper minerals (malachite, covellite, chalcocite) suggests that consumption of cyanide during cyanidation may be high.

4. CYANIDE LEACHING

In order to determine the suitability of agitation cyanide leaching for treatment of the dump material, five sub-samples of approximately 300 g each were riffled from the head sample. Of these, two were ground to approximately 80% -150 μm and one to 75% -212 μm.

One 'as received' sample and one ground to 77% -150 μm were leached in sulphuric acid at pH1 for 6 hours to reduce the quantity of cyanide-consuming copper present. Acid additions are shown in Table 2. Washed residues from this acid pre-leach were then used for cyanide leaches (nos. 1 and 2).

The five dump samples (including the acid pre-leached residues) were agitation cyanide leached for 48 hours. Leach slurries consisted of approximately 300 g ore, 1000 ml solution at 0.05% NaCN and 0.025% CaO. Slurries were aerated continuously throughout the leach period. After 5, 23 and 29 hours, leach liquors were sampled for gold assay and titration to determine NaCN and CaO levels. Reagents were added if necessary to maintain the desired levels.

At the completion of 48 hours, slurries were filtered and the residues washed thoroughly. Residues and liquor samples were assayed for gold and extractions were calculated from the amount of gold in the residue and the total gold in solution. Results are shown in Table 3 and demonstrate the effects of acid pre-leaching, grind size and leach time.

Highest extractions of approximately 95% were obtained by leaching the finest ground samples (77% -150 μm) with acid pre-leaching reducing the consumption of NaCN from 3.2 to 1.1 kg/t. Leaching of 'as received' ore extracted 79% of the gold with slightly higher extraction (86%) being obtained from the acid leached 'as received' ore. Again, NaCN consumption was reduced to approximately one-third by acid leaching. Grinding of ore to 75% -212 μm reulted in an intermediate gold extraction of 88%.

Leaching time required was not shown accurately but indications are that in all cases gold extraction was completed in less than 24 hours.

5. PRELIMINARY COSTING OF LEACHING PLANT

5.1 Introduction

For such a small deposit as the Deloraine dump (15 000 t at 4 g/t Au) it is essential to keep treatment as simple as possible and hence the capital cost of plant to a minimum. On the basis of the mineralogical and laboratory cyanidation investigations, the most suitable process for treating this material, considering size and extraction characteristics, appears to be direct cyanidation.

However, several variations to the flow scheme are possible, depending on such factors as availability and cost of second-hand equipment, environmental restrictions and the validity of assumptions made regarding the process. These variations and influencing factors will be expanded upon in the following sections.

Acid pre-leaching will not be considered in detail. Laboratory work shows that acid leaching saves approximately 2 kg/t NaCN ($2.90/t) at a cost of $5.27/t for H_2SO_4. Even if the actual leach time required is reduced, the acid cost can at best only be similar to the saving in cost for NaCN. When the extra capital cost for leaching tanks and associated equipment is taken into consideration, acid pre-leaching does not appear to be worthy of consideration for this situation. Although acid consumption might be able to be reduced considerably further by recycle of solution, this in turn involves more equipment for solids/liquid separation and copper concentration, and hence more capital cost.

5.2 Basic Flowsheet

The basic flowsheet (Fig. 1) on which variations will be considered involves simply leaching dump material without any prior treatment (e.g. grinding). Ore is fed by front-end loader into two agitated leach tanks (36 tonnes per tank) and water added to make a 50% slurry. Reagents (NaCN, CaO) are added and the mixture agitated and aerated for 24 hours. Included in the leach slurry is 48 kg of granular activated carbon which adsorbs the gold from solution continuously. Use of the batch leaching system means that no sophisticated feeding equipment is required and that two instead of four leach tanks can be used.

At the end of the 24 hours the slurry is pumped over a vibrating screen fitted with water jets for washing. The carbon reports as screen oversize where 36 kg is returned to the next leach batch with the addition of 12 kg of fresh carbon. The 12 kg of loaded carbon removed at this stage is burnt to recover the gold. Leached slurry (screen undersize) is pumped directly to a tailings dam with addition of calcium hypochlorite ($Ca(OCl)_2$) to oxidise free cyanide. In turn, the free chlorine residual might have to be destroyed by the addition of sodium thiosulphate. It was assumed that the tailings could be pumped satisfactorily through a 760 mm diameter pipe and that a distance of 2 km is required.

80% extraction of gold (i.e. 48 kg) is assumed with NaCN consumption of 3.2 kg/t and CaO consumption of 3.3 kg/t.

5.2.1 Schedule of Equipment and Power Requirements

Item	No.	Size	Cost ($)	kWh/day
Front-end loader	1	1 m^3	10 000	
Feed chute	2		1 000	
Leach tank (mild steel)	2	5×10^4 1 (4m$\phi\times$4m)	14 000	
Agitator	2	30 kW	46 000	1 150
Pump #1	1	3kW, 2/1½	2 000	20
Compressor	1	5.5 kW, 15 1/sec.	1 800	106
Vibrating screen	1	0.6×1.2 m	3 500	
Pump #2 (tailings)	1	19 kW, 3/2	3 500	122
			81 800	1 398

5.2.2 Total Capital Investment

The total capital investment is estimated by the factored method whereby associated costs are based on experience factors of the total delivered cost of the major items listed in the section above. The estimate is summarized as follows:

	Item	Cost ($)
1a	Purchased equipment costs	81 800
1b	Delivery and location costs (10% of 1a)	8 180
1	Equipment cost - delivered	89 980
2-	Installation: 12% delivered equipment	10 800
3	Instrumentation: 3% delivered equipment	2 700
4	Piping: 20% delivered equipment	18 000
5	Electrical: 15% delivered equipment	13 500
6	Buildings: assumed	4 000
7	Foundations, structures: 5% delivered equipment	4 500
8	Land: assumed	nil
9	Yard improvements: 2% delivered equipment	1 800
10	Utilities: 2% delivered equipment	1 800
11	TOTAL DIRECT COSTS	147 080
12	Engineering, construction fees: 8% total direct costs	11 800
13	TOTAL DIRECT AND INDIRECT COSTS	158 880
14	Contingency: 25% total costs	39 720
15	FIXED CAPITAL INVESTMENT	198 600
16	Working capital: 4 months' operating costs	102 000
17	TOTAL CAPITAL INVESTMENT	300 600

5.2.3 Total Production Costs

	Item	Cost Total $	Cost $/kg Au
1	Raw materials		
	NaCN: 48 t at $1 450/t	69 600	1 450
	CaO: 71 t at $100/t	7 100	148
	Activated carbon: 2.5 t at $3 300/t	8 250	172
	$Ca(OCl)_2$: 8.3 t at $1 700/t.	14 110	294
	Total raw materials	99 060	2 064
2	Utilities		
	Water: 20 000 t at 14¢/t	2 800	58
	Power: 0.5×10^6 kWh/a at 6¢/kWh	30 000	625
	Total utilities	32 800	683
3	Direct labour		
	8 men, shifts at $15 000/a	120 000	2 500
	1 supervisor at $20 000/a	20 000	417
	Total direct labour	140 000	2 917
4	Maintenance: 10% fixed capital	19 860	414
5	Operating supplies: 10% direct labour	14 000	292
6	DIRECT MANUFACTURING COSTS (sum of 1 to 5)	305 720	6 370
7	Payroll overhead: 20% direct labour	28 000	583
8	Plant overhead: 50% direct labour	70 000	1 458
9	INDIRECT MANUFACTURING COSTS (sum of 7 and 8)	98 000	2 041
10	Depreciation	160 000	3 333
11	Property taxes, insurance: 1% fixed capital	1 990	41
12	FIXED MANUFACTURING COSTS (sum of 10 and 11)	161 990	3 374
13	TOTAL MANUFACTURING COSTS (sum of 6,9 and 12)	565 710	11 785

5.3 Solution Recycle

If disposal of the final slurry with chemical oxidation of the NaCN does not meet environmental specifications, one alternative is to install a three-stage counter-current decantation (CCD) system (see Fig. 2 for flowsheet). Use of this system would necessitate continuous rather than batch operation of the plant and hence two agitated slurrying tanks to premix slurry and control feed rate would be required. The purchase cost of equipment additional to that listed in section 5.2.1 for the basic flowsheet is shown below:

Item	No.	Size	Cost ($)
Slurry tank	2	2.3mϕ×2.6 m	2 600
Agitator	2	8 kW	16 000
Pump	1	3 kW, 2/1½	2 000
Thickener	3	4.4 mϕ×3 m	59 000
U/F pump	3		4 500
O/F pump	3		4 500
Sundry tanks (e.g. repulpers, flocculant)			10 000
			98 600

Adding the cost of extra equipment to that for the basic flow scheme gives \$180 400 purchased cost or \$198 440 delivered cost. Applying the factors shown in section 5.2.2 produces a Fixed Capital Investment of \$431 350 and a Total Capital Investment of approximately \$535 000.

Assuming the cost of raw materials remains the same (NaCN will be recycled but flocculant will have to be used, counteracting the saving), that power increases by 20%, and that all other factors remain the same as shown in section 5.2.3, then the Director Manufacturing Costs are \$335 000, while Indirect Manufacturing Costs remain at \$98 000. All but \$40 000 of capital cost is written off, giving a depreciation figure of \$390 000. Allowing for property taxes and insurance of \$4 300, the Total Manufacturing Costs become \$827 300 or \$17 235/kg Au.

5.4 Grinding

Laboratory tests show that if the dump material is ground to 80% −150 μm, then 95% extraction of the gold (57 kg) can be expected. A circuit involving grinding requires slurry preparation tanks, as described in the previous section, to control feed rate, as well as a ball mill. In view of the fine nature of the ore, the mill could be run as open circuit. A flowsheet of this circuit is shown in Fig. 3. A grinding time of 8 hours per batch (9 tonnes/hour) is assumed.

Again referring to the flowsheet described in section 5.2, the extra equipment requirement is shown below:

Item	No.	Size	Cost (\$)
Slurry tank	2	2.3 mϕ×2.6 m	2 600
Agitator	2	8 kW	16 000
Pump	2	3 kW, 2/1½	4 000
Ball mill	1	37.5 kW, 1.5 m× 1.5 m	72 000
			94 600

Applying the factors as in section 5.2.2 shows an equipment delivered cost of \$194,000, a Fixed Capital Investment of \$422 000, and a Total Capital Investment of \$524 000.

Assuming raw materials cost remains the same, power consumption is increased by 30% and that other factors remain the same as shown in section 5.2.3, then Direct Manufacturing Costs of \$338 000 are obtained while Indirect Manufacturing Costs remain at \$98 000. All but \$60 000 of fixed capital is written off, giving a depreciation figure of \$362 000. The Total Manufacturing Cost then calculates to be \$802 300 or \$14 075/kg Au.

5.5 Transport of Ore

Due to the fact that the Deloraine dump is in a water catchment area, ore might have to be transported to another site for treatment. Calculations have been made assuming the ore is transported to the Sedan/Cambrai area, and the estimated cost for transporting the entire dump of 15 000 t over a period of 2 to 3 weeks (using 10 trucks) is $31 500.

As the transport operation has no other costs attached to it, Total Manufacturing Costs for any of the three flow schemes considered would be increased by $31 500 total ($656/kg Au or $553/kg Au, depending on gold extraction).

6. DISCUSSION

The three processing options, with costs, are summarized below, with accuracy being ±30%.

Process	Basic ($)	Solution Recycle ($)	Grinding ($)
Total Capital Investment	300 600	535 000	524 000
Total Manufacturing Cost -			
On site	565 700	827 300	802 300
With ore transport	597 200	858 800	833 800
Value of Au extracted*	849 600	849 600	1 008 900
Profit before tax -			
On site	283 900	22 300	206 600
With ore transport	252 400	- 9 200	175 100

*Value of Au taken at $17 700/kg ($550/oz.)

From the estimations made, treating the ore by the basic flow scheme (i.e. no grinding, no liquor recycle) is the most profitable operation. If the ore has to be transported for environmental reasons, this process remains considerably more attractive than solution recycle on site. If, after relocation of the plant liquor recycle still has to be practised, then our estimations indicate the operation would be unprofitable. Although grinding would increase gold extraction from 48 to 57 kg, the added costs involved in buying a new mill more than compensate for this gain. The cost estimation for this flow scheme should be considered further, however, if a second mill can be obtained at a sufficiently reduced cost.

It must be emphasised that the work contained in this report is a preliminary cost estimation only and not a process design. The flow schemes used for the estimations include several assumptions and omissions which would have to be considered in detail to more accurately cost the proposed plant.

The depreciation figure assumes little resale value of the plant after one year. If, however, the plant could be sold as a unit, a higher price might be obtainable. Alternatively, if, at the completion of treating the Deloraine dump further material could be found for treating (e.g. on contract), the depreciation of the plant could be spread over several years, thereby reducing the manufacturing cost per kilogram of gold.

No allowance in the estimate has been made for building a tailings dam as this depends very much on local topography. The dam would have to hold 15 000 tonnes of solids plus approximately 20 000 to 25 000 tonnes of water unless some water recycle from the dam is carried out. Overflow in a water catchment area would probably not be allowed. Hence, the cost of building and fencing a dam for total tailings would be substantial.

Further detailed investigation is required to determine factors such as required leaching time, carbon adsorption and burning characteristics, and the conditions for satisfactory oxidation of the cyanide ion.

7. CONCLUSIONS

The main conclusions of the present investigation are as follows:

(1) The tailing contains 3.9 ppm Au and 0.21% Cu and consists predominantly of quartz and muscovite with minor to trace amounts of magnesian siderite, dolomite, barite, pyrite, chalcopyrite, covellite, chalcocite, malachite, goethite, hematite, rutile and native gold. About 70% of the tailings is -150 μm and about 30% is -53 μm in particle size.

(2) Relatively little of the native gold is liberated, most observable native gold occurring as inclusions in, or intergrowths with, pyrite, chalcopyrite and goethite. However, some native gold may be locked in silicates. The grain size of the native gold is mainly less than 25 μm. The colour of the native gold indicates a silver content of less than 10%.

(3) The distribution of gold with specific gravity and particle size indicates that gravity concentration would give low (probably less than 50%) recoveries of gold and, although finer grinding would improve gold liberation, recoveries by gravity concentration would be unlikely to increase significantly because of the less efficient separation at the finer particle size.

(4) Cyanidation of the tailing gives high extraction of gold - about 79% on the as-received material and 94% on material ground to 78% -150 μm. Acid-leaching prior to cyanidation increases the gold extraction slightly (to 86% on as-received material and 96% on material ground to 77% -150 μm) and reduces the cyanide consumption from about 3.3 kg/t to 1.0 kg/t, presumably by removing cyanicides such as malachite.

(5) The feasibilities of economic extraction of gold by (a) simple cyanidation, (b) cyanidation plus solution recycle (to minimize environmental pollution), and by (c) cyanidation after grinding have been examined, considering both treatment on site and treatment after transport to the Sedan/Cambrai area. Results are as follows:

Process	Basic $	Solution Recycle $	Grinding $
Total capital investment	300 600	535 000	524 000
Total manufacturing costs:			
On site	565 700	827 300	802 300
With ore transport	597 200	858 800	833 800
Value of Au extracted*	849 600	849 600	1 008 900
Profit before tax:			
On site	283 900	22 300	206 600
With ore transport	252 400	- 9 200	175 100

*Value of Au taken as $17 700/kg ($550/oz.)

It is emphasised that these are preliminary cost estimates only and no allowance has been made for building a tailings dam, should this prove necessary.

8. RECOMMENDATIONS

This investigation has been of a preliminary nature only and before proceeding to plant operation it is recommended that further metallurgical studies be undertaken to determine:

(a) the required leaching time for optimum gold extraction,

(b) the adsorption characteristics of various carbons,

(c) the conditions necessary for satisfactory oxidation of the cyanide ion, and

(d) the settling characteristics of the tailings.

TABLE 1: GOLD DISTRIBUTION WITH SPECIFIC GRAVITY AND PARTICLE SIZE

Size Fraction (μm)	Sp.Gr. Product	In Size Fraction Wt, %	In Size Fraction Gold Assay, ppm	In Size Fraction Gold Dist'n, %	In Head Sample Wt, %	In Head Sample Gold Dist'n, %	Cumulative Data* Wt, %	Cumulative Data* Gold Assay, ppm	Cumulative Data* Gold Dist'n, %
+150	<2.96	96.9	3.20	55.7	29.25	22.46	29.25	3.20	22.46
	2.96-4.20	2.9	69.00	36.1	0.88	14.59	0.88	69.00	14.59
	>4.20	0.2	206.00	8.2	0.07	3.31	0.07	206.00	3.31
	TOTAL	100.0	(5.57)	100.0	30.20	40.36	30.20	5.57	40.36
-150+105	<2.96	94.1	1.00	33.5	19.82	4.76	49.07	2.31	27.22
	2.96-4.20	5.4	19.80	37.7	1.13	5.35	2.01	41.39	19.94
	>4.20	0.5	160.00	28.8	0.11	4.09	0.17	177.75	7.41
	TOTAL	100.0	(2.81)	100.0	21.05	14.20	51.25	4.44	54.56
-105+53	<2.96	92.3	0.50	17.7	16.60	1.99	65.67	1.85	29.21
	2.96-4.20	6.2	10.00	23.7	1.11	2.66	3.12	30.23	22.60
	>4.20	1.6	98.00	58.6	0.28	6.58	0.45	128.54	13.98
	TOTAL	100.0	(2.60)	100.0	17.99	11.23	69.24	3.96	65.79
-53+10	<2.96	92.9	1.10	22.8	24.81	6.55	90.48	1.65	35.76
	2.96-4.20	5.4	7.50	9.0	1.43	2.58	4.55	23.06	25.18
	>4.20	1.8	172.00	68.2	0.47	19.55	0.93	150.74	33.53
	TOTAL	100.0	(4.47)	100.0	26.71	28.67	95.95	4.10	94.47
TOTALS FOR ABOVE SIZE FRACTIONS	<2.96	94.3	(1.65)	37.9					
	2.96-4.20	4.7	(23.06)	26.7					
	>4.20	1.0	(150.74)	35.5					
OVERALL TOTALS	TOTAL	100.0	(4.10)	100.0					
	+10 μm	96.0	(4.10)	94.5					
	-10 μm	4.0	5.70	5.5					
	TOTAL	100.0	(4.17)	100.0					

Calculated head assay = 4.17 ppm

* Cumulated by specific gravity product

TABLE 2: ACID PRE-LEACH

	1	2
Ore sizing	As Received	77% -150 μm
Weight (g)	339.8	314.2
pH	1	1
H_2SO_4 added (kg/t)		
Initial	44.5	49.3
1 hour	59.9	64.7
2 hours	66.7	71.6
4 hours	74.2	79.1
6 hours	79.5	84.8

TABLE 3: CYANIDE LEACHING

	1*	2*	3	4	5
Ore					
Sizing	As Received	77% -150 μm	As Received	75% -212 μm	78% -150 μm
Weight	328.7	302.2	274.3	277.7	305.6
Au (ppm)	3.9	3.9	3.9	3.9	3.9
Cu (%)	nd	nd	0.21	0.21	0.21
Residue					
Weight (g)	324.8	297.8	274.2	276.3	304.2
Cu (%)	0.23	0.26	0.28	0.25	0.40
Au (ppm)	0.52	0.15	0.95	0.40	0.19
Au (mg)	0.17	0.045	0.26	0.11	0.06
Final Liquor Au (mg)	1.19	1.15	1.22	0.94	1.04
Calculated Head					
Au (ppm)	3.6	3.8	4.5	3.4	3.4
Au Extraction (%)†					
5 hours	90	91	49	69	61
23 hours	86	99	82	90	95
29 hours	86	99	72	90	95
48 hours	85.8	96.1	78.7	88.2	94.4
NaCN Consumption (kg/t)					
5 hours	0.5	0.8	2.3	2.4	2.3
23 hours	0.9	1.1	3.1	3.3	3.1
29 hours	1.0	1.1	3.2	3.3	3.2
48 hours	1.0	1.1	3.2	3.3	3.2
CaO Consumption (kg/t)					
5 hours	1.6	1.8	1.9	1.9	1.8
23 hours	4.6	3.5	3.4	3.5	3.3
29 hours	4.8	4.0	3.5	3.7	3.4
48 hours	5.2	4.3	4.5	4.6	4.2

nd = not determined

*Ore samples for #1 and 2 are residues from acid pre-leach, Table 2

†Extractions at 5, 23 and 29 hours are approximate only

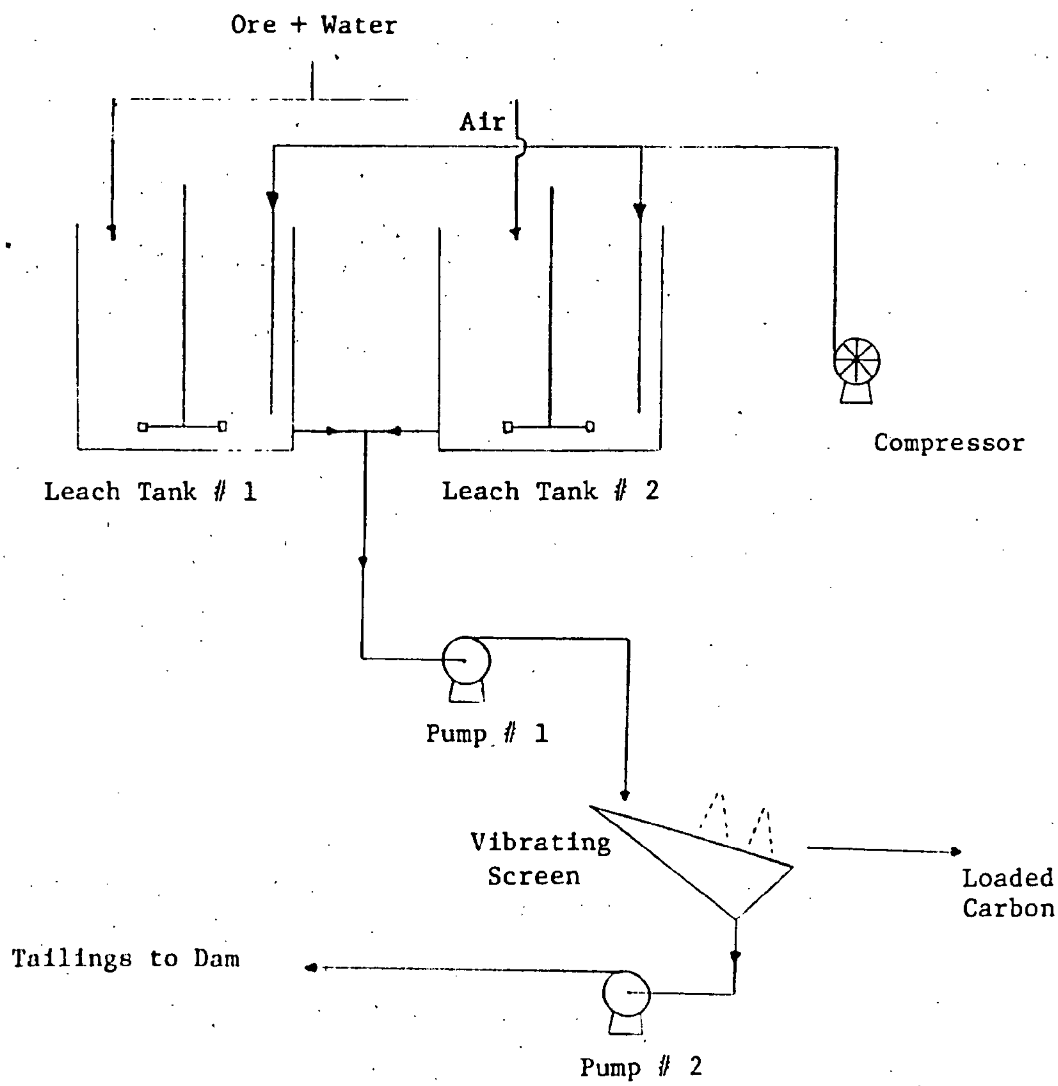


Fig. 1. Basic Flowsheet

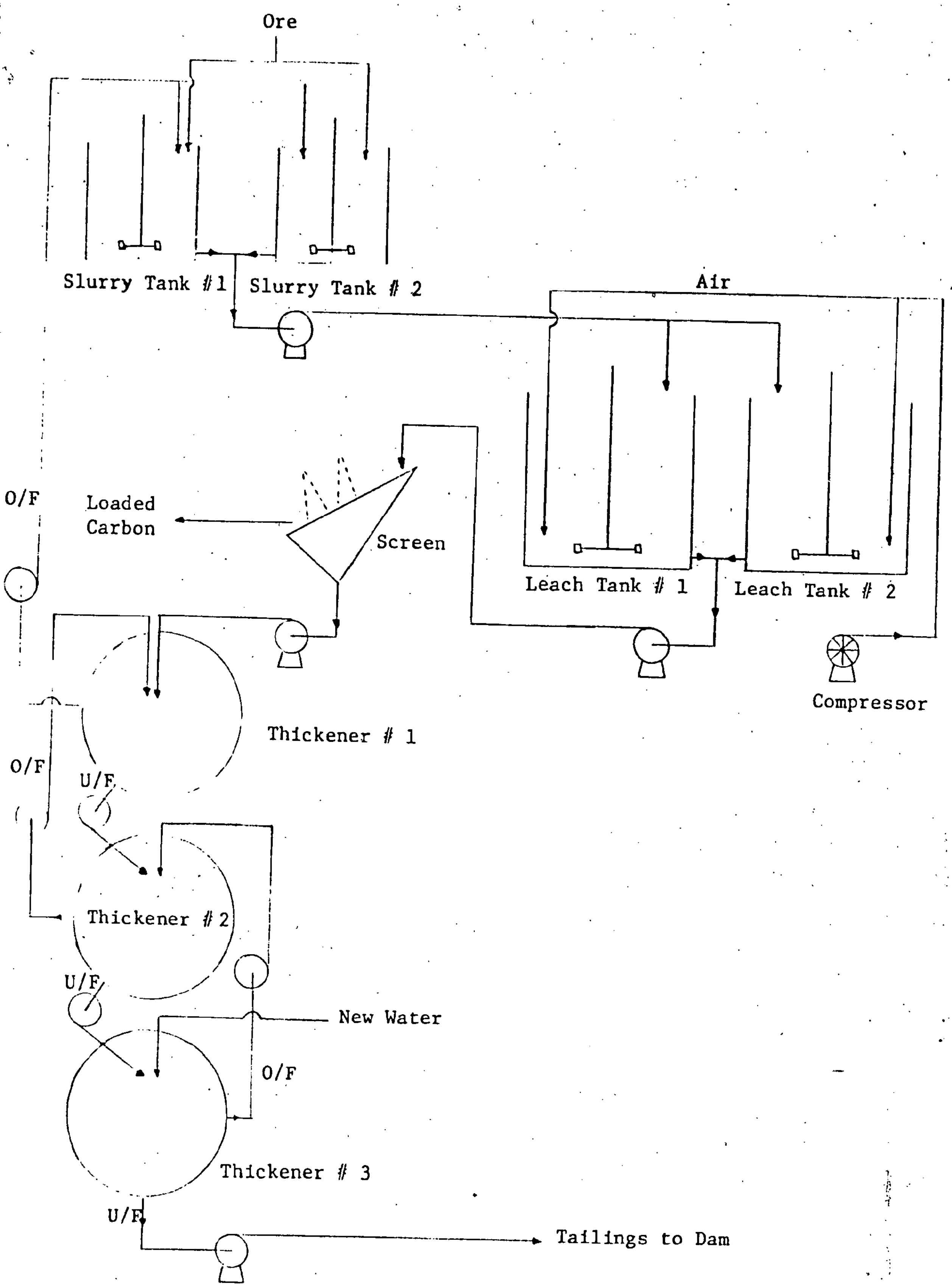


Fig. 2. Solution Recycle

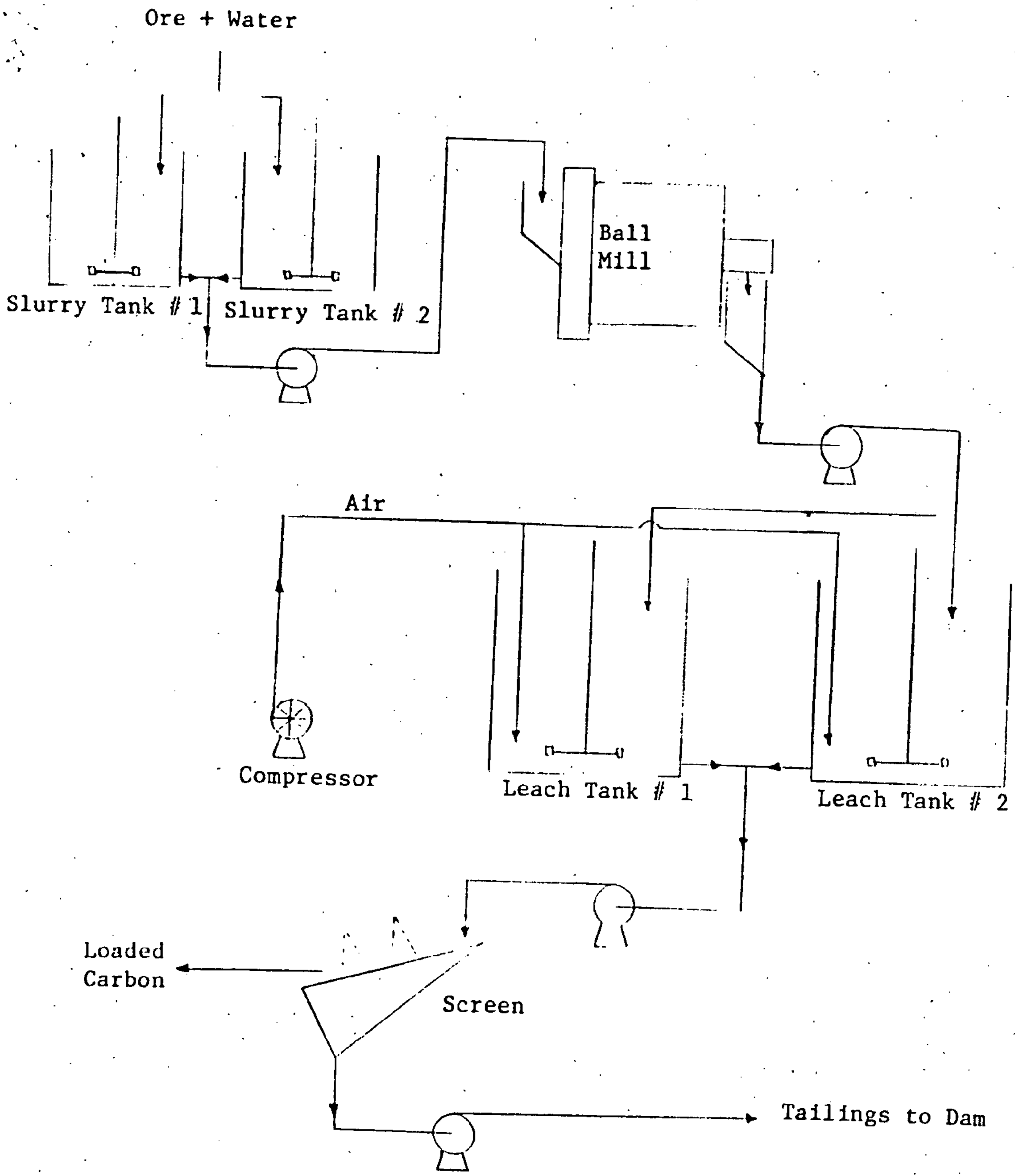


Fig. 3. Grinding

APPENDIX C

Geological Logs of Auger Holes with Gold and Copper Assays from AMDEL Report AC 2046/81.

HAND AUGER DRILL LOGS AND ASSAY RESULTS
* - duplicate sample assayed.

Hole No. (coordinates)	Sample No	Au (gms/t)	Cu (%)	Sample Interval (m)	Log
H.D.1 (65N,8E)	A850/80	2.05	0.19	0-1 m	Brown-grey fine grained silty sand with thin (2-5 mm) layers of yellow silty-clay
	*A900/80	1.90	0.19		
	A851/80	1.05	0.26	1.0-1.5	As above with more silty-clay layers.
	A852/80	1.55	0.40	1.5-2.0	Wet grey silty clay.
	A853/80	4.25	0.24	2.0-3.0	Yellow-brown fine silty sand and grey fine-med. sand
	*A901/80	3.30	0.20		
	A854/80	1.75	0.16	3.0-3.9	Grey-brown fine silty sand with silty clay layers.
	A855/80	0.65	0.09	3.9-4.0	Grey-black organic clay soil
H.D.2 (65N,16E)	A856/80	2.90	0.18	0-1.0	Yellow-brown fine silty sand to 60 cms then grey-brown fine silty sand.
	*A902/80	1.55	0.17		
	A857/80	1.55	0.33	1.0-2.0	Grey-brown fine silty sand to 1.3 m, wet grey silty clay to 1.4 m then grey-brown fine silty sand with thin layers of yellow silty clay.
	A858/80	0.15	0.24	2.0-3.0	Grey-brown fine silty sand grading to fine-medium sand.
	*A903/80	4.10	0.24		
	A859/80	2.15	0.13	3.0-4.0	Grey-brown fine silty sand with occassional silty-clay layer.
	A860/80	3.75	0.07	4.0-4.5	Yellow-brown fine silty sand. Hole bottomed on black organic clay soil.
H.D.3 (80N,10E)	A861/80	1.85	0.12	0-1.0	Brownish fine silty sand to 50cm yellow silty-clay to 1m.
	*A904/80	1.75	0.16		
	A862/80	1.25	0.26	1.0-2.0	Yellow and grey silty clay with minor sandy layer to 1.5m, followed by wet grey silty clay.
	A863/80	3.55	0.36	2.0-3.35	Grey fine-med. silty sand to 2.5m wet grey silty-clay to 3m, brown silty clay to 3.35m, hole bottomed on black organic clay soil.
H.D.4 (80N,20E)	A864/80	1.45	0.30	0-1.0	Pale brown fine silty sand with minor layers of yellow silty clay
	*A924/80	1.95	0.24		
	A865/80	2.85	0.23	1.0-2.0	Pale brown fine silty sand, wet grey silty clay from 1.2m-1.3m.
	*A906/80	2.15	0.31		
	A866/80	3.35	0.27	2.0-3.0	Pale brown fine silty sand with minor yellow silty clay bands, hole bottomed on black organic clay soil.
H.D.5 (80N,30E)	A867/80	3.40	0.24	0-1.0	Grey brown fine-med. silty sand with minor silty clay bands.
	*A907/80	2.85	0.23		
	A868/80	1.95	0.14	1.0-2.0	As above.

	A869/80	2.05	0.15	2.0-3.0	As above.
	*A908/80	2.35	0.20		
	A870/80	1.15	0.09	3.0-4.0	As above
	A871/80	2.60	0.07	4.0-5.3	Yellow-brown fine silty sand, some wet grey silty-clay layers at 4.6m, hole bottomed on black organic clay soil.
	*A909/80	3.30	0.10		
H.D.6 (100N,14E)	A872/80	1.95	0.17	0-1.0	Yellow-brown fine silty sand with minor yellow silty-clay layers.
	A873/80	1.25	0.23	1.0-2.0	Grey wet silty-clay.
	*A910/80	1.25	0.21		
	A874/80	2.45	0.25	2.0-3.1	Grey-brown, fine-med. silty sand to 2.2m then fine silty sand to 2.5m, then grey-brown silty clay, hole bottomed on black organic clay soil.
H.D.7 (100N,28E)	A875/80	1.85	0.14	0-1.0	Grey-brown, fine-med. silty sand
	*A911/80	2.20	0.19		
	A876/80	1.25	0.20	1.0-2.0	As above with minor yellow silty clay layers.
	A877/80	5.05	0.28	2.0-3.0	As above.
	*A912/80	4.85	0.32		
	A878/80	1.85	0.12	3.0-4.0	Fine silty sand to 3.5m, then wet grey silty clay.
	A879/80	2.35	0.18	4.0-4.9	Wet grey silty clay.
	*A913/80	2.00	0.19		
	A880/80	2.85	0.10	4.9-5.2	Pale grey, fine silty sand with minor yellow silty clay layers. Dark grey organic med. sand at 4.9 to 5.0m, hole bottomed on brown clay soil.
H.D.8 (100N,42E)	A881/80	3.55	0.25	0-1.0	Light brown fine silty sand with minor yellow silty clay layers to 0.5m, then fine-med. brown silty sand.
	*A914/80	2.80	0.20		
	A882/80	2.85	0.33	1.0-2.0	Brown, fine-med silty sand with minor yellow silty clay layers.
	A883/80	3.90	0.28	2.0-3.0	As above.
	*A915/80	3.10	0.21		
	A884/80	0.25	0.14	3.0-4.0	As above
	A885/80	2.25	0.09	4.0-5.0	As above with wet grey silty clay band from 4.90-4.95m.
	*A916/80	2.65	0.17		
	A886/80	16.1	0.13	5-5.50	As above. Hole bottomed on black organic clay soil.
H.D.9 (140BN,43E)	A887/80	1.55	0.19	0-1.0	Pale brown fine silty sand with minor yellow silty clay layers.
	*A917/80	1.85	0.19		
	A888/80	2.50	0.34	1.0-2.5	Yellow-brown fine sandy silt to 1.5m, then grey clay silt to 2.5m
	A889/80	2.20	0.46	2.5-2.95	Grey silty clay, hole bottomed on brown soil.
	*A918/80	2.35	0.30		
H.D.10 (140N,29E)	A890/80	2.00	0.31	0-1.0	Brown-yellow sandy silt with grey silty-clay at 1m.
	A891/80	2.00	0.37	1.0-1.8	Grey silty-clay,hole bottomed on brown soil.
	*A919/80	2.00	0.32		

H.D.11 (200N,50E)	A892/80	5.20	0.25	0-0.7	Pale brown fine silty sand to 0.5m then yellow-brown silty clay to 0.7m, hole bottomed on brown soil.
H.D.12 (200N,25E)	A893/80	6.70	0.23	0-0.4	Fine yellow-brown silty sand, hole bottomed on brown soil.
H.D.13 (200N,37E)	A894/80	6.25	0.21	0-0.8	Yellow-brown fine silty sand, hole bottomed on brown soil.
	*A920/80	6.50	0.21		
H.D.14 (250N,50E)	A895/80	6.95	0.22	0-1.0	Grey fine-med. sand with yellow silty clay layers to 0.6m, then yellow-brown silty clay to 1m.
	*A921/80	6.95	0.18		
	A896/80	6.70	0.18	1.0-1.7	Yellow-brown silty clay to 1.3m, then fine silty sand and fine-med silty sand, hole bottomed on brown soil.
H.D.15 (250N,27E)	A897/80	5.40	0.15	0-0.5	Yellow-brown fine silty sand, hole bottomed on brown clay and weathered rock.
	*A922/80	7.80	0.16		
H.D.16 (400N,38E)	A898/80	5.50	0.22	0-1.0	Yellow-brown and grey clay-silt layers with fine silty sand layers.
	A899/80	6.70	0.29	1-1.7	As above, hole bottomed on black organic clay soil.
	*A923/80	6.70	0.32		

*Duplicate sample assayed.

APPENDIX D

Calculations

1. Reserves of Tailings
2. Gold Grade of Tailings

1. Reserves of Tailings

Main Tailings Dump

Volume calculations were determined by calculating the area of each section on figure 4. multiplied by its distance of influence, taken as half way between sections or to the edge of the dump where applicable.

An average specific gravity (S.G.) of the tailings was assumed to be 1.73, which is the S.G. of loose quartz sand containing several percent heavy mineral.

Section	Area(m^2)	Distance(m)	Volume(m^3) calculated	rounded
A-A'	155.6	20	3 112	3 100
B-B'	173.5	17.5	3 036	3 000
C-C'	195.1	23.5	4 585	4 600
D-D	92.6	26.7	2 472	2 500
Southern Batter	30	23	690	700
Batter between C-C' and D-D'	12.5	25	312	300
TOTAL			14 207	14 200

Tonnes = 14 200 x 1.73
= 24 566 say 24 600

Scattered Tailings

Area assumed to be 200 m long by 50 m wide with an average depth of tailings of 0.5 m.

Volume:- 5 000m^3
Tonnes:- 8 650
say 8 600

Slimes Dump

Area taken as 770m^2, depth assumed to be 1.70m as determined in auger hole H.D.16.

Volume:- 1 309m^3 say 1300m^3
Tonnes:- 2 249
say 2 200

Summary

	Volume(m^3)	Tonnes
Main dump	14 200	24 600
Scattered	5 000	8 600
Slimes	1 300	2 200
TOTAL	20 500	35 400

2. Gold Grade of Tailings

- In general samples represented one metre intervals from auger holes
- Assay results are shown in Appendix C.
- Where duplicate samples were analysed the average of the two results was taken as representative of that interval.
- Weighted averages, based on sample interval, were calculated for each hole (Table 4).
- The average gold content for each cross section (Fig. 4) is the average of the weighted averages for each hole on that section.
- The weighted average for the Main Tailings Dump was calculated from the average for the section and the volume represented by the section.
- For the Scattered Tailings and the Slimes Dump the average of the weighted averages for the holes in the area was used.
- The arithmetic mean (A.M.) and standard deviation (S.D.) was calculated for the Main Tailings Dump and the Scattered Tailings area, insufficient data is available for the Slimes Dump.
- At the 95% confidence level the range of variability of the A.M. lies between ± 2 S.D., and it is assumed that this range is the same for the weighted average.

Main Tailings

A.M.: 2.35 g/t, Au. weighted average 2.46g/t
S.D.: 0.85 g/t, Au.
2S.D: 1.70 g/t, Au.

Scattered Tailings

A.M.: 6.,31 g/t, Au. weighted average 6.21g/t
S.D.: 0.65 g/t, Au.
2S.D.: 1.30 g/t, Au.

TABLE 4
Average Grade Per Drill Hole

Hole No.	Depth(m)	Weighted Av gold grade (g/t)
H.D.1	4.00	2.22
H.D.2	4.50	2.19
H.D.3	3.35	2.38
H.D.4	3.00	2.52
H.D.5	5.30	2.32
H.D.6	3.10	1.90
H.D.7	5.20	3.03
H.D.8	5.50	3.69
H.D.9	2.95	2.20
H.D.10	1.80	2.00
H.D.11	0.70	5.20
H.D.12	0.40	6.70
H.D.13	0.80	6.38
H.D.14	1.70	6.85
H.D.15	0.50	6.65
H.D.16	1.70	5.99

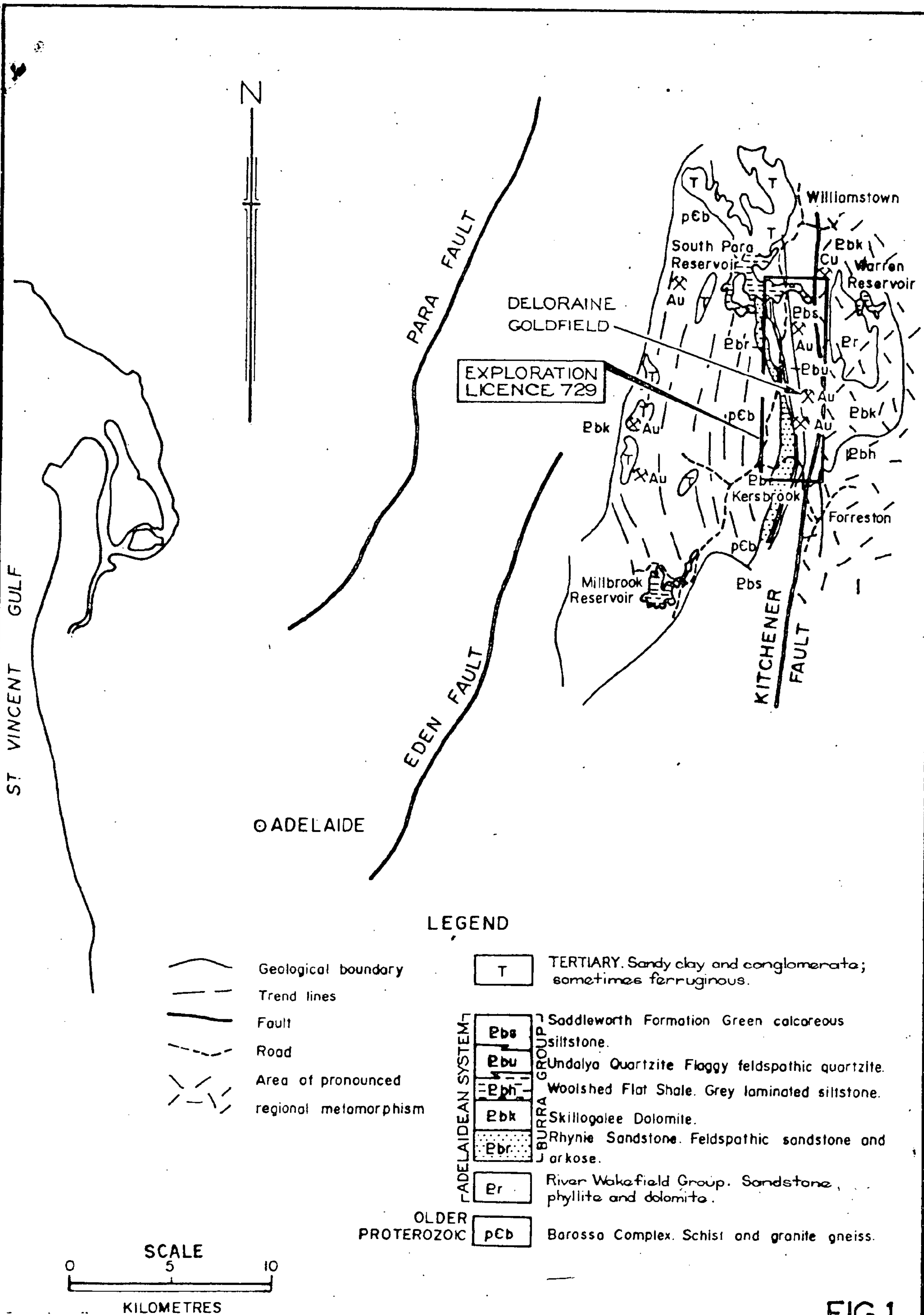
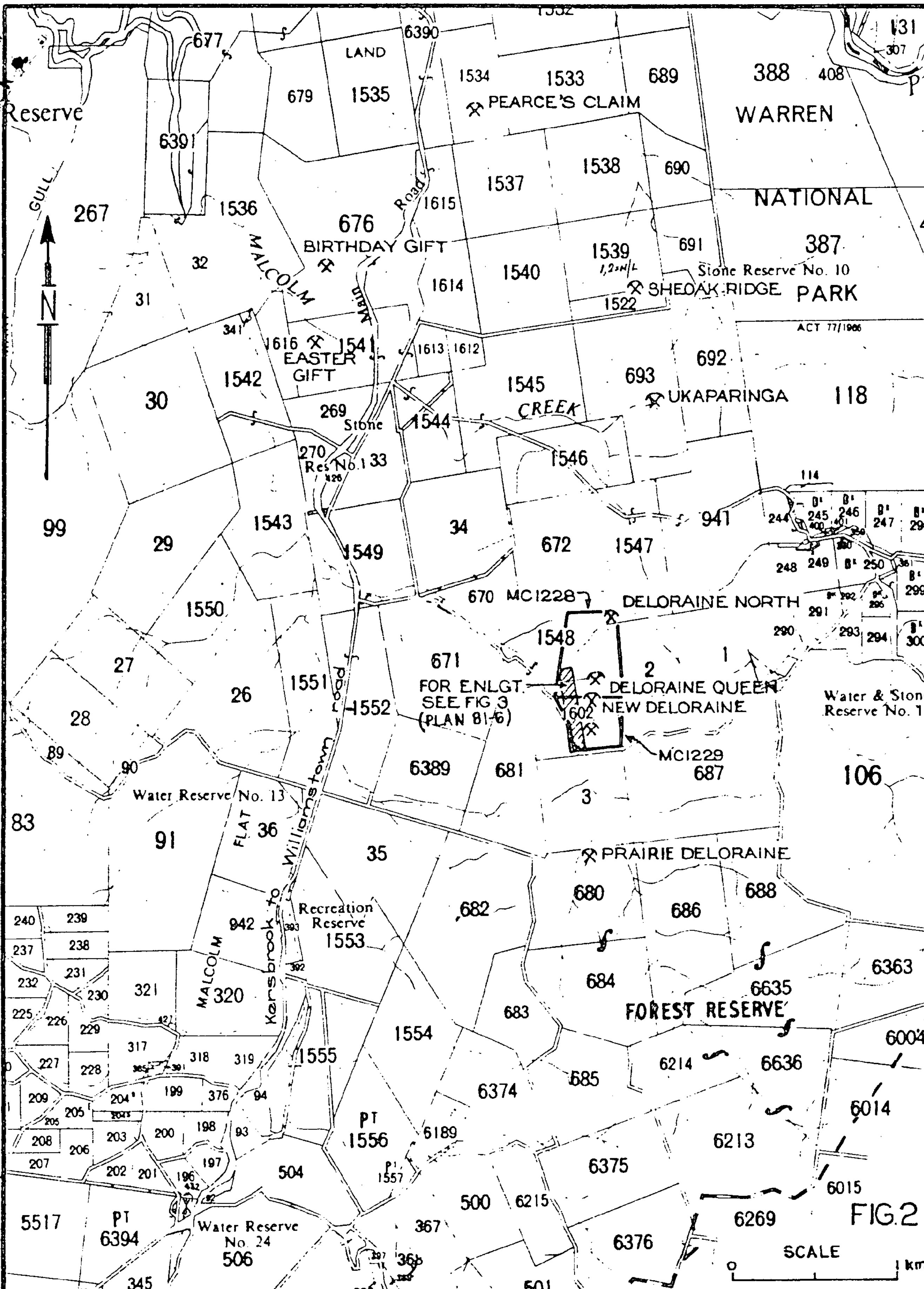


FIG.1

	DEPARTMENT OF MINES – SOUTH AUSTRALIA	SCALE 1:250,000
COMPILED: B. Morris	DELORAINE GOLDFIELD TAILINGS SECS. 1548 & 1602 HD. PARA WIRRA LOCATION AND REGIONAL GEOLOGY	DATE: Jan. 81
DRN: J.W. CKD.		PLAN NUMBER:
13·3·81		S15255

6628-1



DEPARTMENT OF MINES AND ENERGY
SOUTH AUSTRALIA

DELORAINE GOLDFIELD TAILINGS
SECS. 1548, 1602 HD. PARA WIRRA
LOCALITY PLAN

COMPILED B. Morris	C.D.O. DATE 13-3-81
DRAWN J.W.	SCALE 1:31,680
DATE Jan. '81	PLAN NUMBER
CHECKED	S15256

LEGEND

Weighted average values for hole - { Au 2·00(g/t) Cu 0·33(%) H.D.10 o (1·8m) Location and number of hand auger hole with depth in metres.

Stn.1 310m Survey Station.

.304·5m Spot elevations in metres. Arbitrary Datum.

Cross sections shown in Fig. 4. (Plan 81-7)

Embankment.

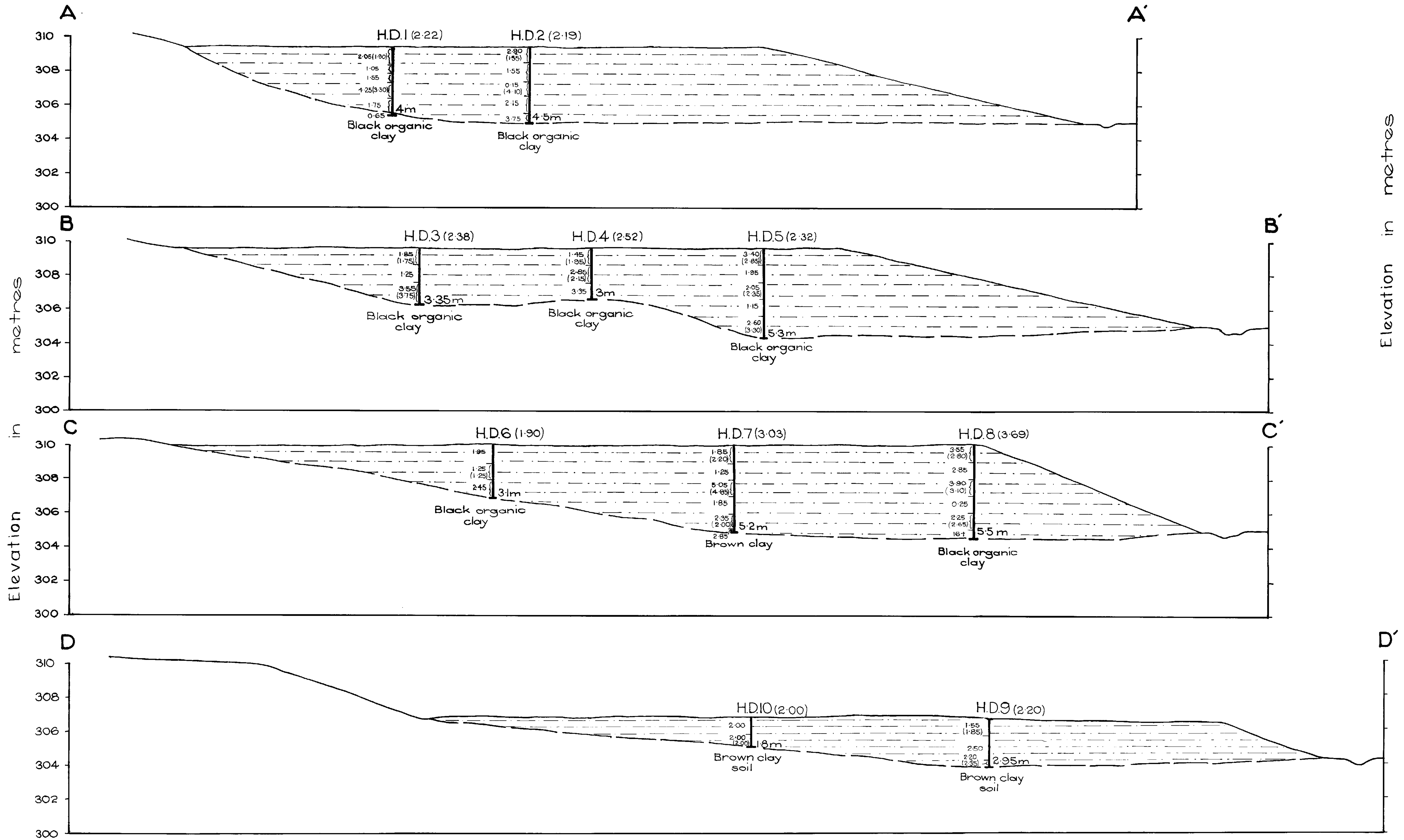
100N x 0E Grid pegs with coordinates.

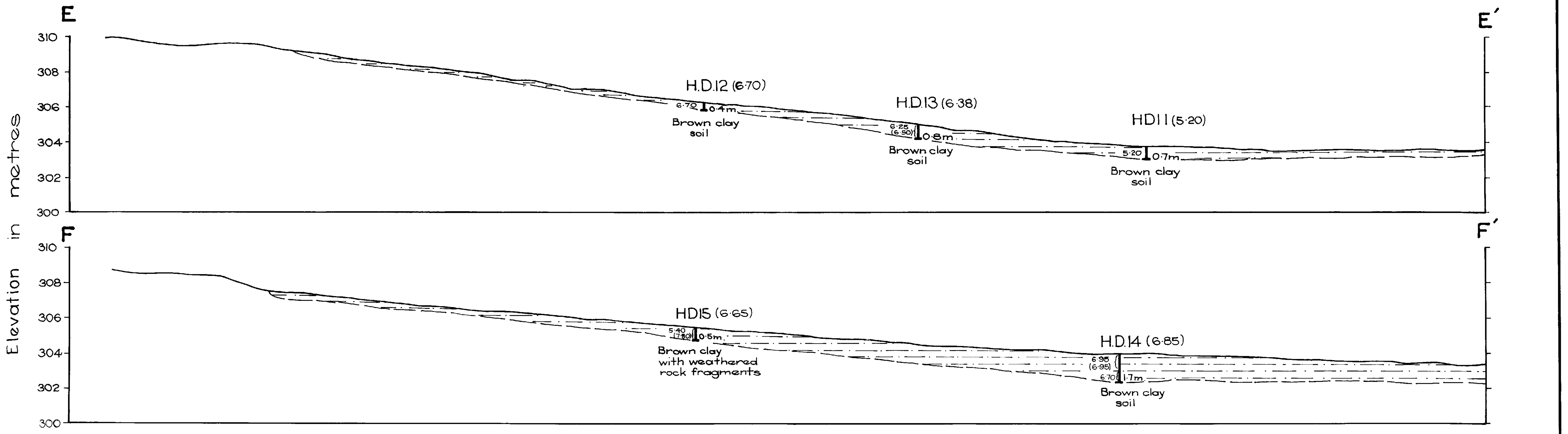
Note: Location of plan shown on Fig. 2. (Plan S15256)
S.F.B. 595 for survey reference.

SCALE
metres 0 10 20 30 40 metres

FIG.3

DEPARTMENT OF MINES AND ENERGY SOUTH AUSTRALIA	COMPILED B.Morris	13·3·81 DATE
DELORAINE GOLDFIELD TAILINGS SECS. 1548, 1602 HD. PARA WIRRA SLIMES DUMPS & DRILL HOLE LOCATIONS	DRAWN J.W.	SCALE 1:500
	DATE 5-12-80	PLAN NUMBER
	CHECKED	81-6





LEGEND

H.D.5 (2·32) ...... Weighted average of gold for hole in grams per tonne.

Gold value in grammes/tonne → 3·40; Duplicate assay → (2·85); 1·95; 5·3m — Number and location of hand auger hole with depth in metres.

Tailings : Interlayered fine sand, silt and clay.

SCALE

metres 0 5 10 15 20 metres

Note: Location of cross-sections shown on Fig. 3. (Plan 81-6)

FIG. 4

DEPARTMENT OF MINES AND ENERGY SOUTH AUSTRALIA	COMPILED B. Morris
	DRAWN J. Williams
	DATE Dec. '80
DELORAINE GOLDFIELD TAILINGS	CHECKED 13·3·81 C.D.O. DATE
SECS. 1548, 1602 HD. PARA WIRRA	SCALE 1:200
CROSS SECTIONS OF TAILINGS DUMPS	PLAN NUMBER 81-7