BILLITON AUSTRALIA GOLD PTY. LTD.
MT. TODD JOINT VENTURE
Annual Report for Period Ending 28 January, 1989

APPENDIX X
Groundwater Drilling and Testing,
Batman Water Supply Mt. Todd, NT
Vols. I & II
- Rockwater

BILLITON AUSTRALIA PTY LTD

GROUNDWATER DRILLING AND TESTING
BATMAN WATER SUPPLY
MT TODD, NORTHERN TERRITORY

VOLUME I - REPORT
DECEMBER, 1988

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Rockwater
PROPRIETARY LIMITED

consultants in groundwater exploration and development

BILLITON AUSTRALIA PTY LTD

GROUNDWATER DRILLING AND TESTING REPORT
BATMAN WATER SUPPLY
MT TODD, NORTHERN TERRITORY

NOVEMBER 1988

1.0 INTRODUCTION

This report details the drilling programme for which Billiton Australia Pty Ltd engaged Rockwater Pty Ltd, to evaluate potential water sources for use in the development of the Batman Gold prospect, located in the Mount Todd area, approximately 50 km north-west of Katherine.

A groundwater supply of 600,000 cu m/yr (1,640 cu m/d) is required for the initial plant. This would increase should the treatment plant be subsequently expanded. In this report, consideration is given to locating larger groundwater supplies, up to 2 x 10^6 cu m/year.

Potential sources for process water were to be assessed by drilling ten exploration holes, of which five high-yielding areas were to be developed by the installation of test-production bores. Exploration was restricted to two exploration leases held by Billiton Australia Pty Ltd, EL 2204 to the west and EL 2935 to the east. These leases occupy areas to the south and west of Mt Todd, and they contain the Batman prospect.

Site selection for the exploratory holes was made by means of examination of local geology, in conjunction with aerial photograph mapping and interpretation. Bore locations are presented in Figure 1.

Exploratory drilling began in mid October 1988 and was immediately followed by drilling and construction of production bores at the end of October. All drilling was completed in early November 1988. Test pumping began at the end of October and was completed by mid November. All drilling, construction of bores and test pumping was carried out by Gorey and Cole Drillers.

2.0 PREVIOUS WORK

In July 1988, Rockwater was engaged to carry out a water resource evaluation for process and potable water. In addition to background research, a field inspection was conducted in late July 1986. Details of this study were included in the Rockwater Report Number 128.2/88/1, dated August 1988. The reader is referred to this report for an analysis of the region's climate, meteorology and hydrogeology. However, a brief summary will be given below.

3.0 PHYSICAL SETTING

3.1 PHYSIOGRAPHY

The Mount Todd region is hilly with the most prominent physical feature, Mount Todd, situated about 3.5 km to the east north east of the Batman Prospect. Surface drainage from the prospect area trends to the south and west by means of tributaries, into the Edith River, located about 3.5 km to the south.

3.2 CLIMATE

The climate in the Mt Todd Region is sub-tropical, comprising a warm dry winter from May to September with prevailing south-easterly winds, and a hot wet summer from October to April, with dominating north-west to westerly winds.

3.2.1 Rainfall

The mean annual rainfall recorded at Katherine is 948 mm per annum with the mean monthly rainfall being highest during the months of December to March.

3.2.2 Evaporation

The mean annual evaporation from a free water surface is 2,566 mm per annum, being highest in October at 303 mm, and lowest in February at 142 mm.

3.2.3 Temperature

Temperatures are generally high, with the highest mean daily maximum in June/July of 38°C. The mean daily minima range from 12°C to 24°C respectively.

3.2.4 Humidity

Humidities are relatively high, with 0900 hours readings ranging from a mean maximum in February of 82%, to a mean minimum in August/September of 51%.

3.2.5 Vegetation

The region is covered with a light Eucalypt woodland and a grassy groundcover. Land use is not dominated by any significant human influence. The natural groundwater recharge systems are, therefore, likely to be currently operating.

4.0 HYDROLOGY

In the vicinity of the Batman Prospect and the Robin Zone, groundwater was encountered in fractured bedrock during mineral exploration drilling. Relatively large volumes were estimated from airlifting some of these holes, but the results remained unconfirmed. Bedrock in this region is composed of siltstone, shale and greywacke, which, in the vicinity of the prospect area, is known to be hornfelsed and locally mineralised.

From analyses carried out on bores sunk in the region to date, salinities are in general low. Contaminants such as arsenic and iron are likely only to be found in aquifers that are mineralised.

5.0 DRILLING AND BORE CONSTRUCTION

5.1 PROGRAMME

Gorey and Cole drillers of Alice Springs were awarded the drilling and test-pumping contract. Drilling commenced on September 17 and the field operations were completed on November 16 1988. A total of eleven exploratory holes (BW1 - BW10 and BW13), and five production bores (BW1P, BW2P, BW6P, BW8P and BW10P), were drilled.

5.2 METHODS

Exploration and production bores were drilled by air-rotary and air-hammer methods with some use of water and foam injection. An Ingersoll-Rand TH-60 rig was used.

All holes were lithologically logged and sampled. Water production rates achieved by air-lift were measured at regular intervals to allow for identification of water-bearing zones. Water salinities were measured on site, by conductivity. Samples were taken from completed production bores for laboratory analysis.

Exploratory holes were drilled at 190 mm diameter. Those that yielded significant water were cased with 50 mm ID UPVC Class 9, slotted through the water-bearing zones. They were used for water level monitoring during test pumping.

Five production bores (BW1P, BW2P, BW6P, BW8P and BW10P) were subsequently constructed at sites that yielded high air-lift rates. They are located more than 600 m apart to minimise any interference effects. Bore locations are shown in Figure 1.

Each production bore was drilled near to the observation bore. Holes were drilled at 254 mm diameter and completed using 155 mm ID Class 9 UPVC casing, with slots set in the water-bearing layers.

Drill holes not completed as production or observation bores were backfilled and the sites levelled, in accordance with environmental procedural guidelines. They are designated as abandoned.

6.0 RESULTS

6.1 TEST HOLES

Eleven exploration holes were drilled within a 3 km radius of the prospect. They are located mainly to the east and south-east of the Batman Prospect. Five of these holes developed sufficient water to warrant production bore construction.

Results of the exploratory and observation bores are presented in Appendix I and summarized in Table 1. Details of production bores are presented in Appendix II and in composite bore logs in Figures 2 to 6. A summary is presented in Table 2.

Groundwater was generally intersected at about 30 m in fractures within the greywacke and mudstones of the Burrell Creek formation. Water levels in these holes stand above the first level of intersection, indicating that the aquifer zone is confined by impermeable rock.

6.2 PRODUCTION BORES

Production bores were drilled and constructed at five sites (BW1P, BW2P, BW6P, BW8P and BW10P) to depths ranging from 60 m to 72 m below ground level.

Field salinities of groundwater samples taken during drilling ranged from 84 to 303 mg/l total dissolved solids (TDS), measured by conductivity. These are similar to values obtained from the test holes.

Bore BW1P penetrated fractured greywacke with minor seepage from 29 m, and the main supply from below 51 m. During drilling, salinity measurements ranged from 89 to 102 mg/l TDS, similar to the adjacent observation bore. Static water level is 18.53 m below ground level.

Bore BW2P intersected minor seepage from 33 m, increasing to 144 cu m/d at 48 m and 576 cu m/d at 72m. Once cased with 155 mm ID UPVC, the airlift yield was the same or more than prior to casing. Salinities of groundwater ranged from 147 to 164 mg/l TDS.

TABLE 1

SUMMARY OF EXPLORATORY DRILLING

NOVEMBER 1988

BORE No.	STATUS	GRID REFFERANCE	DATE COMPLETED	S.W.L (m bgl)	TOTAL DEPTH DRILLED (m)	AIRLIFT YIELD (cu m/d)	TOTAL DISSOLVED SOLIDS (mg/l)
BW 1	Observation	9896N 9805E	17/10/88	18.66	65	576	107
BW 2	Observation	10492N 9684E	18/10/88	26.65	77	376	150
BW 3	Exploration	9035N 9130E	19/10/88	-	71	Dry	-
BW 4	Exploration	8910N 8752E	19/10/88	-	52	Dry	-
BW 5	Observation	10600N 9000E	20/10/88	18.27	65	43	105
BW 6	Observation	8955N 10365E	20/10/88	18.87	71	258	300
BW 7	Exploration	9450N 11100E	21/10/88	-	36	Dry	-
BW 8	Observation	9598N 10253E	22/10/88	24.65	68	216	270
BW 9	Observation	10146N 10200E	23/10/88	23.76	65	190	250
BW10	Observation	9163N 9960E	23/10/88	23.22	71	288	84
BW13	Exploration	10505N 8351E	7/11/88	3.00	58	54	140

TABLE 2

SUMMARY OF PRODUCTION BORE DATA

BORE NUMBER	LOCATION	DEPTH* DRILLED (m)	SPECIFICATIONS	SLOTTED* INTERVAL (m)	STATIC* WATER LEVEL (m bgl)	RECOMMENDED PUMPING RATE (cu m/d)	PUMP* SETTING (m)	INTERNAL DIAMETER OF PUMP HOUSING (mm)	LABORATORY SALINITY (mg/1 TDS)+
BW 1P	9906N 9806E	62.6	+0.25-5,75 m 310 mm ID 323 mm OD Steel Surface Casing. +0.25-62.6 m 155 mm ID 168 mm OD Class 9 UPVC	26.00-62.6	18.53	250	48	155	195
BW 2P	10482N 9689E	72	+0.60-5.40 m 310 mm ID 323 mm OD Steel Surface Casing. +0.60-70.90 m 155 mm ID 168 m OD Class 9 UPVC	34.90-70.90	25.99	300 or 350	55	155	242
BW 6P	8984N 10362E	70	+0.30-2.70 m 310 mm ID 323 mm OD Steel Surface Casing. +0.30-70.00 m 155 mm ID 168 mm OD Class 9 UPVC	28.00-70.00	18.79	300 or 350	50	155	477
BW 8P	9615N 10266E	60	+0.80-27.84 m 310 mm ID 323 mm OD Steel. +0.80-58.63 m 155 mm ID 168 mm OD Class 9 UPVC	28.63-58.63	23.70	300 or 250	50	155	410
BW10P	9162N 9982E	70	+0.33-32.09 m 310 mm ID 323 mm OD Steel Surface Casing +0.25 m-70.00 m 155 mm ID 168 mm OD Class 9 UPVC	29.90-70.00	22.65	250	65	155	149
					SUM	1400			

* Metres Below Ground Level
+ Salinity - Milligrams Per Litre Total Dissolved Solids (Field Determination - By Conductivity)

Bore BW6P was drilled to 70 m, penetrating fractured greywacke, with a thin cover of alluvium. The bulk of the water was produced below 39 m, with flow rates ranging from 247 to 375 cu m/day. Groundwater salinity ranged from 299 to 306 mg/l TDS.

Bore BW8P penetrated 60 m of fractured greywacke with minor chert veins, Flow rates from the main aquifer in the production bore were measured by airlift methods from below 45 m and ranged from 288 to 864 cu m/day. Field measurements indicate the groundwater salinity to be within the range 258 to 276 mg/l TDS.

Bore BW10P was drilled to 71.5 m, penetrating completely weathered greywacke clays near the surface and grading to fractured, fresh greywacke with depth. Water was first encountered at 36 m depth. During the final airlift, flow rates ranged from 216 to 575 cu m/day. Groundwater salinities were measured at 83 mg/l TDS.

7.0 TEST PUMPING

Six production bores were test-pumped to assess bore yields and aquifer characteristics. This phase of the investigation extended from 27 September through to 16 November 1988. Five of the bores were pumped using a shaft-driven turbine pump with an orifice weir assembly to monitor the pumping rate. (One existing bore was pumped using the existing facilities, and monitored for one day,) Water levels were measured in the pumped and observation bores, using electric probes.

On the five new bores, two types of pumping tests were conducted: step-rate and constant-rate tests. The step-rate tests (comprising four steps of one hour each at incrementally increasing rates), were used to determine suitable pumping rates for the constant-rate tests. On the existing bore a constant-rate test alone was conducted. Plots and interpretations of the step-rate tests are not presented herein.

The constant-rate tests were each run for two days with the exception of BP70 which ran for 24.5 hrs. Pumped water was channelled from the site, by using nearby or adjacent stream beds. This was done to ensure that the possibility of artificial recharge was kept to a minimum.

Data from the test pumping are presented in Figures 7 to 12, and a brief summary of results is presented below. In the discussion, long-term refers to a time span of about two years.

7.1 BORE BW1P

Figures 2 and 7

The constant-rate test of bore BW1P commenced at 350 cu m/d following a step-rate test at 200, 300, 400 and 500 cu m/d. Drawdowns were measured in the pumped and observation bores. Water levels were also monitored intermittently in observation bores about 500 m on either side.

The plot of drawdown in the pumped bore, on semi-logarithmic scale, shows steepening of the trend, particularly after 700 minutes. This indicates that the transmissivity in the surrounding aquifer beyond an apparent hydraulic barrier is lower than that at the bore. From the late time data, the aquifer transmissivity is calculated to be 9 cu m/d/m, while the storage coefficient is calculated to be 1.4×10^{-3}. Extrapolation of the trend indicates that the available drawdown of 29.5 m between static water level and the main aquifer interval at 48 m would be exceeded within two years.

A reduced rate of 250 cu m/d is, therefore, recommended as a production duty rate. At this rate, assuming that there is no interference with bores other than BW2P, and that no other deleterious boundaries become effective, the long-term pumping water level is expected to be about 44 m below ground level. A pump inlet level of 48 m below ground level will be suitable for this bore. Intermittent monitoring of bores BW2, BW8 and BW9 during the constant rate test showed that only BW2 responded, with a measurable drawdown of 0.09 m after 2 days. Both BW1P and BW2P penetrate the mineralisation in the Robin area, through which a hydraulic connection is indicated. Therefore, some interference drawdown is likely to occur.

Regular monitoring of pumping water levels will be important in assessing whether any deleterious hydraulic effects are occurring.

7.2 BORE BW2P — Figures 3 and 8

A constant-rate test was run at 300 cu m/d on bore BW2P, following a step-rate test at discharges of 76, 250, 300 and 400 cu m/d.

The drawdowns in the pumped and observation bores followed similar trends on semi-logarithmic scale, being essentially linear after about 120 minutes of pumping. The steeper trend within the initial 120 minutes subsequently shallows out and is probably due to aquifer development adjacent to the bore.

Aquifer parameters calculated from the observation bore data are:

Transmissivity:	84.5 cu m/d/m
Storativity:	3.8×10^{-9}

The results indicate a moderately high aquifer transmissivity (for fractured rock aquifer) and a low storativity reflecting the confined aquifer condition.

The maximum drawdown in the pumped bore was 5.89 m. Extrapolation of the drawdown trend indicates that the bore could sustain more than 300 cu m/d, and a pumping rate of 300 - 350 cu m/d is recommended.

Including some minor interference drawdown from bore BW1P, the long term pumping water level is estimated to be 34 m below ground level. A pump setting of 55 m should be satisfactory.

7.3 BORE BW6P

Figures 4 and 9

The constant-rate test of bore BW6P was commenced at 350 cu m/d following a step-rate test at 150, 200, 300 and 400 cu m/d.
Drawdown in both the pumped and observation bores followed similar trends on a semi-logarithmic scale. The curves steepened at about 300 minutes, indicating the presence of an hydraulic boundary. The value of the effective aquifer transmissivity calculated from late time data for the bore is 30.5 cu m/d/m, with a storage coefficient of 1.3×10^{-5}. The available drawdown between static water level and the main aquifer is about 43 m.

The bore is capable of 300 - 350 cu m/d provided no further aquifer boundaries are intersected and there is no interference from the other bores in the region. The long-term pumping water level is estimated to be 33 m below ground level with a pumping rate of 350 cu m/d. A pump setting of 50 m is recommended.

7.4 BORE BW8P

Figures 5 and 10

Bore BW 8P was test pumped at a constant rate of 400 cu m/d, following a four hour step-rate test at 200, 400, 600 and 800 cu m/d.

The plots of drawdown on semi-logarithmic scale show steepening of the trends, particularly after 400 minutes. At the end of the test the drawdown reached 7.78 m. Analysis of the late time data indicates an aquifer transmissivity of 20.3 cu m/d/m, and a storage coefficient of 1.5×10^{-3}.

The bore has an available drawdown of 30.3 m between static water level and the main aquifer intersection at 54 m. Extrapolation of the drawdown trend indicates that the available drawdown would be exceeded within two years. A reduced rate of 250 - 300 cu m/d is, therefore, recommended as a production duty rate. At a production rate of 300 cu m/d, in the long term, a pumping water level of 36.7 m below ground level is estimated, assuming no interference with other bores, or any other deleterious hydraulic boundary effects are realised. Regular monitoring of pumping water levels will be important in assessing whether the recommended rate is appropriate. A pump setting of 50 m is recommended.

7.5 BORE BW10P

Figures 6 and 11

Bore BW10P was test-pumped with a step-rate test at rates of 200, 250, 300 and 350 cu m/d, followed by a constant rate test at 250 cu m/d. The drawdown was rapid for the first 1000 minutes reaching 17.64 m, after which the rate of drawdown decreased. At the end of the test the drawdown reached 19.61 m. Analysis of the late time data indicates an aquifer transmissivity of 14.76 cu m/d/m, and a storage coefficient of 9.6×10^{-7}.

The bore has an available drawdown of 42 m The main aquifer interval is from 65 to 70 m. Given a projected long-term drawdown of about 40 m, ie a pumping water level of 62 m (provided no interference from other bores or aquifer boundaries are realised) the bore should sustain 250 cu m/d. A pump setting of 65 m is recommended.

7.6 BORE BP70 Figure 12

This existing bore was monitored for several days, during which the pumping rate was measured by means of an orifice weir assembly. This bore is currently used intermittently to supply camp facilities and mineral exploration drilling operation. Presently, the pump is set at 35 m below ground level with 17.3 m of available drawdown below a static water level of 17.7 m below ground level.

The test began with a pumping rate of 150 cu m/d, but after 300 minutes the rate intermittently fell to 120 cu m/d, when demand for the water changed the discharge rate. From Figure 12, the plot of drawdown on semi-logarithmic scale, shows a curve with three segments, which increase in gradient with time, from 0.26 m to 0.74 m per log cycle. After 300 minutes, the drawdown readings became very erratic.

Using a worst-case scenario with the late time data, the trend follows a gradient of about 1.6 m per log cycle. In the medium term, i.e. 6 months to 2 years, this bore should be able to sustain a pumping rate of 120 cu m/d, assuming no interference from other bores or other deleterious hydraulic boundary effects are realised.

7.7 DISCUSSION OF TEST RESULTS

The 48-hour pumping tests give a reasonable basis for extrapolation of water-level trends in the long term, although under continued pumping the trends can change. Reduced rates of drawdown can result from recharge events and leakage from adjacent aquifer layers. Increased drawdown can be caused by hydraulic barrier boundaries or lower aquifer transmissivity/storage in the surrounding area.

Two of the tests show flattening drawdown trends which are favourable in suggesting some leakage into the aquifer being pumped. These are Bores BW2P and BW10P (in the case of BW2P, the leakage is very small.)

Four of the tests showed steepening trends which give less confidence in continuity of supply. Accordingly, the recommended pumping rates from these four bores - BW1P (250 cu m/d), BW6P (300 - 350 cu m/d), BW8P (250 - 300 cu m/d) and BP70 (120 cu m/d) are all lower than the rates for the pumping tests. If monitoring shows strong declines in water levels, the pumping rates might need to be further reduced.

More detailed evaluation of the extent and yield of the aquifers is not justified in a project of this nature because of the expense and the variability of results in fractured rock aquifers. It is more practical to have adequate standby capacity to cover at least the highest producing bore.

8.0 GROUNDWATER CHEMISTRY

Water samples were collected from all pumped bores after one hour, and at the end of testing. An additional sample (from the end of the test) was acidified with nitric acid and subsequently tested for metal elements. The samples were analysed by Rapley Wilkinson Laboratories for selected major and trace ions. Results are shown in Table 3.

The groundwater is relatively fresh, with laboratory salinities ranging from 150 to 480 milligrams per litre Total Dissolved Solids (TDS) (measured by conductivity). For drinking water, the usual limit is 1,000 mg/1.

Values of pH range from 6.15 to 7.30, indicating that the waters are essentially neutral to slightly acidic and are unlikely to be very corrosive.

Except for iron, the concentrations of the major ions analysed are all within acceptable limits for drinking water. Low concentrations of dissolved solids make the waters very suitable for processing gold ore. Magnesium, for example, has low concentrations which will reduce chemical dosing requirements.

Bores BW8P and BW10P have iron concentrations of less than 1 mg/1, which is the desired upper concentration limit. Bores BW1P, BW2P BS6P and BP70 range from 1.2 to 5.0 mg/1, which are all above the desired upper limits.

Arsenic concentrations were above the recommended limit for drinking water (0.05 mg/1, W.H.O) in three samples, with values of 0.08, 0.11, and 0.50 in bores BP 70, 1P, and 2P respectively (Table 3). Water from the mineralised zones should not be used for drinking.

Bore BW10P has a lower than acceptable pH value for drinking water. Bore BW8P, would be the suggested bore to be used as a drinking water source.
The relative concentrations of ions in the several samples are shown in Figure 13, in which the plotted values are milli-equivalents per litre. Notable features are:

1. Sodium, chloride, and sulphate values are unusually low in all the samples, compared with calcium, magnesium, and bicarbonate values. This is a feature of 'young' groundwater, and suggests favourable recharge conditions.

2. Magnesium values are higher than calcium in all samples, reflecting magnesium-rich aquifer material, i.e. greywacke and shale/schist.

TABLE 3

CHEMICAL ANALYSES OF WATER SAMPLES FROM PUMPING TESTS

BORE	BW1P	BW1P	BW2P	BW2P	BW6P	BW6P	BW8P	BW8P	BW10P	BW10P	BP70	BP70
DATE SAMPLED	28.10.88	30.10.88	10.11.88	12.11.88	31.10.88	2.11.88	14.11.88	16.11.88	3.11.88	9.11.88	13.11.88	14.11.88
						DETERMINATION mg/1						
Calcium	7	8	24	10	37	22	35	24	6	5	7	7
Magnesium	9	9	19	20	41	42	35	30	12	12	9	8
Sodium	28	28	34	35	84	83	58	60	18	18	12	11
Potassium	3	3	9	8	4	4	5	5	3	4	5	5
Total Iron	-	4	-	2.6	-	1.2	-	0.7	-	0.3	-	5.0
Manganese	-	0.11	-	0.25	-	0.20	-	0.39	-	0.069	-	0.185
Strontium	-	0.05	-	0.08	-	0.33	-	0.23	-	0.01	-	<0.01
Lead	-	0.01	-	0.03	-	0.01	-	0.005	-	0.01	-	0.02
Nickel	-	<0.002	-	0.002	-	0.002	-	0.002	-	0.002	-	0.007
Copper	-	<0.002	-	0.002	-	<0.002	-	<0.002	-	0.002	-	<0.002
Zinc	-	0.07	-	0.26	-	0.21	-	0.14	-	0.225	-	0.24
Cadmium	-	<0.002	-	<0.002	-	<0.002	-	<0.002	-	<0.002	-	<0.002
Bicarbonate	125	125	170	165	405	425	335	335	85	110	82	75
Sulphate	<5	<5	25	25	<5	<5	10	10	<5	<5	<5	<5
Chloride	70	85	50	60	115	115	105	115	45	55	45	55
Nitrate	0.	<0.1	<0.1	8.8*	<0.1	<0.1	<0.1	<0.1	<0.1	0.1	<0.1	<0.1
Arsenic	-	0.11	-	0.50	-	<0.05	-	<0.05	-	<0.05	-	0.08

continued ...

TABLE 3 (continued)

CHEMICAL ANALYSES OF WATER SAMPLES FROM PUMPING TESTS

BORE	BW1P	BW1P	BW2P	BW2P	BW6P	BW6P	BW8P	BW8P	BW10P	BW10P	BP70	BP70
DATE SAMPLED	28.10.88	30.10.88	10.11.88	12.11.88	31.10.88	2.11.88	14.11.88	16.11.88	3.11.88	9.11.88	13.11.88	14.11.88
						DETERMINATION mg/1						
Total Dissolved Solids By Conductivity												
(Labortory)	179	195	226	242	480	477	413	410	126	149	118	128
(Field)	102	–	151	–	302	–	254	–	84	–	–	–
Total Alkalnity	103	103	139	135	332	349	275	275	70	90	67	62
pH	7.30	6.65	6.65	7.05	7.00	6.80	6.80	6.95	6.95	6.15	6.20	7.20

* This sample was possibly contaminated with NO3 from addition of Nitric acid used to stabilise metal elements

9.0 FUTURE GROUNDWATER DEVELOPMENT

The groundwater supplies of 540,000 cu m/yr (1470 cu m/d) located and developed during the completed Stage I of the groundwater programme are contained predominantly in the shales and greywackes of the Tollis Formation. A belt of country running northerly through Robin Prospect is the most productive. It contains quartz veining in the Robin Prospect and covers the contact zone between the lower and middle units of the Tollis Formation.

Process water requirements according to present planning will be either 1×10^6 cu m/yr or 2×10^6 cu m/yr. Development of such supplies is proposed to be undertaken in 1989 as Stages II and III of the groundwater programme. The scope of works and summary of cost estimates are presented below.

9.1 STAGE II - ONE MILLION CUBIC METRES PER ANNUM

The additional water supplies required are 460,000 cu m/yr (1,260 cu m/d) not including standby capacity. It is prudent to have standby capacity equal at least to the highest producing bore, in the present case 350 cu m/d. Thus the total amount of groundwater to be proven is 1,600 cu m/d.

We propose that the Stage II groundwater search area extend to the north from Robin Prospect through mineral tenements EL 5175 and EL 5792 (Fig 14). We expect that the additional requirement of 1,600 cu m/d will be obtainable from five or six new production bores with an average pumping rate of about 300 cu m/d. To be conservative, we allow for six production bores, in the scope of work.

Locations of two of the production bores have already been decided: at BW 9 and the old Pacific camp. To locate the additional four, it is expected that eight exploration holes will be required. The programme to develop an additional 1,600 cu m/d, therefore, comprises: eight exploration holes
six production bores
six pumping tests.

Tentative sites for the drilling are shown in Figure 14.

9.1.1 Cost Estimates

(1) Drilling Contractor

The cost estimate to complete the above works is based on the actual rates from the first programme, using the same drilling contractor.

Drilling of 8 exploration holes	
Mobilisation	3,000
8 Exploration holes @ average $4,500 per hole	36,000

Drilling, construction, and completion of 6 production bores @ $12,500 per bore	75,000
Conduct pumping tests on 6 production bores at $4,000 per test	24,000
Estimate of Contractors costs	$138,000

(2) Cost Summary

Contractor costs	138,000
Consultants costs	23,500
Total	$161,500

9.2 STAGE III - TWO MILLION CUBIC METRES PER ANNUM

The scope of work for the 2×10^6 cu m/yr scenario is based on the premise that the 1×10^6 cu m/yr programme has already been successfully implemented.

We expect that there will not be many sites available in the northern area for further construction of production bores. Therefore, it is assumed that the additional 1×10^6 cu m/yr (2,740 cu m/d) will be developed in the southern area, between the present bores and the Edith River. This will necessitate an agreement being reached with the owners of EL 4823.

The programme is costed out on the basis of nine production bores and fifteen exploration holes. Results are expected to be somewhat better than in the northern area because the area is lower in the drainage basin, and might draw recharge from Stow Creek and Edith River.

Sites for the exploration bores are not shown in Figure 14; they will be selected at a later date.

9.2.1 Cost Estimates

(1) Drilling Contractor

Presented below are estimates based on the previously completed programme.

Drilling of 15 exploration holes	
Mobilisation	3,000
15 exploration holes @ $4,500/hole	67,500

Construction and development of 9 production bores @ $12,500 per bore	112,500
Test pumping of 9 production bores	36,000
Estimate of Contractors costs	$219,000

(2) Cost Summary

Contractors costs	219,000
Consultants costs	37,000
	$256,000

9.3 SUMMARY OF PROGRAMME

To develop a groundwater supply of 1×10^6 cu m/yr for the Mount Todd project, an additional eight exploration holes and six production bores are estimated to be required. It is proposed that these be drilled on EL's 5175 and 5742 to the north of Robin Prospect. The estimated cost is $162,000 for this Stage II of the groundwater development.

To develop a groundwater supply of 2×10^6 cu m/yr, the additional 1×10^6 is proposed to be developed from an area that is the southern extension of the present borefield, in the vicinity of the Edith River. This Stage III is estimated to comprise fifteen exploration holes and nine production bores, and to cost $256,000.

There is an alternative to the above programme, i.e. that the Stage II drilling is located to the south rather than the north. This has not been costed herein because it presently appears that access will first be obtained to the northern ground.

10.0 SUMMARY AND CONCLUSIONS

Five production water bores have been constructed to supply process and domestic water for the Batman Gold Project. They are completed to depths of 59 to 71 m with UPVC casing of 155 mm internal diameter. From analyses of 48-hour pumping tests, the bores are recommended to be pumped at rates of 120 to 300 cu m/d totalling 1470 cu m/d. Recommended rates, pump settings, and other data are listed in Table 4. With close monitoring, pumping rates for bores BW2P, BW6P and BW8P could be slightly increased from those given, but should excessive drawdowns occur, then the rates would need to be reduced.

With the given pump settings, the pump inlets will be within the slotted casing, and in one bore it will be within the main aquifer interval. If electric submersible pumps are to be used, that in bore BW10P should be fitted with a shroud or bleeder tube to direct water over the motor for cooling purposes.

Extrapolation of drawdown trends from the pumping tests gives good confidence that bores BW2P and BW10P will maintain supply, while the rates for BW1P, BW6P, BW8P and BP70 might have to be reduced in the long term. Monitoring of water levels under pumping conditions will provide advance information.

The water is relatively fresh, with salinities in the range 150 to 480 milligrams per litre Total Dissolved Solids. From the chemical analyses, we recommend that Bore BW8P is the most suitable source of water for domestic use.

TABLE 4

Summary of Water Supply

Bore	Depth (m)	Static Water Level (mbgl)	Pump inlet setting (m)	Recommended Pumping Rate (cu m/d)	Expected long term Water Level (m)	Salinity (mg/l TDS)
BW1P	62.6	18.5	48	250	44	200
BW2P	70.9	26.0	55	300	34	240
BW6P	70.0	18.8	50	300	33	480
BW8P	58.6	23.7	50	250	37	410
BW10P	70.0	22.7	65	250	40	150
BP70	60.0	17.7	35	120	*	130
				1,470		

* Data insufficient for a reliable estimate to be made.

The completed programme established 540,000 cu m/yr of process water. Stage II is proposed to develop an additional 580,000 cu m/yr to give the 1×10^6 cu m/yr requirements plus standby capacity. Exploration and the construction of six production bores is estimated to cost $162,000. The suggested location is to the north of Robin Prospect, on EL's 5175 and 5792.

Stage III is proposed to develop an additional 1×10^6 cu m/yr. Exploration and the construction of nine production bores is estimated to cost $256,000. The suggested location is to the south of Robin Prospect, on EL 4823.

DATED 20TH NOVEMBER 1988

ROCKWATER PTY LTD

G Brophy

G BROPHY
Groundwater Technologist

J Passmore

J R PASSMORE
Principal Hydrogeologist

APPENDIX I

BORE COMPLETION DATA
OBSERVATION BORE

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW 1
LOCATION:	Mine Grid 9896 N 9805E
HEIGHT OF COLLAR ABOVE GROUND:	0.23 m
STATUS:	Observation Bore
DATE COMPLETED:	17/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	65 m
DIAMETER DRILLED:	0 - 5.5 m 203 mm Dia Hole, Hammer 5.5 - 65 m 190 mm Dia Hole, Hammer
CASING:	+0.23-5.50 m 155 mm ID 168 mm OD UPVC Surface Casing +0.23-63.84m 50 mm ID 60 mm OD UPVC
SLOTS:	27.84 - 63.84 m
WATER LEVEL:	18.53 m Below Ground Level
AIRLIFT YIELD:	576 cu m/d
WATER SALINITY:	107 mg/l (By Conductivity)

LITHOLOGY:

(m)		
0 - 18	GREYWACKE -	Grey brown highly weathered fine even grained, becoming fresh
18 - 27	GREYWACKE -	Grey slightly weathered fine even grained
27 - 64	GREYWACKE -	Black fresh fine even grained

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW 2
LOCATION:	Mine Grid 10492N 9684E
HEIGHT OF COLLAR ABOVE GROUND:	0.27 m
STATUS:	Observation Bore
DATE COMPLETED:	18/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	77 m
DIAMETER DRILLED:	0 - 77 m 190 mm Dia Hole, Hammer
CASING:	+0.22-5.78 m 155 mm ID 168 mm OD UPVC Surface Casing +0.27-76.73m 50 mm ID 60 mm OD UPVC
SLOTS:	40.73 - 76.73 m
WATER LEVEL:	25.99 m Below Ground Level
AIRLIFT YIELD:	376 cu m/d
WATER SALINITY:	153 mg/1 (By Conductivity)

LITHOLOGY:

(m)		
0 - 6	GREYWACKE	- Light brown highly weathered fine even grained, well foliated
6 - 30	GREYWACKE	- As above moderately to slightly weathered
30 - 76.9	GREYWACKE	- Black grey, fresh fine even grained

APPENDIX I
BORE COMPLETION DATA
EXPLORATION HOLE

BORE:	BW 3
LOCATION:	Mine Grid 9035N 9130E
HEIGHT OF COLLAR ABOVE GROUND:	-
STATUS:	Exploration Hole - Abandoned
DATE COMPLETED:	19/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	71 m
DIAMETER DRILLED:	0 - 71 m 190 mm Dia Hole, Hammer
CASING:	-
SLOTS:	-
WATER LEVEL:	-
AIRLIFT YIELD:	Dry
WATER SALINITY:	-

LITHOLOGY:

(m)		
0 - 3	ALLUVIUM -	Light brown fine grained silt
3 - 27	GREYWACKE -	Light brown moderately to slightly weathered fine even grained
27 - 71	GREYWACKE -	Dark grey, black, fresh fine even grained

APPENDIX I
BORE COMPLETION DATA
EXPLORATION HOLE

BORE:	BW 4
LOCATION:	Mine Grid 8910N 8752E
HEIGHT OF COLLAR ABOVE GROUND:	-
STATUS:	Exploration Hole - Abandoned
DATE COMPLETED:	19/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	52.5 m
DIAMETER DRILLED:	0 - 52.5 m 190 mm Dia Hole, Hammer
CASING:	-
SLOTS:	-
WATER LEVEL:	-
AIRLIFT YIELD:	Dry
WATER SALINITY:	-

LITHOLOGY:

(m)		
0 - 3	ALLUVIUM/WEATHERED HORNFELS	- Light brown fine grain silt and light brown highly to moderately weathered hornfels
3 - 52.5	HORNFELS	- Dark grey black fresh fine even grained

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW 5
LOCATION:	Mine Grid 10600N 9000E
HEIGHT OF COLLAR ABOVE GROUND:	0.63 m
STATUS:	Observation Bore
DATE COMPLETED:	20/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	65 m
DIAMETER DRILLED:	0 - 65 m 190 mm Dia Hole, Hammer
CASING:	+0.38-5.62 m 155 mm ID 168 mm OD UPVC Surface Casing +0.63-65.39m 50 mm ID 60 mm OD UPVC
SLOTS:	41.39 - 65.39 m
WATER LEVEL:	17.64 m Below Ground Level
AIRLIFT YIELD:	40 cu m/d
WATER SALINITY:	105 mg/1 (By Conductivity)

LITHOLOGY:

(m)		
0 - 3	ALLUVIUM	- Grey brown, silt and clay
3 - 24	GREYWACKE	- Brown to dark grey. Moderately to slightly weathered
24 - 30	GREYWACKE (HORNFELSED)	- Grey to dark grey, slightly weathered fine even grained
30 - 64.7	GREYWACKE (HORNFELSED)	- Dark grey black fresh fine even grained

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW 6
LOCATION:	Mine Grid 8955N 10365E
HEIGHT OF COLLAR ABOVE GROUND:	0.48 m
STATUS:	Observation Bore
DATE COMPLETED:	20/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	71 m
DIAMETER DRILLED:	0 - 71 m 193 mm Dia Hole, Hammer
CASING:	+0.48-5.52 m 155 mm ID 168 mm OD UPVC Surface Casing +0.48-70.06m 50 mm ID 60 mm OD UPVC
SLOTS:	28.06 - 70.06
WATER LEVEL:	18.79 m Below Ground level
AIRLIFT YIELD:	258 cu m/d
WATER SALINITY:	303 mg/1 (By Conductivity)

LITHOLOGY:
(m)

0 - 3	ALLUVIUM	- Grey brown silt and clay
3 - 6	GREYWACKE	- Light grey clay after completly weathered greywacke
6 - 21	GREYWACKE	- Light to dark grey moderately to slightly weathered fine even grained schistose
21 - 30	GREYWACKE	- Dark grey as above
30 - 70.8	GREYWACKE	- Dark grey black fresh fine to even grained with white vein quartz from 33 to 60 metres

APPENDIX I
BORE COMPLETION DATA
EXPLORATION HOLE

BORE:	BW 7
LOCATION:	Mine Grid 9450N 11100E
HEIGHT OF COLLAR ABOVE GROUND:	-
STATUS:	Exploration Hole - Abandoned
DATE COMPLETED:	21/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	36 m
DIAMETER DRILLED:	0 - 5.8 m 190 mm Dia Hole, Hammer 5.8 - 36 m 152 mm Dia Hole, Hammer
CASING:	-
SLOTS:	-
WATER LEVEL:	-
AIRLIFT YIELD:	Dry
WATER SALINITY:	-

LITHOLOGY:
(m)

0 - 12	HORNFELS	- Light brown moderately weathered fine even grained
12 - 18	HORNFELS	- Grey brown slightly weathered with minor ferruginous stain on joints
18 - 36	HORNFELS	- Dark grey, black, fresh, fine even grained

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW 8
LOCATION:	Mine Grid 9598N 10253E
HEIGHT OF COLLAR ABOVE GROUND:	0.55 m
STATUS:	Observation Bore
DATE COMPLETED:	22/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	68 m
DIAMETER DRILLED:	0 - 5.5 m 190 mm Dia Hole, Hammer 5.5- 68 m 152 mm Dia Hole, Hammer
CASING:	+0.55-5.45 m 155 mm ID 168 mm OD UPVC Surface Casing +0.55-37.16m 50 mm ID 60 mm OD UPVC
SLOTS:	1.16 - 37.16 m
WATER LEVEL:	23.70 m Below Ground Level
AIRLIFT YIELD:	216 cu m/d
WATER SALINITY:	272 mg/1 (By Conductivity)

LITHOLOGY:
(m)

0 - 27	GREYWACKE	- Light brown completely to lightly weathered - clay
27 - 39	GREYWACKE	- Dark grey slightly weathered fine even grained
39 - 67.7	GREYWACKE	- Dark grey black fresh fine even grained with minor quartz veins

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW 9
LOCATION:	Mine Grid 10146 N 10200E
HEIGHT OF COLLAR ABOVE GROUND:	0.60 m
STATUS:	Observation Bore
DATE COMPLETED:	23/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	65 m
DIAMETER DRILLED:	0 - 65 m 192 mm Dia Hole, Hammer
CASING:	+0.35-5.65 m 155 mm ID 168 mm OD UPVC Surface Casing +0.60-64.35m 50 mm ID 60 mm OD UPVC
SLOTS:	34.55 - 64.55 m
WATER LEVEL:	23.76 m Below Ground Level
AIRLIFT YIELD:	192 cu m/d
WATER SALINITY:	256 mg/1 (By Conductivity)

LITHOLOGY:
(m)

0 - 9	GREYWACKE	- Light brown grey, fine grained highly weathered
9 - 27	GREYWACKE	- As above becomming fresher
27 - 64.4	GREYWACKE	- Black fresh fine even grained

APPENDIX I
BORE COMPLETION DATA
OBSERVATION BORE

BORE:	BW10
LOCATION:	Mine Grid 9163N 9960E
HEIGHT OF COLLAR ABOVE GROUND:	0.54 m
STATUS:	Observation Bore
DATE COMPLETED:	23/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	71 m
DIAMETER DRILLED:	0 - 71 m 192 mm Dia Hole, Hammer
CASING:	+0.54-5.46 m 155 mm ID 168 mm OD UPVC Surface Casing +0.54-69.0 m 50 mm ID 60 mm OD UPVC
SLOTS:	33 - 69 m
WATER LEVEL:	23.16 m Below Ground Level
AIRLIFT YIELD:	288 cu m/d
WATER SALINITY:	84 mg/1 (By Conductivity)

LITHOLOGY:
(m)

0 - 15	CLAY	-	Light brown clay dry after competely weathered greywacke
15 - 30	GREYWACKE	-	As above but highly to slightly weathered
30 - 70.8	GREYWACKE	-	Black, fresh, fine even grained

APPENDIX I
BORE COMPLETION DATA
EXPLORATION HOLE

BORE:	BW13
LOCATION:	Mine Grid 10505N 8351E
HEIGHT OF COLLAR ABOVE GROUND:	0.26 m
STATUS:	Exploration Hole - abandoned.
DATE COMPLETED:	7/11/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer
DEPTH DRILLED:	58 m
DIAMETER DRILLED:	0 - 38 m 200 mm Dia Hole, Hammer
CASING:	+0.26-1.24 m 206 mm ID 219 mm OD Steel Surface Casing
SLOTS:	-
WATER LEVEL:	3.0 m Below Ground Level
AIRLIFT YIELD:	54 cu m/d
WATER SALINITY:	137 mg/1 (By Conductivity)

LITHOLOGY:
(m)

0 - 3	GREYWACKE -	Grey black. Moderatly weathered bands. Moderatly fractured
3 - 15	GREYWACKE -	As above but well fractured. Fine grained, minor limonite staining on fracture planes
15 - 21	GREYWACKE -	Black Blue. Minor Fracturing, fresh, becomming harder
21 - 30	SILT STONE -	Brown black non fractured, very fine grained glassy texture. Monor quartz and pyrite
30 - 33	AS ABOVE -	Moderatly fractured, increasing quartz
33 - 42	GREYWACKE -	Blue black. Moderatly fractured, minor pyrite and quartz, fresh, moderatly hard
42 - 58	GREYWACKE/QUARTZ -	As above minor quart veins. massive non-fractured. Less quartz with depth. Minor pyrite

APPENDIX II

BORE COMPLETION DATA
PRODUCTION BORE

APPENDIX II
BORE COMPLETION DATA
PRODUCTION BORE

BORE:	BW 1P
LOCATION:	Mine Grid 9906N 9806E
HEIGHT OF COLLAR ABOVE GROUND:	0.10 m
STATUS:	Production Bore
DATE COMPLETED:	25/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer, Air Rotary
DEPTH DRILLED:	62 m
DIAMETER DRILLED:	0 - 5.5 m 311 mm Dia Hole, Roller Bit 5.5 - 62 m 254 mm Dia Hole, Hammer
CASING:	+0.10-5.75 m 260 mm ID 272 mm OD Steel Surface Casing +0.10-61.60m 155 mm ID 168 mm OD UPVC Class 9
SLOTS INTERVAL:	26.60 - 61.60 m
INTERNAL DIAMETER OF PUMP HOUSING:	155 m
WATER LEVEL:	18.66 m Below Ground Level
AIRLIFT YIELD:	432 cu m/d
PUMPING TEST:	1. Step Rate Test — 4 x 1 Hr Steps at Rates of 200, 300, 400 and 500 cu m/d 2. Constant Rate Test — 48 Hrs at a Constant Rate of 350 cu m/d. Final Drawdown 17.15 m
RECOMMENDED PUMPING RATE:	250 cu m/d
RECOMMENDED PUMP INLET SETTING:	45 m
PUMPING WATER LEVEL:	44 m
WATER SALINITY:	Field Determination - 102 mg/1 TDS (By Conductivity) Laboratory Determination - 195 mg/1 TDS (By Conductivity).

APPENDIX II
BORE COMPLETION DATA
PRODUCTION BORE

BORE:	BW 2P
LOCATION:	Mine Grid 10482N 9689E
HEIGHT OF COLLAR ABOVE GROUND:	0.48 m
STATUS:	Produciton Bore
DATE COMPLETED:	5/11/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer, Air Rotary
DEPTH DRILLED:	72 m
DIAMETER DRILLED:	0 - 5.5 m 311 mm Dia Hole, Roller Bit 5.5 - 72m 254 mm Dia Hole, Hammer
CASING:	+0.48-5.40 m 250 mm ID 272 mm OD Steel Surface Casing +0.48-70.90m 155 mm ID 168 mm OD UPVC Class 9
SLOTS INTERVAL:	34.90 - 70.90 m
INTERNAL DIAMETER OF PUMP HOUSING:	155 mm
WATER LEVEL:	26.65 m Below Ground Level
AIRLIFT YIELD:	576 cu m/d
PUMPING TEST:	1. Step Rate Test — 4 x 1 Hr Steps at Rates of 76, 250, 300 and 400 cu m/d 2. Constant Rate Test — 48 Hrs at a constant Rate of 300 cu m/d Final Drawdown 5.89 m
RECOMMENDED PUMPING RATE:	300 or 350 cu m/d
RECOMMENDED PUMP INLET SETTING:	55 m
PUMPING WATER LEVEL:	34 m
WATER SALINITY:	Field Determination - 151 mg/1 TDS (By Conductivity) Laboratory Determination - 242 mg/1 TDS (By Conductivity)

APPENDIX II
BORE COMPLETION DATA
PRODUCTION BORE

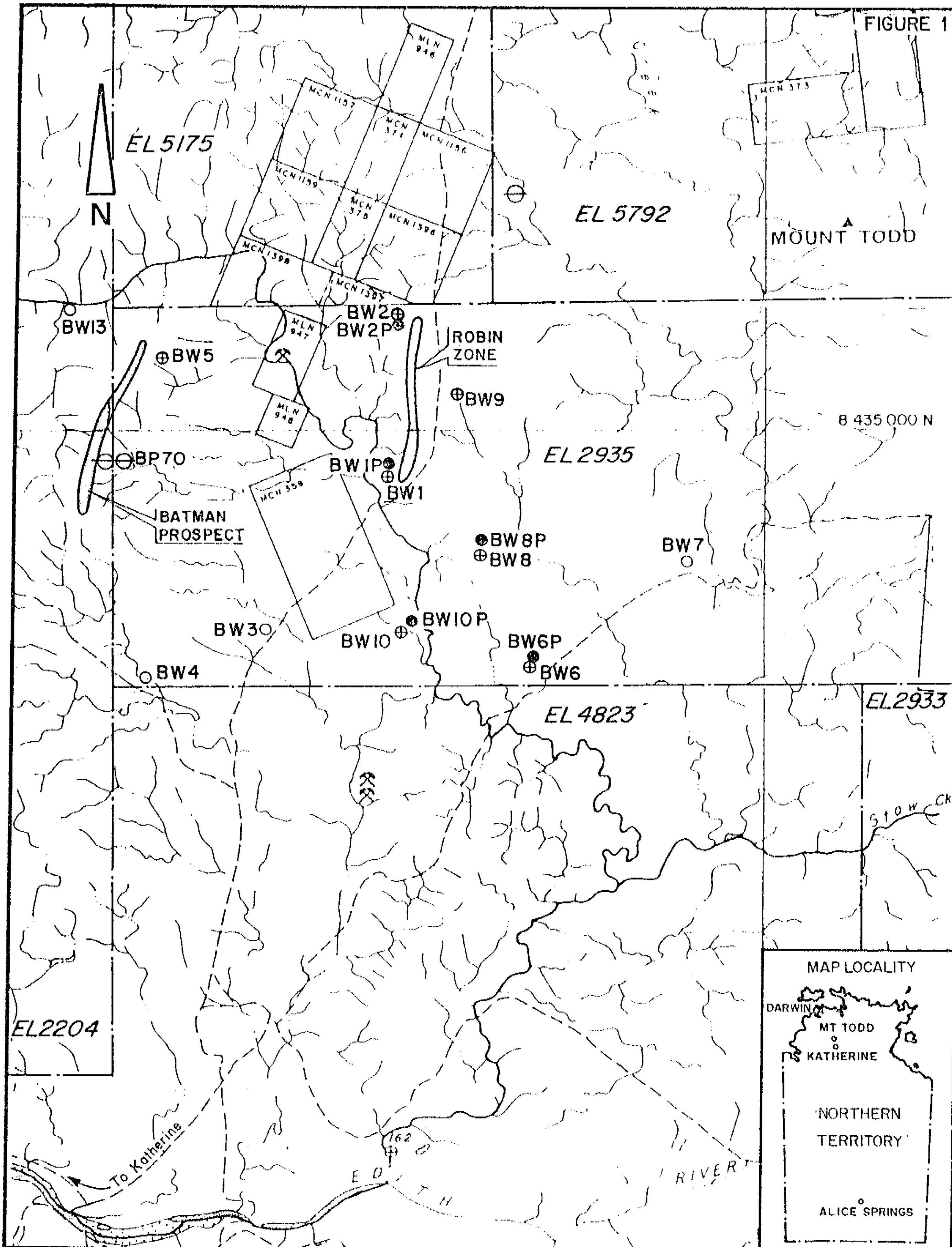
BORE:	BW 6P
LOCATION:	Mine Grid 8984N 10362E
HEIGHT OF COLLAR ABOVE GROUND:	0.30 m
STATUS:	Production Bore
DATE COMPLETED:	26/10/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer, Air Rotary
DEPTH DRILLED:	70 m
DIAMETER DRILLED:	0 - 2.5 m 311 mm Dia Hole, Roller Bit 2.5-70 m 254 mm Dia Hole, Hammer
CASING:	+0.30-2.70 m 260 mm ID 272 mm OD Steel Surface Casing +0.30-69.70m 155 mm ID 168 mm OD UPVC Class 9
SLOTS INTERVAL:	27.70 - 69.70 m
INTERNAL DIAMETER OF PUMP HOUSING:	155 mm
WATER LEVEL:	18.79 m Below Ground Level
AIRLIFT YIELD:	375 cu m/d
PUMPING TEST:	1. Step Rate Test — 4 x 1 Hr Steps at Rates of 150, 200, 300 and 400 cu m/d 2. Constant Rate Test — 48 Hrs at Constant Rate of 350 cu m/d. Final Drawdown 8.83 m
RECOMMENDED PUMPING RATE:	300 or 350 cu m/d
RECOMMENDED PUMP INLET SETTING:	50 m
PUMPING WATER LEVEL:	33 m
WATER SALINITY:	Field Determination - 302 mg/1 TDS (By Conductivity) Laboratory Determination - 477 mg/1 TDS (By Conductivity)

APPENDIX II
BORE COMPLETION DATA
PRODUCTION BORE

BORE:	BW 8P
LOCATION:	Mine Grid 9615N 10266E
HEIGHT OF COLLAR ABOVE GROUND:	0.80 m
STATUS:	Production Bore
DATE COMPLETED:	6/11/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer, Air Rotary
DEPTH DRILLED:	60 m
DIAMETER DRILLED:	0 - 28 m 311 mm Dia Hole, Roller Bit 28 -60 m 254 mm Dia Hole, Hammer
CASING:	+0.80-27.84 m 260 mm ID 272 mm OD Steel Surface Casing +0.80-57.83 m 155 mm ID 168 mm OD UPVC Class 9
SLOTS INTERVAL:	27.83 - 57.85 m
INTERNAL DIAMETER OF PUMP HOUSING:	155 mm
WATER LEVEL:	23.70 m Below Ground Level
AIRLIFT YIELD:	864 cu m/d
PUMPING TEST:	1. Step Rate Test — 4 x 1 Hr Steps at Rates of 200, 400, 600 and 800 cu m/d 2. Constant Rate Test — 48 Hrs at Constant Rate of 400 cu m/d Final Drawdown 7.78 m
RECOMMENDED PUMPING RATE:	250 or 300 cu m/d
RECOMMENDED PUMP INLET SETTING:	50 m
PUMPING WATER LEVEL:	37 m
WATER SALINITY:	Field Determination - 254 mg/1 TDS (By Conductivity) Laboratory Determination - 410 mg/1 TDS (By Conductivity)

APPENDIX II
BORE COMPLETION DATA
PRODUCTION BORE

BORE:	BW10P
LOCATION:	Mine Grid 9162N 9982E
HEIGHT OF COLLAR ABOVE GROUND:	0.15 m
STATUS:	Production Bore
DATE COMPLETED:	1/11/88
DRILLING CONTRACTOR:	Gorey and Cole Drillers
DRILLING RIG:	Ingersoll Rand TH-60
DRILLING METHOD:	Air Hammer, Air Rotary
DEPTH DRILLED:	71.5 m
DIAMETER DRILLED:	0 - 33 m 311 mm Dia Hole, Roller Bit 33- 71.5 m 254 mm Dia Hole, Hammer
CASING:	+0.33-32.09 m 260 mm ID 272 mm OD Steel Surface Casing +0.25-69.90 m 155 mm ID 168 mm OD UPVC Class 9
SLOTS INTERVAL:	29.90 - 69.90 m
INTERNAL DIAMETER OF PUMP HOUSING:	155 mm
WATER LEVEL:	22.65 m Below Ground Level
AIRLIFT YIELD:	576 cu m/d
PUMPING TEST:	1. Step Rate Test — 4 x 1 Hr Steps at Rates of 200, 250, 300 and 350 cu m/d 2. Constant Rate Test — 48 Hrs at Constant Rate of 250 cu m/d. Final Drawdown 19.61 m
RECOMMENDED PUMPING RATE:	250 cu m/d
RECOMMENDED PUMP INLET SETTING:	65 m
PUMPING WATER LEVEL:	62 m
WATER SALINITY:	Field Determination - 84 mg/1 TDS (By Conductivity) Laboratory Determination - 149 mg/1 TDS (By Conductivity)



LEGEND

○ Exploration bore (abandoned)

Production bore

Existing water bore

⊕ Observation bore.

0 — 1km

BILLITON AUSTRALIA GOLD PTY. LTD.

MOUNT TODD PROJECT

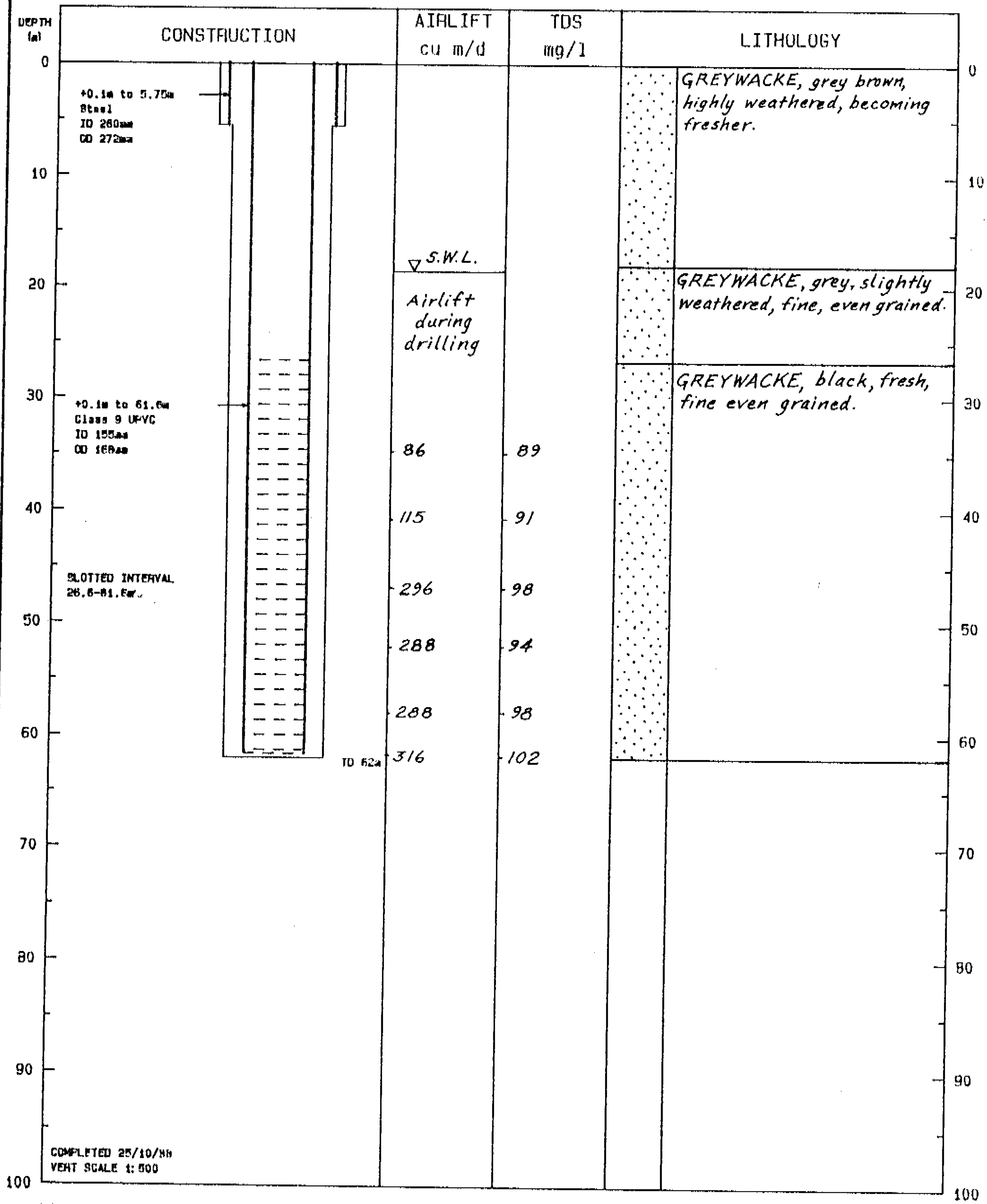
BORE LOCALITY MAP

November 1988

128-1/88/2-1

BILLITON AUST PTY LTD
BATMAN WATER SUPPLY
COMPOSITE BORE LOG, BW1P

Figure 2

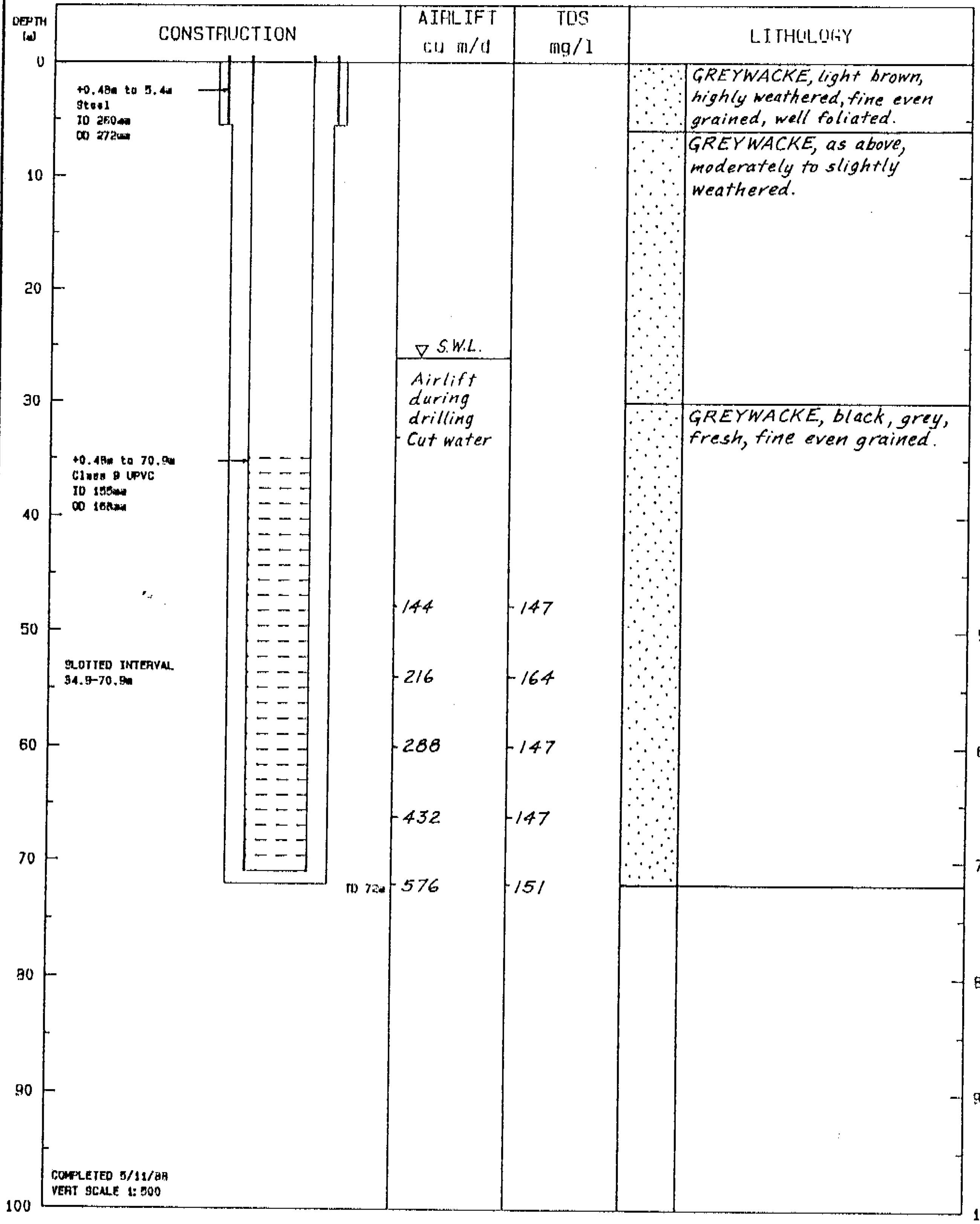


DRILLING DETAILS : 0 - 5.5m, 311mm
5.5 - 62m, 254mm

128.1/88/2-2

BILLITON AUST PTY LTD
BATMAN WATER SUPPLY
COMPOSITE BORE LOG, BW2P

Figure 3

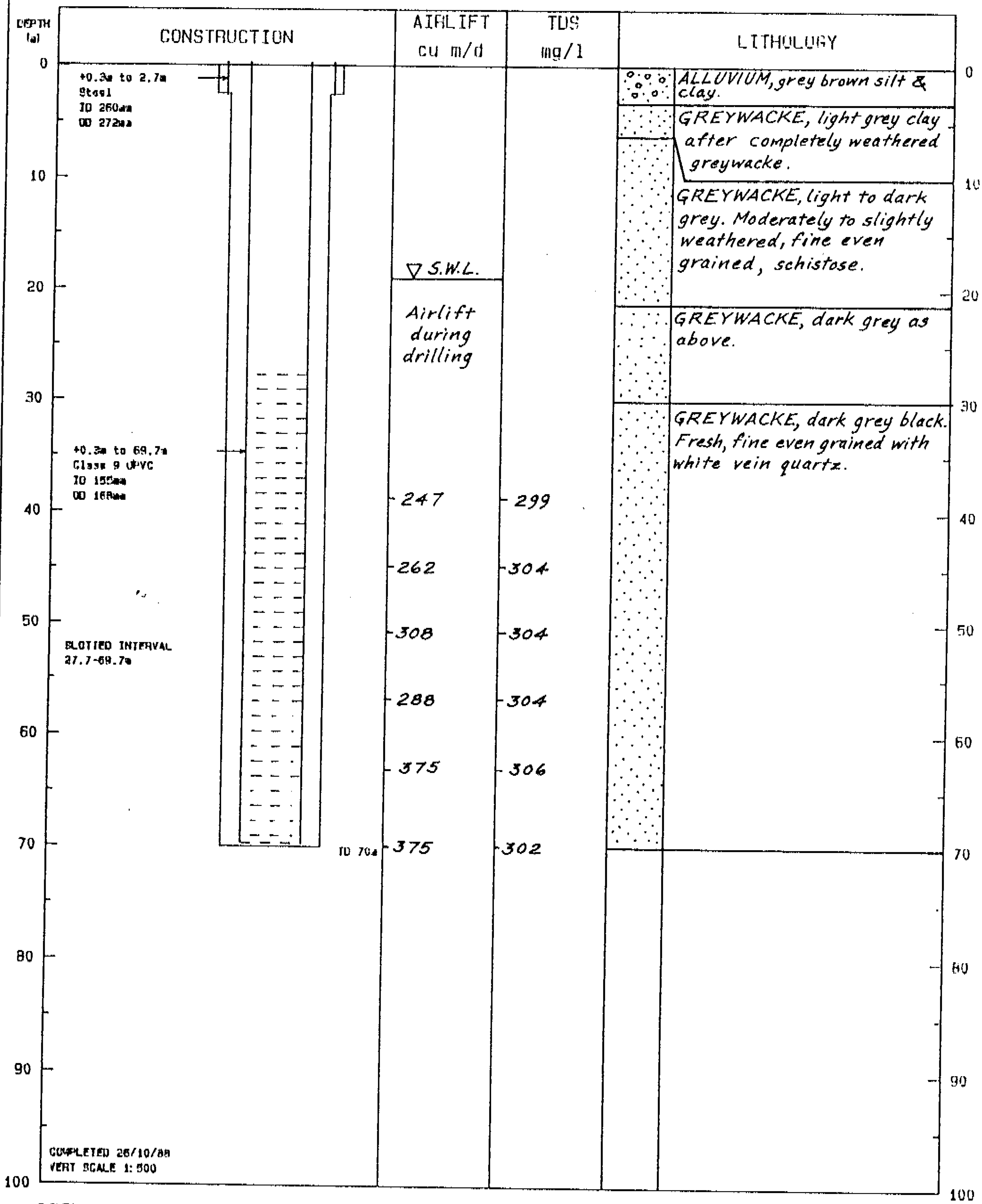


DRILLING DETAILS : 0 - 5.5m, 311mm
5.5 - 72m, 254mm

128.1/88/2-3

BILLITON AUST PTY LTD
BATMAN WATER SUPPLY
COMPOSITE BORE LOG, BW6P

Figure 4

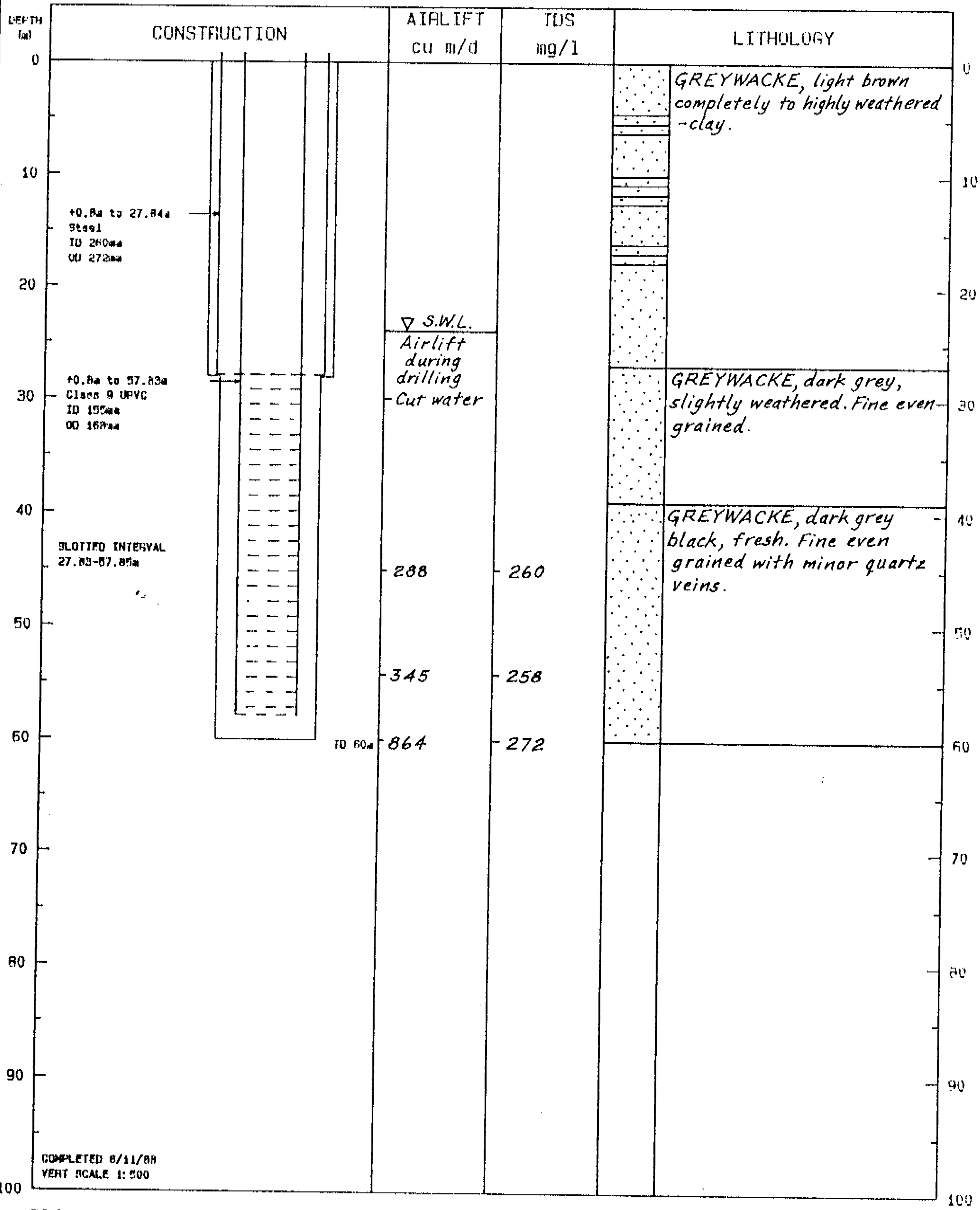


DRILLING DETAILS : 0 - 2.5m, 311mm
2.5 - 70m, 254mm

128.1/88/2-4

BILLITON AUST PTY LTD
BATMAN WATER SUPPLY
COMPOSITE BORE LOG, BW8P

Figure 5

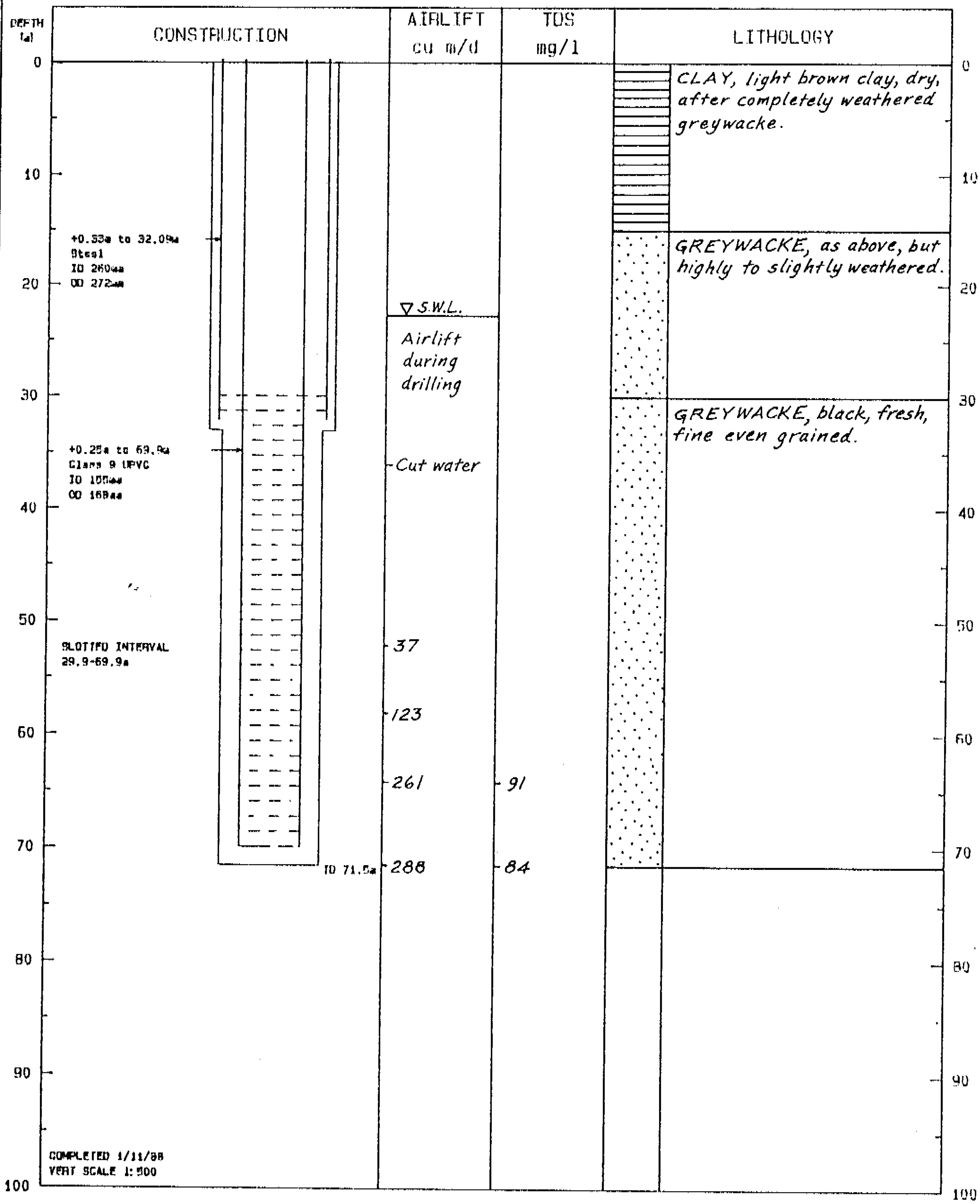


DRILLING DETAILS : 0 - 28m, 311mm
28 - 60m, 254mm

128.1/88/2-5

BILLITON AUST PTY LTD
BATMAN WATER SUPPLY
COMPOSITE BORE LOG, BW10P

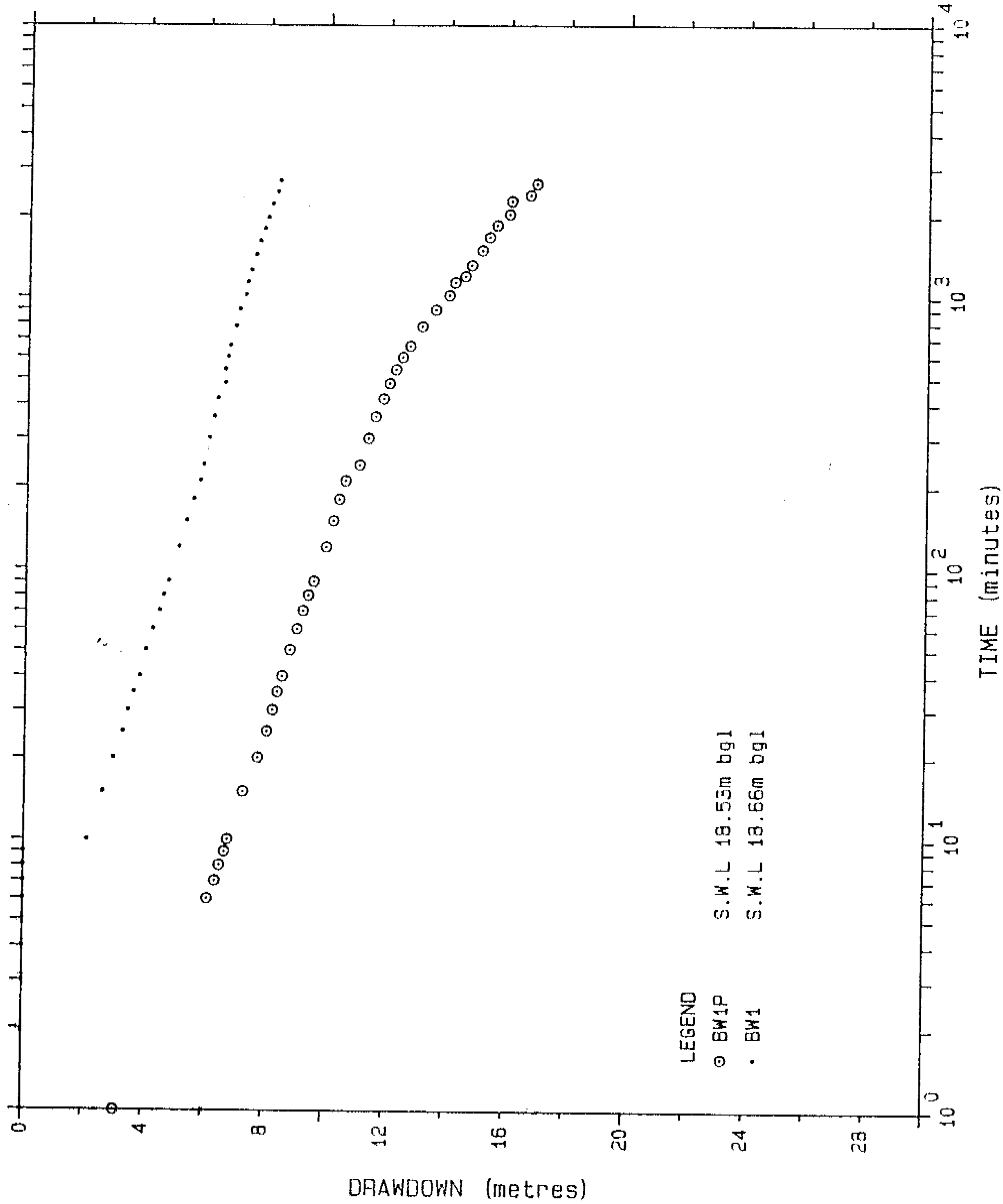
Figure 6



DRILLING DETAILS : 0 - 33m, 311mm
33 - 71.5m, 254mm

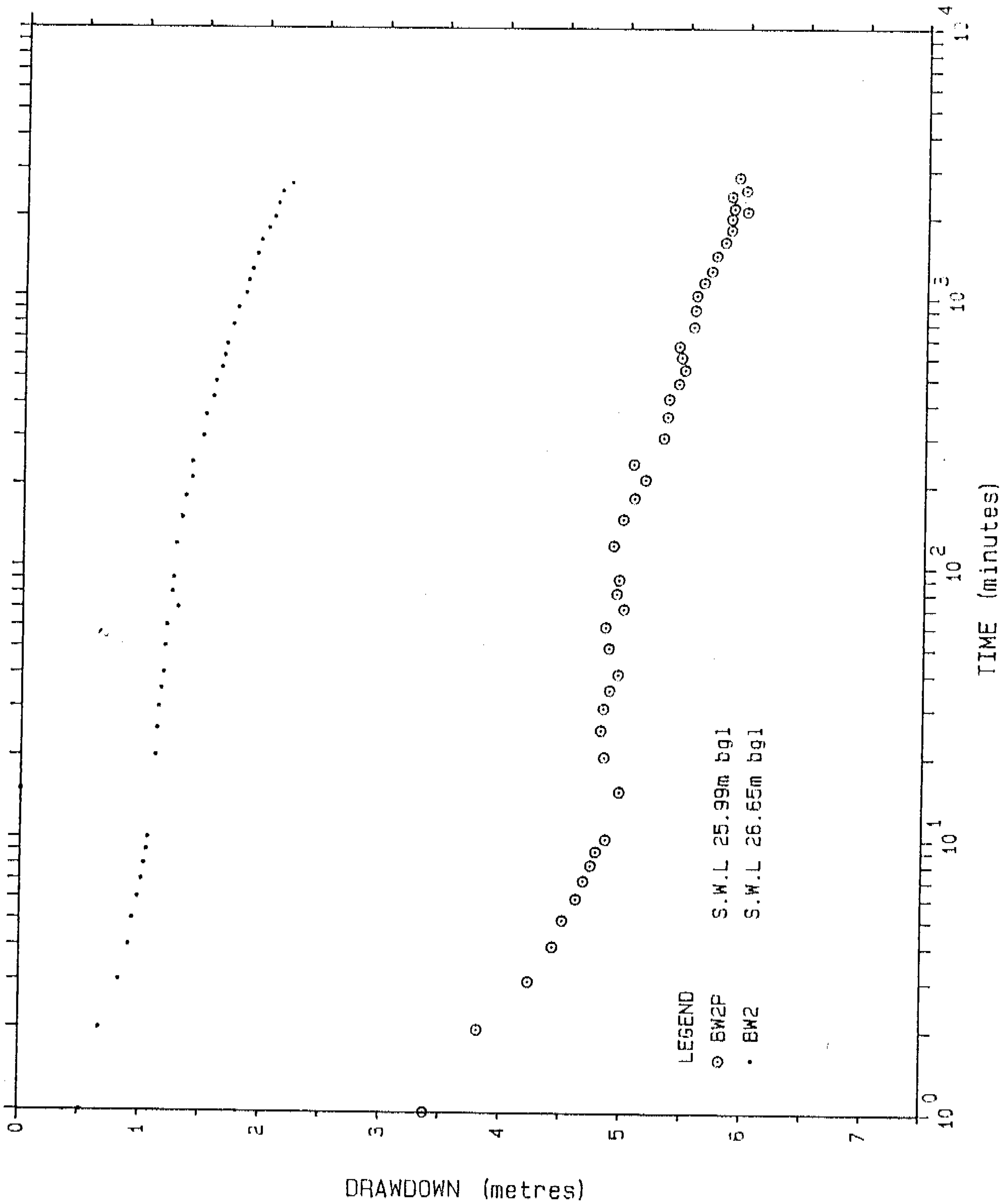
128.1/88/2-6

Figure 7



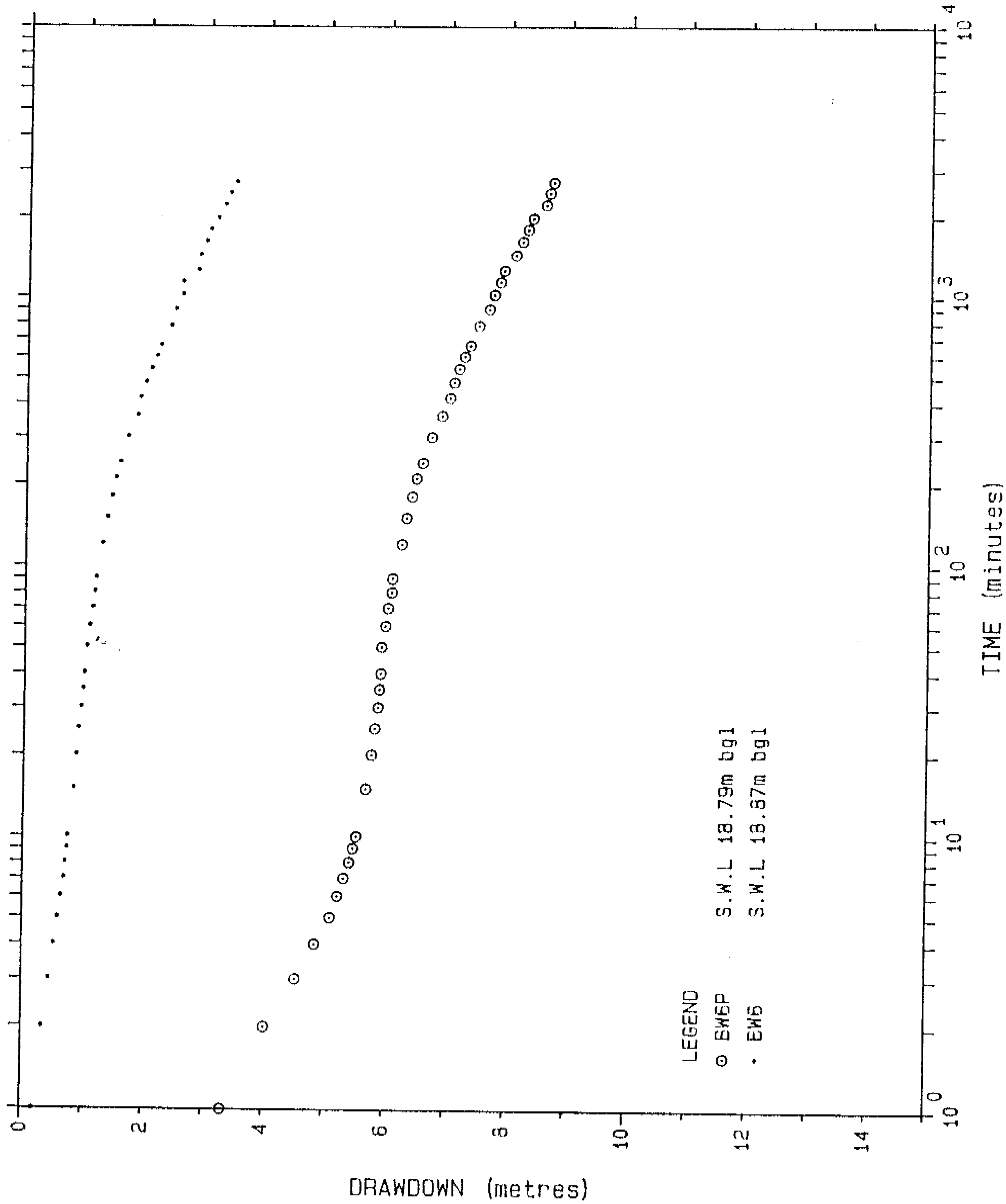
Client : BILLITON AUST PTY LTD	BORE BW1P CONSTANT RATE PUMPING TEST
Project : MT TODD WATER SUPPLY	DRAWDOWN IN PUMPED AND OBSERVATION BORES
Date : DECEMBER 1988	Pumping Rate of BW1P 350 m^3/day
Dwg. No.: 128.1/88/2-7	Commenced test on 28/10/88

Figure 8



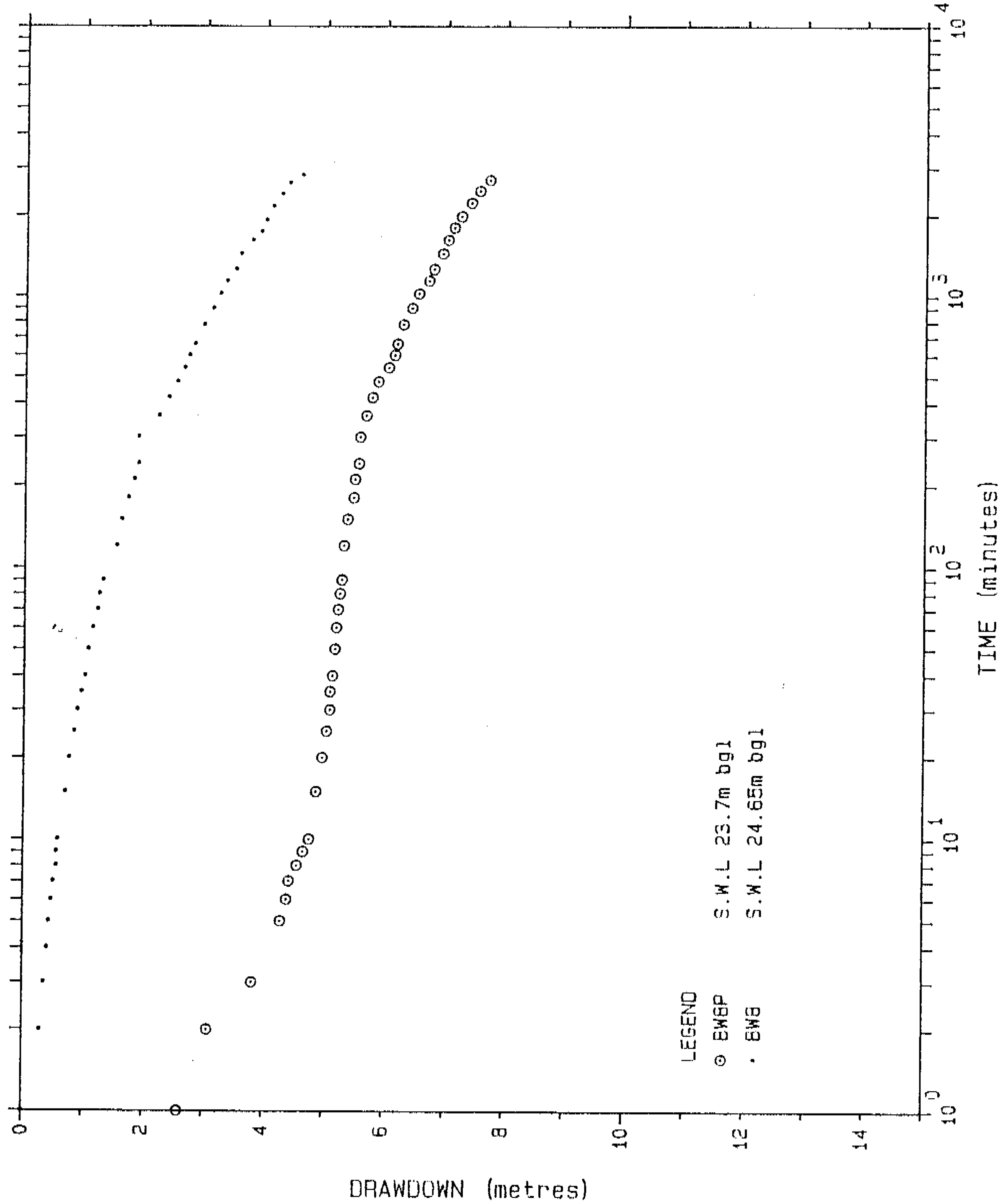
Client : BILLITON AUST PTY LTD	BORE BW2P CONSTANT RATE PUMPING TEST
Project : MT TODD WATER SUPPLY	DRAWDOWN IN PUMPED AND OBSERVATION BORES
Date : DECEMBER 1988	Pumping Rate of BW2P 300 m^3/day
Dwg. No.: 128.1/88/2-8	Commenced test on 10/11/88

Figure 9



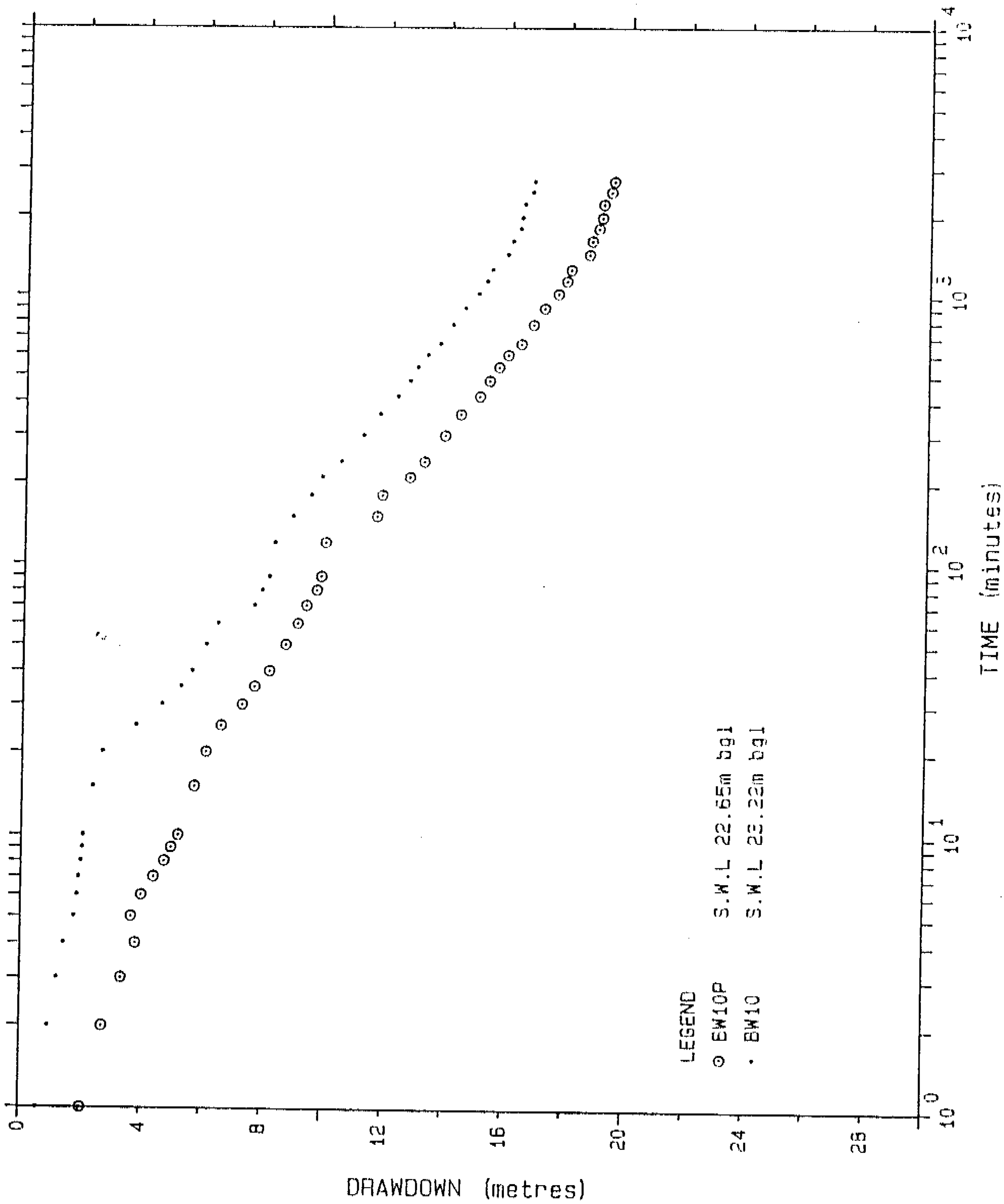
Client : BILLITON AUST PTY LTD	BORE BW6P CONSTANT RATE PUMPING TEST
Project : MT TODD WATER SUPPLY	DRAWDOWN IN PUMPED AND OBSERVATION BORES
Date : DECEMBER 1988	Pumping Rate of BW6P 350 m^3/day
Dwg. No.: 128.1/88/2-9	Commenced test on 31/10/88

Figure 10



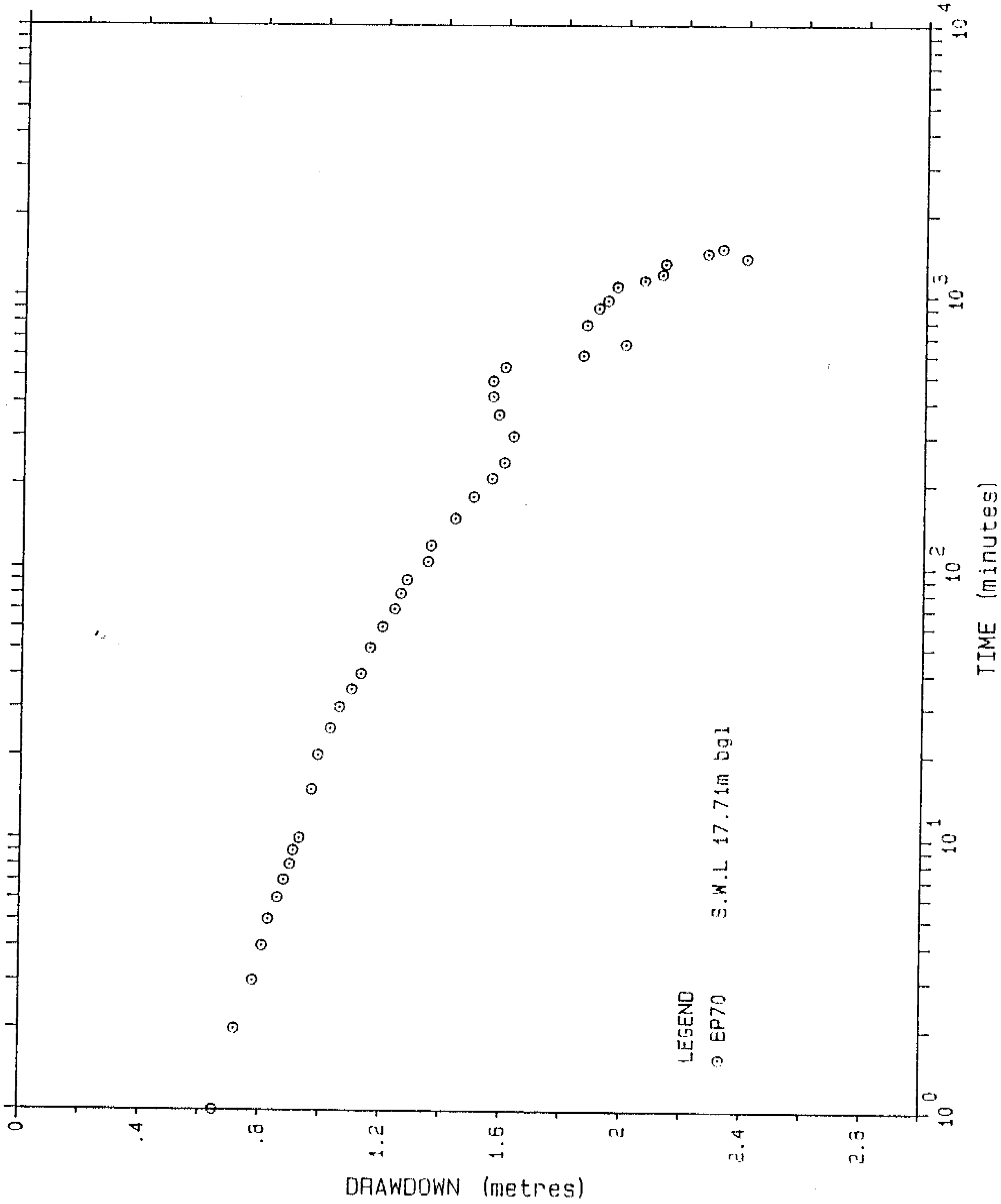
Client : BILLITON AUST PTY LTD	BORE BW8P CONSTANT RATE PUMPING TEST
Project : MT TODD WATER SUPPLY	DRAWDOWN IN PUMPED AND OBSERVATION BORES
Date : DECEMBER 1988	Pumping Rate of BW8P 400 m^3/day
Dwg. No.: 128.1/88/2-10	Commenced test on 14/11/88

Figure 11



Client : BILLITON AUST PTY LTD	BORE BW10P CONSTANT RATE PUMPING TEST
Project : MT TODD WATER SUPPLY	DRAWDOWN IN PUMPED AND OBSERVATION BORES
Date : DECEMBER 1988	Pumping Rate of BW10P 250 m^3/day
Dwg. No.: 128.1/88/2-11	Commenced test on 7/11/88

Figure 12



Client : BILLITON AUST PTY LTD

Project : MT TODD WATER SUPPLY

Date : DECEMBER 1988

Dwg. No.: 128.1/88/2-12

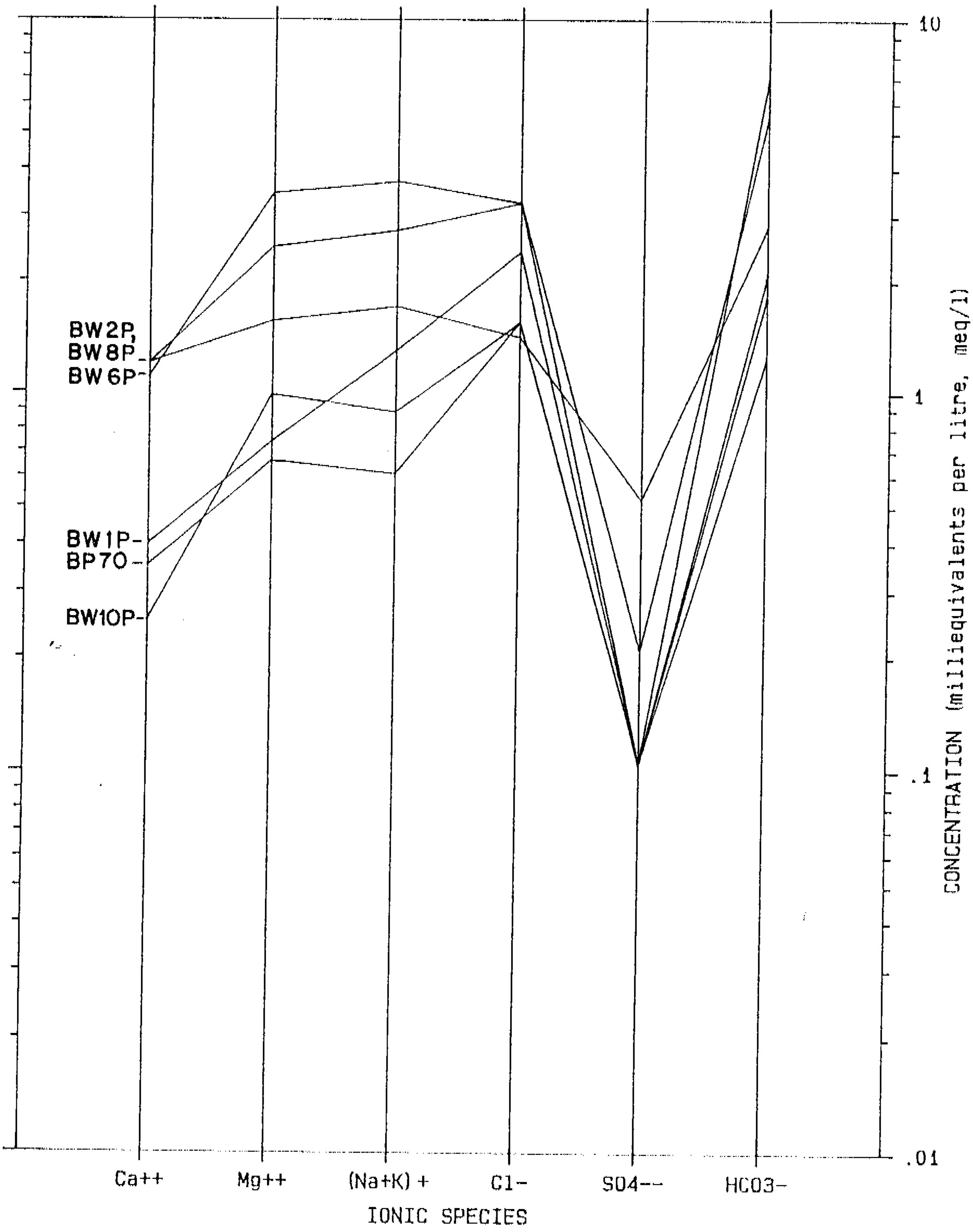
BORE BP70 CONSTANT RATE PUMPING TEST

DRAWDOWN IN PUMPED BORE

Pumping Rate of BP70 150 m^3/day

Commenced test on 13/11/88

FIGURE 13



Client : Billiton Australia Project : Batman Prospect Date : December 1988 Dwg.No. 128.1/88/2-13	SCHOELLER DIAGRAM OF WATER CHEMISTRY

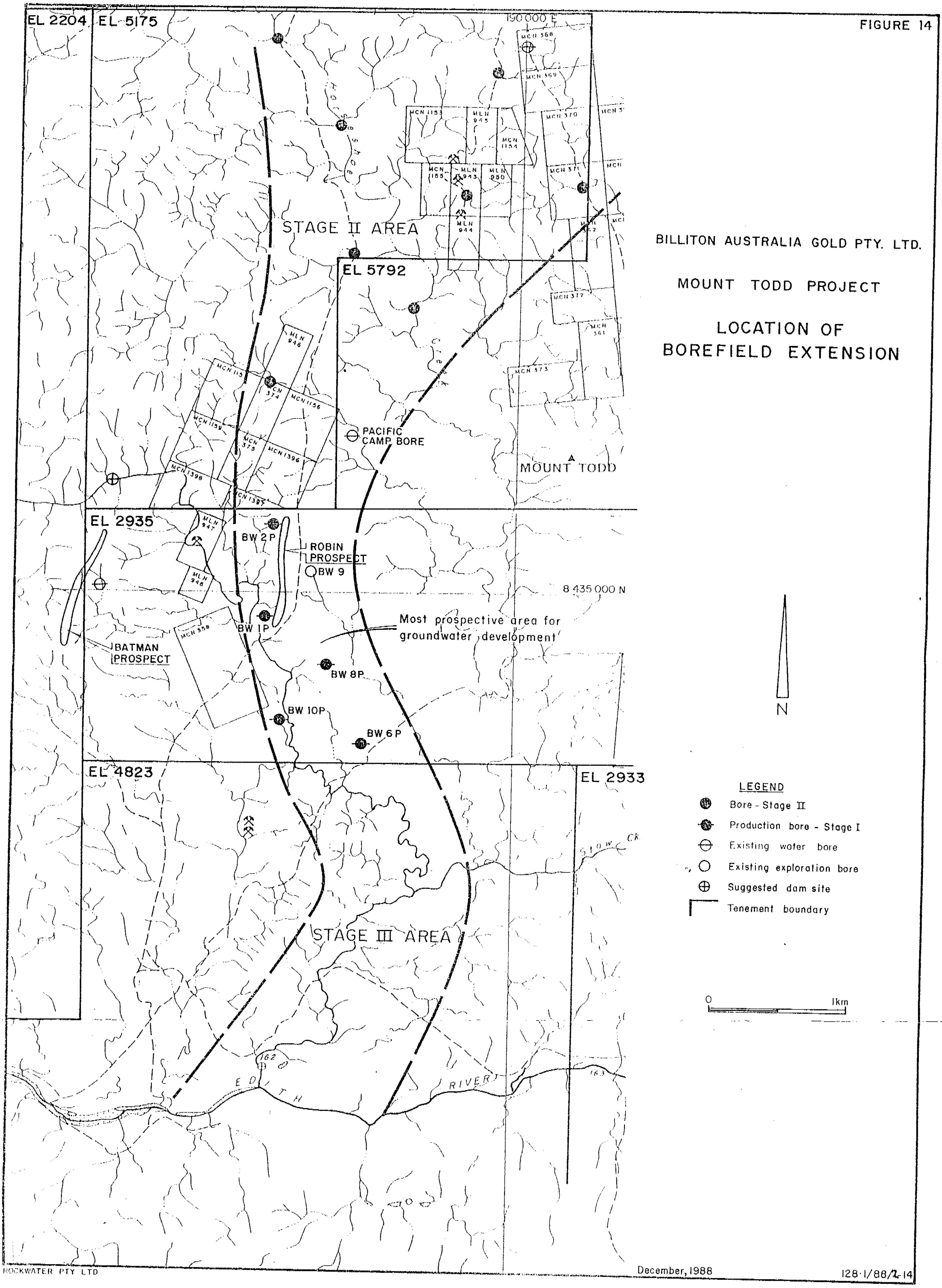


FIGURE 14
BILLITON AUSTRALIA GOLD PTY. LTD.
MOUNT TODD PROJECT
LOCATION OF
BOREFIELD EXTENSION
EL 2204
EL 5175
EL 5792
EL 2935
EL 4823
EL 2933
190 000 E
8 435 000 N
STAGE II AREA
STAGE III AREA
PACIFIC CAMP BORE
MOUNT TODD
ROBIN PROSPECT
BATMAN PROSPECT
BW 2P
BW 9
BW 1P
BW 8P
BW 10P
BW 6P
Most prospective area for groundwater development
EDITH RIVER
N
LEGEND
Bore - Stage II
Production bore - Stage I
Existing water bore
Existing exploration bore
Suggested dam site
Tenement boundary
0
1km
ROCKWATER PTY LTD
December, 1988
128·1/88/2-14

BILLITON AUSTRALIA PTY LTD

GROUNDWATER DRILLING AND TESTING
BATMAN WATER SUPPLY
MT TODD, NORTHERN TERRITORY

VOLUME II - FIELD DATA
NOVEMBER, 1988

128.1/88/2

TABLE OF CONTENTS

SECTION 1

STEP RATE PUMPING TEST

DATA AND ANALYSES

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
94 ROKEBY ROAD, SUBIACO, WESTERN AUSTRALIA 6008. TELEPHONE (09) 382 4922

PROJECT: Billiton Australia Pty Ltd. Batman Water Supply

DATE: ______ SHEET ______ OF ______

Bore Efficiency Results. (Sheahan Analysis).

	Pumping Rate (m^3/d)	Efficiency (%)
BW1P	250	82
BW2P	300	cannot analyse with this method.
BW6P	300	68
BW8P	250	96
BW10P	250	50

Rockwater PROPRIETARY LIMITED — **GROUNDWATER CONSULTANTS**

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

MONITOR BORE BW 1, BW 2 AND BW 9.

ORIFICE PLATES - 1", 1½", 2" AND 2½".

STEP-RATE PUMPING TEST

BORE NO: BW 1P CLIENT: BILLITON

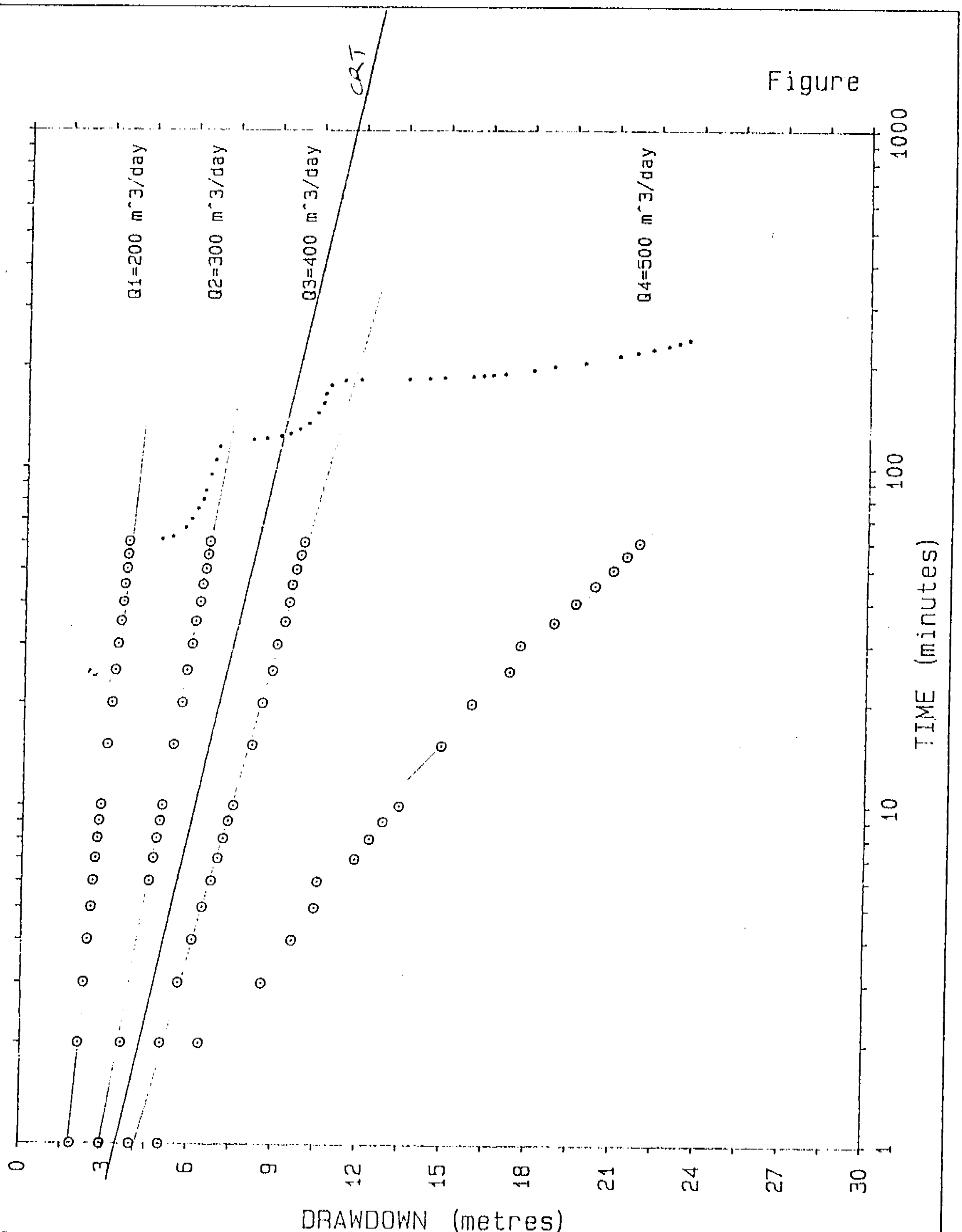
DATE: 27/10/88 PUMP INLET SETTING: 55 m

S.W.L. (m below dip tube): 18.60 AVAILABLE DRAWDOWN: 36.4 m

DIP TUBE HEIGHT ABOVE COLLAR: 0.22 WEIR PIPE DIAMETER: 3"

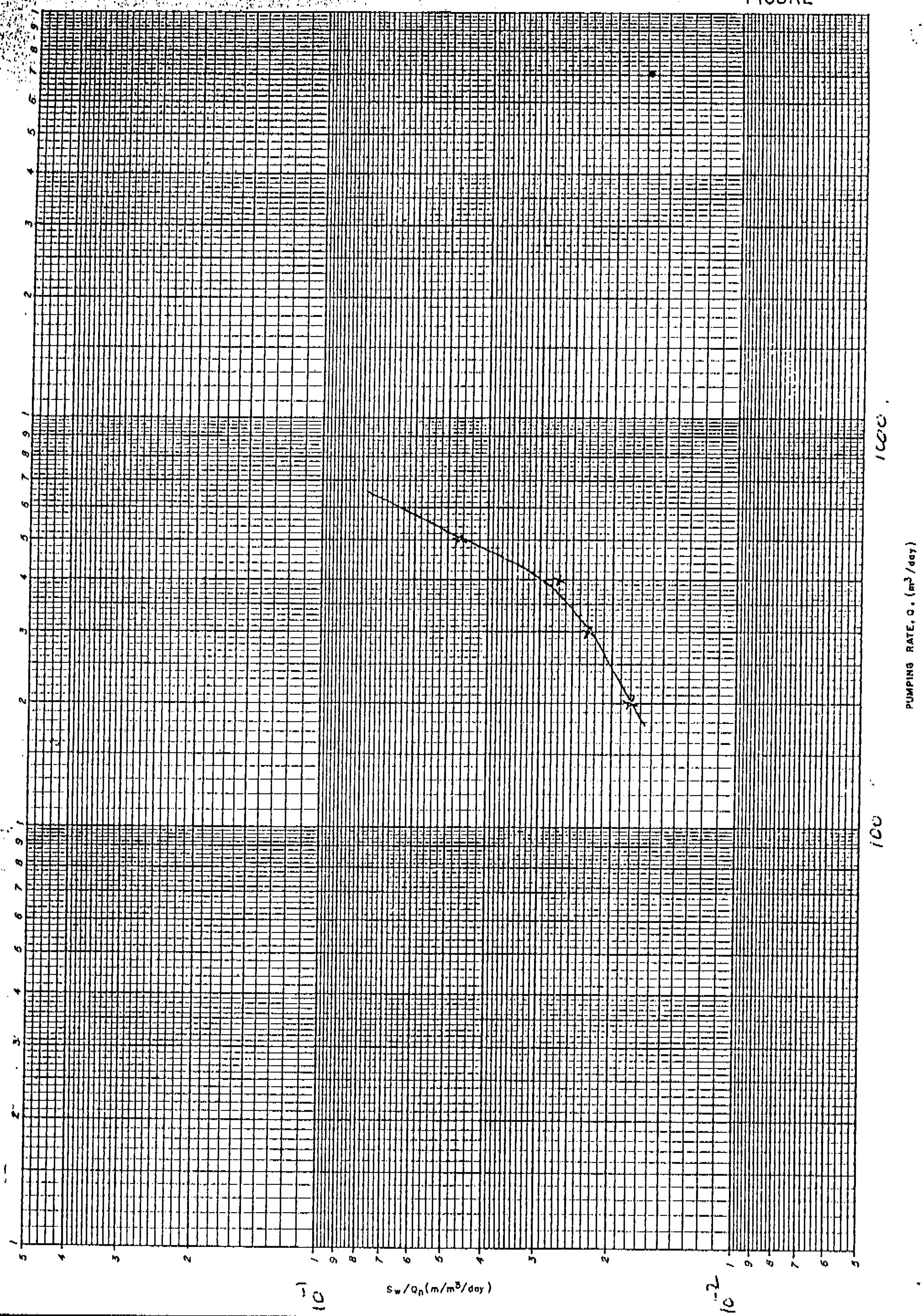
ORIFICE DIAMETER	2 in.		2 in.		2 in.		2½ in.		in.	
MANOMETER HEIGHT	5.2 in.		11.6 in.		20.6 in.		7.8 in.		in. mm	
BORE DISCHARGE	200 m³/d (2.3 L/sec)		300 m³/d (3.5 L/sec)		400 m³/d (4.6 L/sec)		500 m³/d (5.8 L/sec)		m³/d	
TIME (minutes)	WL (m)	DD	WL (m)	DD	WL (m)	DD	WL (m)	DD	WL (m)	DD
1	20.44	1.84	23.44	4.84	26.66	8.06	29.91	11.31		
2	20.68	2.08	23.83	5.23	27.13	8.53	30.48	11.88		
3	20.85	2.25	[illegible]	–	27.41	8.81	32.20	13.60		
4	20.97	2.37	–	–	27.65	9.05	32.92	14.32		
5	21.08	2.48	–	–	27.81	9.21	33.44	14.84		
6	21.14	2.54	24.28	5.68	27.97	9.37	33.32	14.72		
7	21.21	2.61	24.35	5.75	28.07	9.47	34.46	15.86		
8	21.27	2.67	24.40	5.80	28.14	9.54	34.82	16.22		
9	21.32	2.72	24.46	5.86	28.21	9.61	35.16	16.56		
10	21.38	2.78	24.50	5.90	28.31	9.71	35.61	17.01		
15	21.58	2.98	24.72	6.12	28.64	10.04	36.62	18.02		
20	21.73	3.13	24.90	6.30	28.77	10.17	37.36	18.76		
25	21.84	3.24	24.99	6.39	28.96	10.36	38.28	19.68		
30	21.93	3.33	25.11	6.51	28.99	10.39	39.00	20.40		
35	22.03	3.43	25.17	6.57	29.16	10.56	39.67	21.07		
40	22.10	3.50	25.29	6.69	29.22	10.62	40.30	21.70		
45	22.14	3.54	25.33	6.73	29.24	10.64	40.86	22.26		
50	22.21	3.61	25.41	6.81	29.32	10.72	41.40	22.80		
55	22.25	3.65	25.46	6.86	29.42	10.82	41.78	23.18		
60	22.28	3.68	25.49	6.89	29.48	10.88	42.14	23.54		

132
294
523



Client : BILLITON AUSTRALIA PTY LTD	Bore BW1P Step Rate Pump Test
Project : BATMAN WATER SUPPLY	Static Water Level 18.33m bgl
Date : DECEMBER 1988	Commenced 27/10/88
Dwg. No.: 128.1/88/2-	

FIGURE



Client : Billiton Aust P/L

Project : Batman Water supply

Date : Dec 1988 Dwg. No.

Bore BW1P

Sheahan Analysis

$P = 4.0$

$s_w/Q_x = 0.033$

$Q_x = 415$

ROCKWATER PTY LTD

ROCKWATER
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

STEP-RATE PUMPING TEST

BORE NO: BW2P .... CLIENT: BILLITON AUST LTD ....

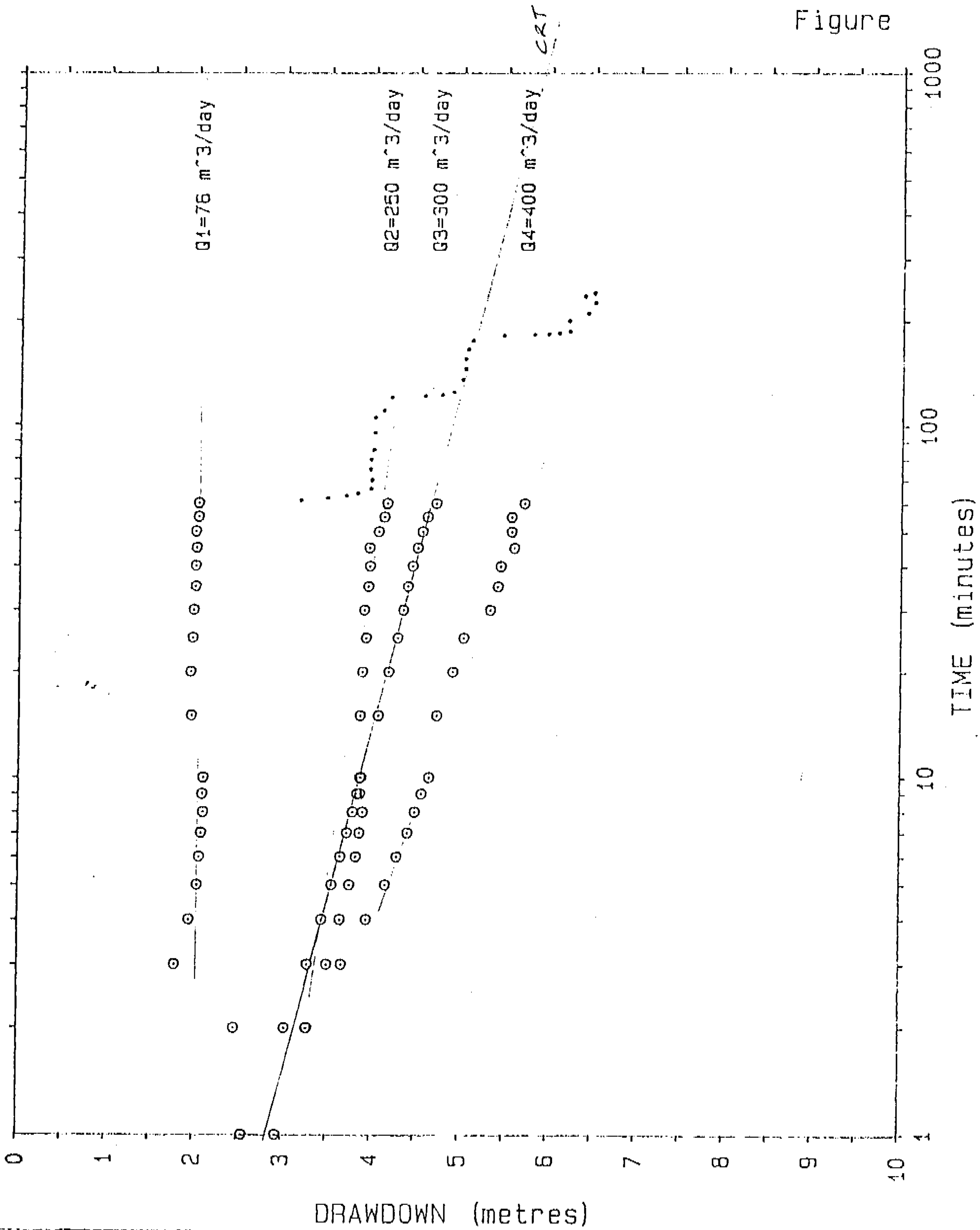
DATE: 9/11/88 .... PUMP INLET SETTING: 56m ....

S.W.L. (m below dip tube): 26.27 .... AVAILABLE DRAWDOWN: ....

DIP TUBE HEIGHT ABOVE COLLAR: .... WEIR PIPE DIAMETER: 3" ....

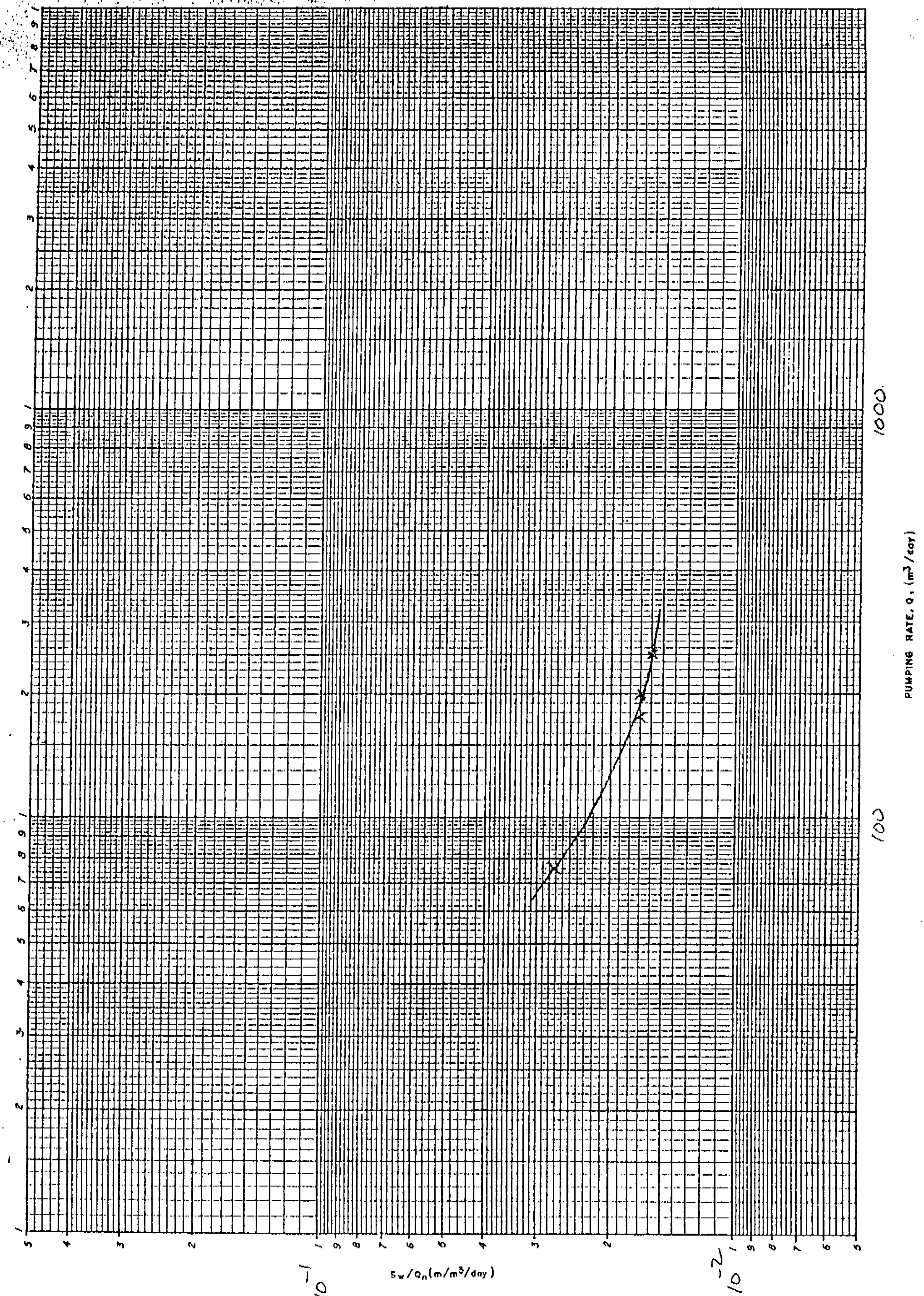
ORIFICE DIAMETER	1½ in.		2 in.		2 in.		2 in.		in.	
MANOMETER HEIGHT	19.6 cm in. ~~mm~~		8.1 in. ~~mm~~		11.6 in. ~~mm~~		20.6 in. ~~mm~~		in. mm	
BORE DISCHARGE	76 ~~200~~ m³/d		250 m³/d		300 m³/d		400 m³/d		m³/d	
TIME (minutes)	m WL	DD	m WL	DD	m WL	DD	m WL	DD	m WL	DD
1	–	–	29.45	3.18	30.85	4.58	31.73	5.46		
2	28.73	2.46	29.75	3.48	31.04	4.77	32.08	5.81		
3	28.06	1.79	29.96	3.69	31.13	4.86	32.24	5.97		
4	28.22	1.95	30.09	3.82	31.17	4.90	32.36	6.09		
5	28.31	2.04	30.18	3.91	31.19	4.92	32.45	6.18		
6	28.33	2.06	30.24	3.97	31.21	4.94	32.48	6.21		
7	28.35	2.08	30.27	4.00	31.22	4.95	32.52	6.25		
8	28.37	2.10	30.30	4.03	31.23	4.96	32.53	6.26		
9	28.36	2.09	30.26	3.99	31.23	4.96	32.54	6.27		
10	28.37	2.10	30.25	3.98	31.24	4.97	32.57	6.30		
15	28.23	1.96	30.23	3.96	31.27	5.00	32.44	6.17		
20	28.22	1.95	30.24	3.97	31.28	5.01	32.47	6.20		
25	28.24	1.97	30.27	4.00	31.30	5.03	32.48	6.21		
30	28.25	1.98	30.24	3.97	31.30	5.03	32.69	6.42		
35	28.27	2.00	30.28	4.01	31.30	5.03	32.70	6.43		
40	28.27	2.00	30.20	4.02	31.31	5.04	32.67	6.40		
45	28.28	2.01	30.28	4.01	31.33	5.06	32.77	6.50		
50	28.27	2.00	30.38	4.11	31.35	5.08	32.69	6.42		
55	28.30	2.03	30.44	4.17	31.38	5.11	32.65	6.38		
60	28.30	2.03	30.47	4.20	31.45	5.18	32.76	6.49		

Figure



Client : BILLITON AUSTRALIA PTY LTD	Bore BW2P Step Rate Pump Test
Project : BATMAN WATER SUPPLY	Static Water Level 25.79m bgl
Date : DECEMBER 1988	Commenced 9/11/88
Dwg. No.: 128.8/88/2-	

FIGURE



Client : Billiton Aust P/L

Project: Batman Water Supply

Date : Dec 1988 Dwg. No.

Bore BW2P

Sheahan Analysis

Cannot Analyse!

ROCKWATER PTY LTD

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

STEP-RATE PUMPING TEST

BORE NO: BW6P CLIENT: BILLITON

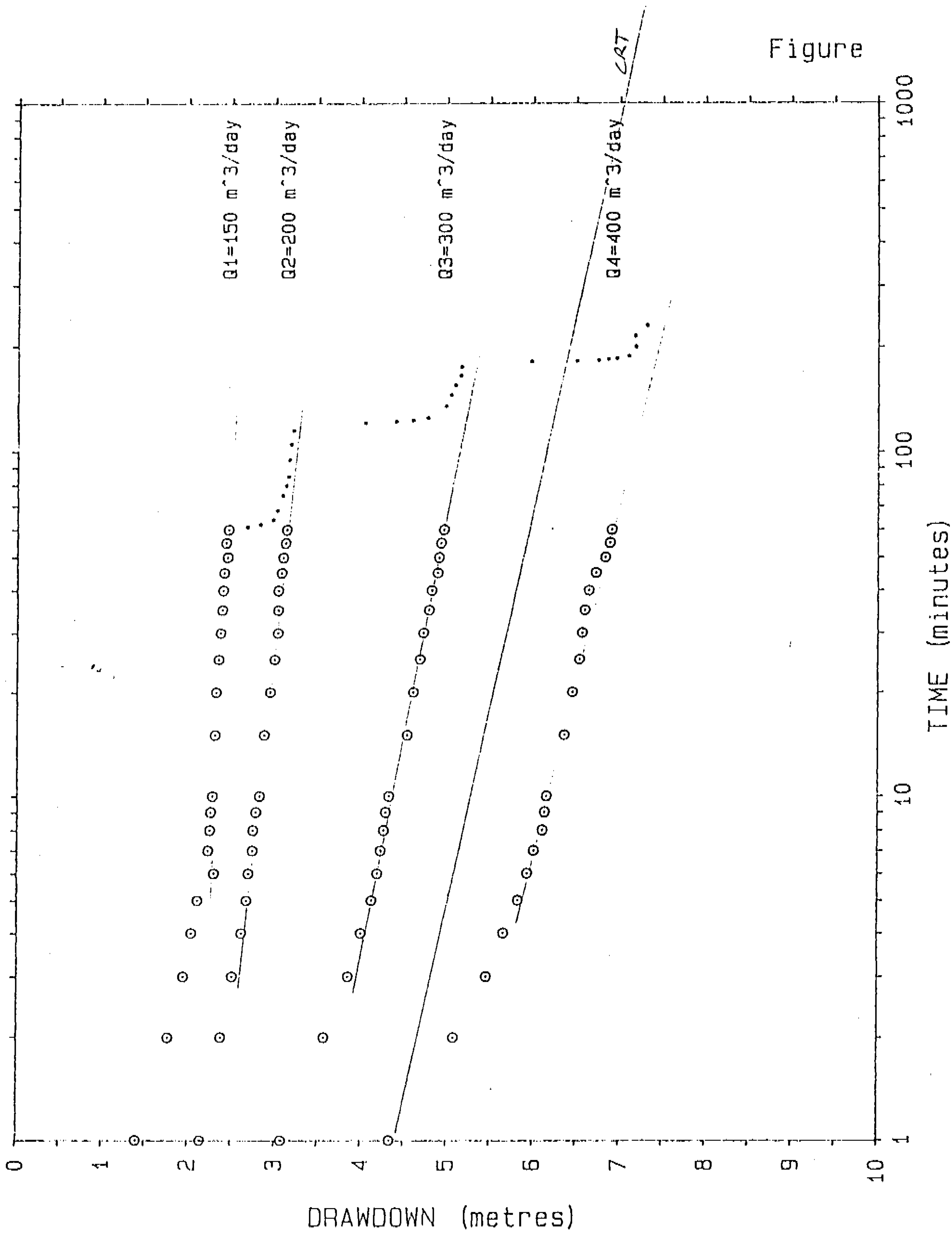
DATE: 30/10/88 PUMP INLET SETTING: 59m

S.W.L. (m below dip tube): 19.05 AVAILABLE DRAWDOWN:

DIP TUBE HEIGHT ABOVE ~~COLLAR~~ GROUND: 0.48 WEIR PIPE DIAMETER: 3"

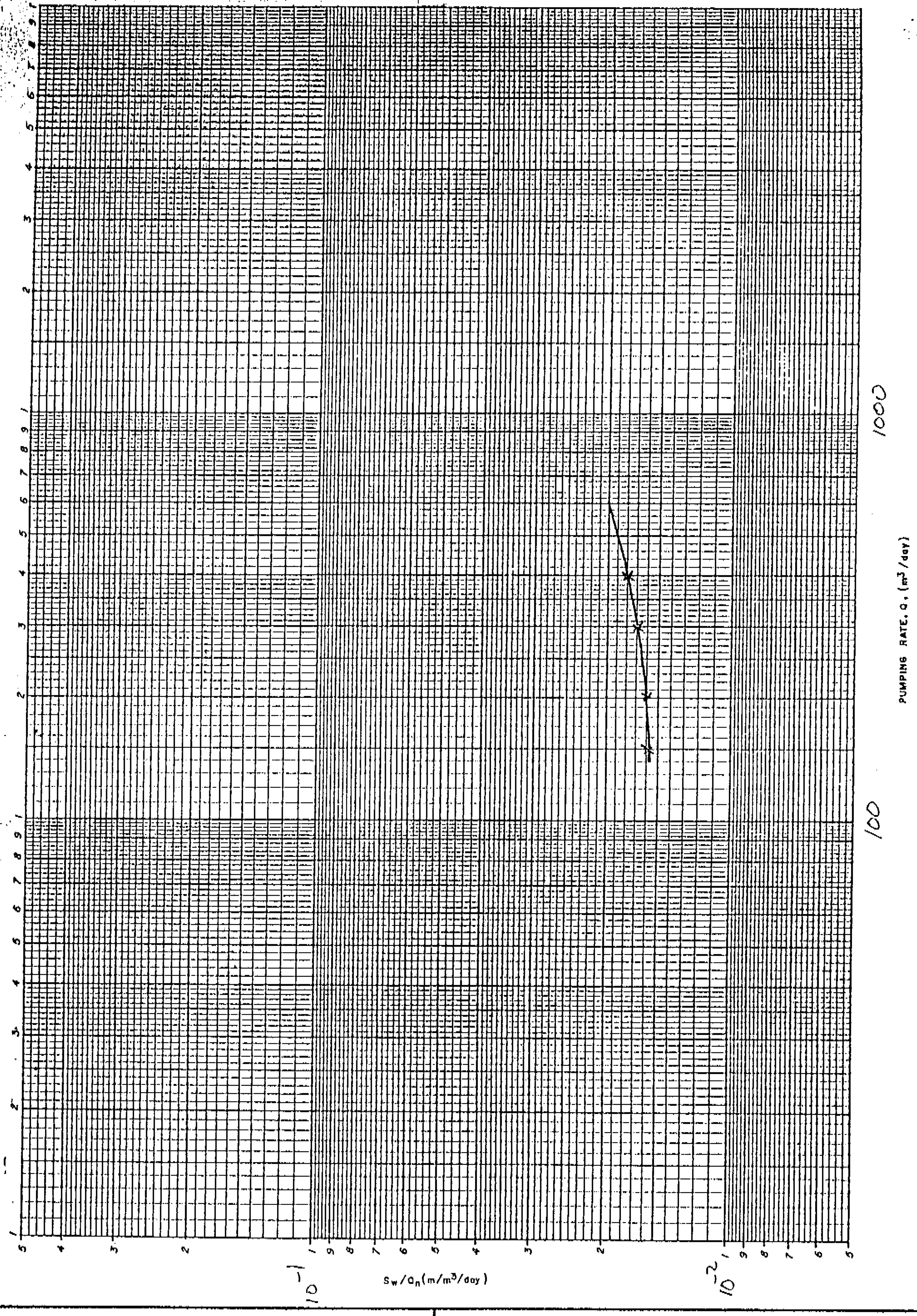
ORIFICE DIAMETER	1½ in.		2 in.		2 in.		2 in.		in.	
MANOMETER HEIGHT	11 in.		5.2 ~~in.~~		11.6 ~~in.~~		20.6 in.		in. mm	
BORE DISCHARGE	150 m³/d		200 m³/d		300 m³/d		400 m³/d		m³/d	
TIME (minutes)	m WL	DD	m WL	DD	m WL	DD	m WL	DD	m WL	DD
1	20.45	1.40	21.72	2.67	23.09	4.04	25.02	5.97		
2	20.82	1.77	21.87	2.82	23.45	4.40	25.55	6.50		
3	21.00	1.95	21.95	2.90	23.65	4.60	25.80	6.75		
4	21.09	2.04	22.02	2.97	23.74	4.69	25.91	6.86		
5	21.16	2.11	22.05	3.00	23.82	4.77	26.01	6.96		
6	21.25	2.20	22.05	3.00	23.85	4.80	26.06	7.01		
7	21.28	2.23	22.08	3.03	23.86	4.81	26.09	7.04		
8	21.30	2.25	22.07	3.02	23.87	4.82	26.15	7.10		
9	21.31	2.26	22.09	3.04	23.87	4.82	26.14	7.09		
10	21.33	2.28	22.12	3.07	23.89	4.84	26.13	7.08		
15	21.36	2.31	22.13	3.08	24.03	4.98	26.22	7.17		
20	21.37	2.32	22.17	3.12	24.05	5.00	26.23	7.18		
25	21.40	2.35	22.20	3.15	24.09	5.04	26.25	7.20		
30	21.42	2.37	22.22	3.17	24.10	5.05	26.23	7.18		
35	21.44	2.39	22.21	3.16	24.14	5.09	26.22	7.17		
40	21.45	2.40	22.20	3.15	24.15	5.10	26.23	7.18		
45	21.46	2.41	22.23	3.18	24.20	5.15	26.28	7.23		
50	21.50	2.45	22.24	3.19	24.20	5.15	26.36	7.31		
55	21.48	2.43	22.26	3.21	24.21	5.16	26.39	7.34		
60	21.51	2.46	22.27	3.22	24.23	5.18	26.39	7.34		

Figure



Client : BILLITON AUSTRALIA PTY LTD	Bore BW6P Step Rate Pump Test
Project : BATMAN WATER SUPPLY	Static Water Level 18.57m bgl
Date : DECEMBER 1988	Commenced 30/10/88
Dwg. No.: 128.1/88/2-	

FIGURE



Client : Billiton Aust P/L

Project : Batman Water Supply

Date : Dec 1988 Dwg. No.

Bore BW6P

Sheahan Analysis

$P = 2.1$

$S_w/Q_x = 0.029$

$Q_x = 600$

ROCKWATER PTY LTD

Rockwater
PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

STEP-RATE PUMPING TEST

BORE NO: BW8P .... CLIENT: Billiton Blast

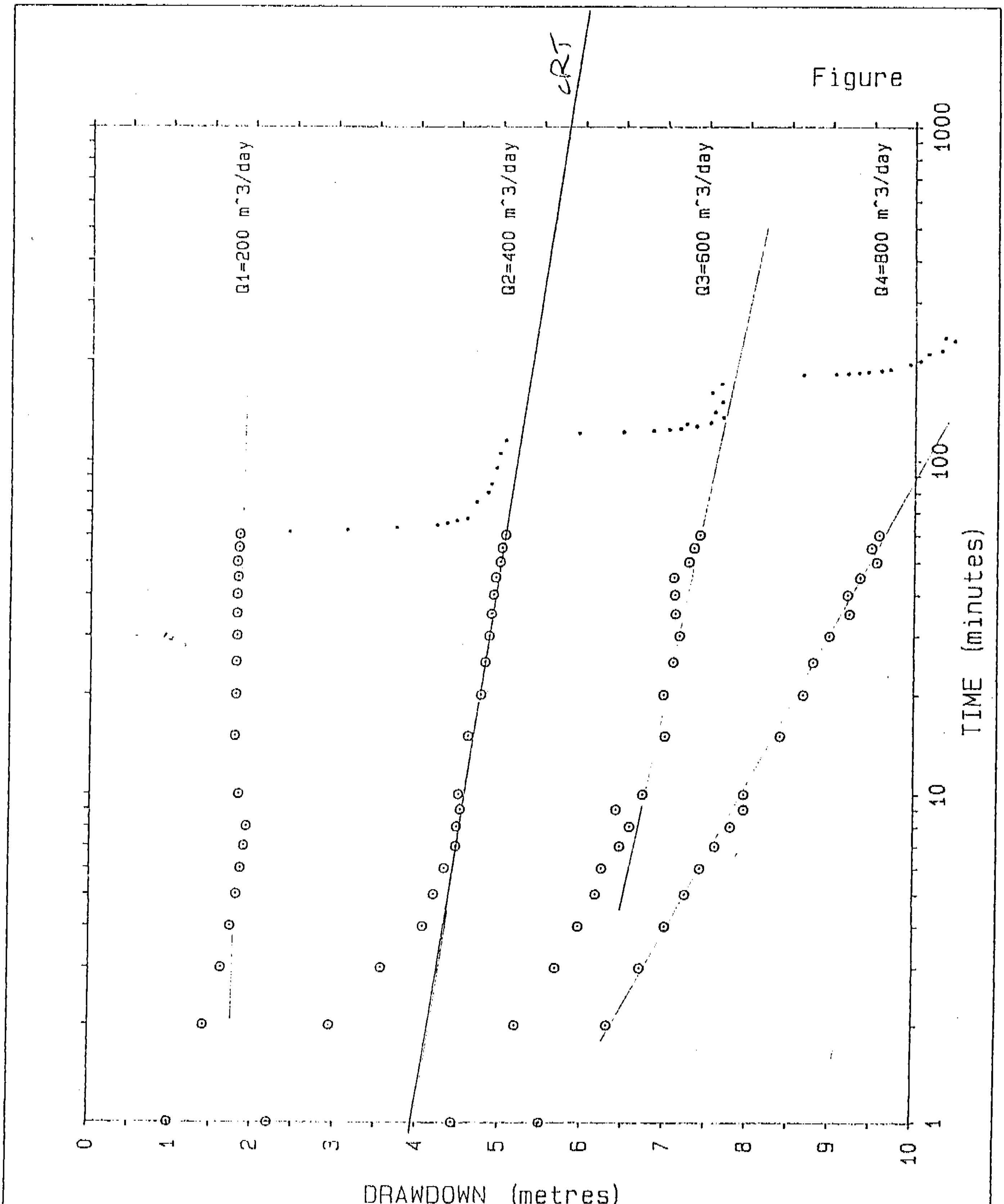
DATE: 12/11/98 .... PUMP INLET SETTING: 56m

S.W.L. (m below dip tube): 24.29m .... AVAILABLE DRAWDOWN: ....

DIP TUBE HEIGHT ABOVE COLLAR: .... WEIR PIPE DIAMETER: 3"

SWL = 24.27

ORIFICE DIAMETER	2 in.		$2\frac{1}{2}$ in.		$2\frac{1}{2}$ in.		$2\frac{1}{2}$ in.		in.	
MANOMETER HEIGHT	5.2 in. mm		5 in. mm		15.2 in. mm		20 in. mm		in. mm	
BORE DISCHARGE	200 m³/d		400 m³/d		600 m³/d		800 m³/d		m³/d	
TIME (minutes)	m WL	DD	m WL	DD	m WL	DD	m WL	DD	m WL	DD
1	25.28	0.97	26.69	2.42	30.20	5.93	32.91	8.64		
2	25.70	1.41	27.39	3.12	30.74	6.47	33.30	9.03		
3	25.91	1.62	27.99	3.72	31.10	6.83	33.45	9.18		
4	26.[illegible]	1.73	28.48	4.21	31.29	7.02	33.58	9.31		
5	26.09	1.80	28.60	4.33	31.43	7.16	33.69	9.42		
6	26.14	1.85	28.72	4.45	31.55	7.28	33.76	9.49		
7	26.18	1.89	28.85	4.58	31.62	7.35	33.85	9.58		
8	26.21	1.92	28.85	4.58	31.70	7.43	33.96	9.69		
9	—	—	28.89	4.62	31.25	7.23	34.05	9.78		
10	26.11	1.82	28.86	4.59	31.79	7.52	33.99	9.72		
15	26.06	1.77	28.96	4.69	31.94	7.67	34.20	9.93		
20	26.07	1.78	29.10	4.83	31.84	7.57	34.32	10.05		
25	26.07	1.78	29.14	4.87	31.90	7.63	34.32	10.05		
30	26.08	1.79	29.18	4.91	31.93	7.66	34.42	10.15		
35	26.08	1.79	29.20	4.93	31.84	7.57	34.58	10.31		
40	26.08	1.79	29.22	4.95	31.80	7.53	34.49	10.22		
45	26.09	1.80	29.24	4.97	31.76	7.49	34.58	10.31		
50	26.08	1.79	29.29	5.02	31.92	7.65	34.73	10.46		
55	26.10	1.81	29.31	5.04	31.96	7.69	34.62	10.35		
60	26.11	1.82	29.35	5.08	32.01	7.74	34.67	10.40		



Client : BILLITON AUSTRALIA PTY LTD	Bore BW8P Step Rate Pump Test
Project : BATMAN WATER SUPPLY	Static Water Level 23.49m bgl
Date : DECEMBER 1988	Commenced 12/11/88
Dwg. No.: 128.1/88/2-	

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

STEP-RATE PUMPING TEST

BORE NO: BW10P .... CLIENT: BILLITON AUST LTD

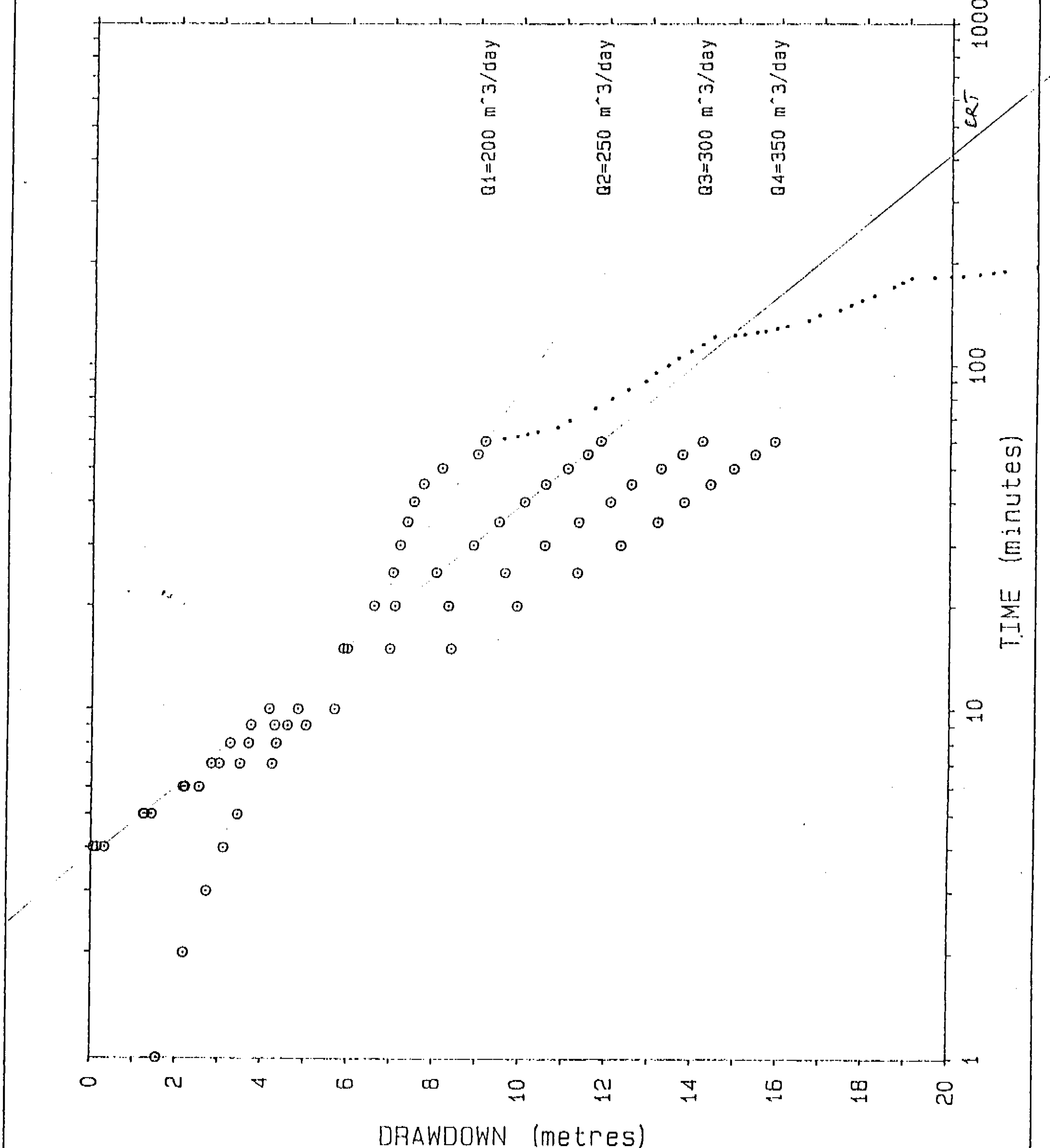
DATE: 2/11/88 .... PUMP INLET SETTING: 56m

S.W.L. (m below dip tube): 20.50 .... AVAILABLE DRAWDOWN: 35.5m

DIP TUBE HEIGHT ABOVE COLLAR: .... WEIR PIPE DIAMETER: 3"

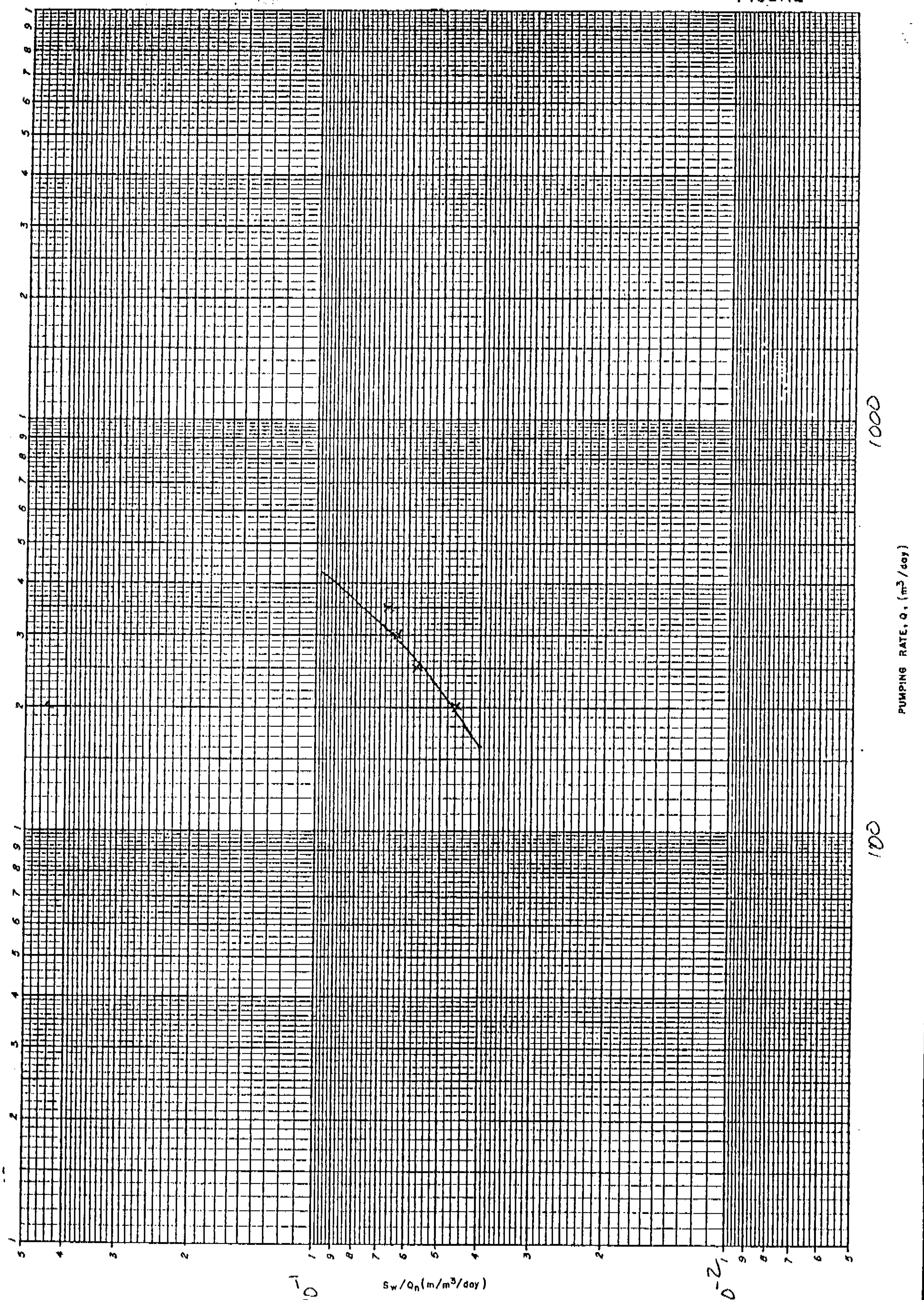
ORIFICE DIAMETER	2 in.		2 in.		2 in.		2½ in.		in.	
MANOMETER HEIGHT	5.2 in. ~~mm~~		8.1 ~~[illegible]~~ in. ~~mm~~		11.6 ~~[illegible]~~ in. ~~mm~~		[illegible] ~~7.8~~ in. ~~mm~~		in. mm	
BORE DISCHARGE	200 m³/d		250 ~~300~~ m³/d		300 ~~400~~ m³/d		350 ~~500~~ m³/d		m³/d	
TIME (minutes)	WL (m)	DD (m)	WL (m)	DD (m)	WL (m)	DD (m)	WL (m)	DD (m)	WL (m)	DD (m)
1	22.06	1.56	30.07	9.57	34.96	14.46	40.10	19.60		
2	22.68	2.18	30.38	9.88	35.44	14.94	40.49	19.99		
3	23.21	2.71	30.61	10.11	35.67	15.17	40.76	20.26		
4	23.60	3.10	30.85	10.35	35.83	15.33	40.95	20.45		
5	23.92	3.42	31.04	10.54	35.96	15.46	41.15	20.65		
6	~~24.72~~	–	31.19	10.82	36.16	15.66	41.31	20.81		
7	24.82	4.22	31.32	10.98	36.30	15.80	41.46	20.96		
8	24.82	4.32	31.48	10.98	36.43	15.93	41.62	21.12		
9	25.07	4.57	31.59	11.09	36.55	16.05	41.72	21.22		
10	25.32	4.82	31.69	11.19	36.66	16.16	41.85	21.35		
15	26.46	5.96	32.19	11.69	37.17	16.67	42.54	22.04		
20	27.08	6.58	32.58	12.08	37.42	16.92	42.67	22.17		
25	27.52	7.02	32.96	12.46	37.90	17.40	43.01	22.51		
30	27.68	7.18	33.37	12.87	38.16	17.66	43.18	22.68		
35	27.85	7.35	33.60	13.10	38.41	17.91	43.34	22.84		
40	28.00	7.50	33.89	13.39	38.70	18.20	43.36	22.86		
45	28.22	7.72	34.12	13.62	38.80	18.30	43.47	22.97		
50	28.65	8.15	34.42	13.92	39.15	18.65	43.57	23.07		
55	29.48	8.98	34.70	14.20	39.35	18.85	43.67	23.17		
60	29.65	9.15	34.84	14.34	39.56	19.06	43.78	23.28		

Figure



Client : BILLITON AUSTRALIA PTY LTD Project : BATMAN WATER SUPPLY Date : DECEMBER 1988 Dwg. No.: 128.1/88/2-	Bore BW10P Step Rate Pump Test Static Water Level 20.12m bgl Commenced 2/11/88

FIGURE



Client : Billiton Aust P/L

Project: Batman Water Supply

Date : Dec 1988 Dwg. No.

Bore BW10P

Sheahan Analysis

$P = 2.8$

$S_w/Q_x = 0.056$

$Q_x = 255$

SECTION 2

CONSTANT RATE PUMPING TEST DATA

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

BW1Pp88

48 Hour Constant Rate Pumping Test

BORE NO: BW 1P CLIENT: BILLITON

DATE: 28-10-88 PUMP INLET SETTING: 55m

S.W.L. (m below dip tube): 18.80 AVAILABLE DRAWDOWN: 36.20 m

DIP TUBE HEIGHT ABOVE COLLAR: 0.02m WEIR PIPE DIAMETER: 3"

COLLAR HEIGHT ABOVE ~~GROUND~~ GROUND: 0.27m PUMPING RATE: 350 m³/d

ORIFICE PLATE DIAMETER: 2" MANOMETER TUBE HEIGHT: 40 cm

START TIME: 06^{30}

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
28/10/88		1	21.90	3.10		10^{30}	240	29.88	11.08
		2	–	–		11^{30}	300	30.17	11.37
		3	–	–		12^{30}	360	30.40	11.60
		4	–	–		1^{30}	420	30.66	11.86
		5	–	–		2^{30}	480	30.85	12.05
		6	24.95	6.15		3^{30}	540	31.06	12.26
		7	25.20	6.40		4^{30}	600	31.28	12.48
		8	25.34	6.54		5^{30}	660	31.53	12.73
		9	25.50	6.70		6^{30}	720	31.75	12.95
	06^{40}	10	25.60	6.80		7^{30}	780	31.93	13.13
	06^{45}	15	26.10	7.30		8^{30}	840	31.92	13.12
	06^{50}	20	26.58	7.78		9^{30}	900	32.38	13.58
	06^{55}	25	26.88	8.08		10^{30}	960	32.52	13.72
	07^{00}	30	27.06	8.26		11^{30}	1020	32.82	14.02
	07^{05}	35	27.21	8.41	29/10/88	12^{30}	1080	33.01	14.21
	07^{10}	40	27.37	8.57		1^{30}	1140	33.01	14.21
	07^{20}	50	27.62	8.82		2^{30}	1200	33.34	14.54
	07^{30}	60	27.85	9.05		3^{30}	1260	33.45	14.65
	07^{40}	70	28.05	9.25		4^{30}	1320	33.55	14.75
	07^{50}	80	28.22	9.42		5^{30}	1380	33.64	14.84
	08^{00}	90	28.40	9.60		6^{30}	1440	33.81	15.01
	08^{30}	120	28.80	10.00		7^{30}	1500	33.90	15.10
	09^{00}	150	29.03	10.23		8^{30}	1560	34.02	15.22
	09^{30}	180	29.22	10.42		9^{30}	1620	34.06	15.26
	10^{00}	210	29.42	10.62		10^{30}	1680	34.14	15.34

PTO...

48 Hour Constant Rate Pumping Test (cont.)

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Recovery Date	Recovery Elapsed Time Hours	Recovery Elapsed Time Minutes	Recovery Water Level m	Recovery Drawdown m
	11^{30}	1740	34.23	15.43			1	30.21	11.41
	12^{30}	1800	34.26	15.46			2	28.55	9.75
	1^{30}	1860	34.39	15.59			3	27.36	8.56
	2^{30}	1920	34.47	15.67			4	26.60	8.00
	3^{30}	1980	34.60	15.80			5	26.54	7.74
	4^{30}	2040	34.80	16.00			6	26.41	7.61
	5^{30}	2100	34.88	16.08			7	26.29	7.49
	6^{30}	2160	34.97	16.17			8	26.17	7.37
	7^{30}	2220	35.00	16.20			9	26.07	7.27
	8^{30}	2280	34.87	16.07			10	26.03	7.23
	9^{30}	2340	34.98	16.18			15	25.70	6.90
	10^{30}	2400	35.48	16.68			20	25.47	6.67
	11^{30}	2460	35.42	16.62			25	25.32	6.52
30/10/88	12^{30}	2520	35.50	16.70			30	25.22	6.42
	1^{30}	2580	35.60	16.80			35	~~24~~ –	–
	2^{30}	2640	35.70	16.90			40	24.95	6.15
	3^{30}	2700	35.68	16.88			50	24.75	5.95
	4^{30}	2760	35.79	16.99			60	24.66	5.86
	5^{30}	2820	35.88	17.08			70		
	6^{30}	2880	35.95	17.15			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater
PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

DISTANCE TO PRODUCTION BORE - BC01

48 Hour Constant Rate Pumping Test

BORE NO: BW.1 CLIENT: BILLITON

DATE: 28-10-88 PUMP INLET SETTING: -

S.W.L. (m below ~~dip tube~~ COLLAR): 18.89 AVAILABLE DRAWDOWN: -

DIP TUBE HEIGHT ABOVE COLLAR: - WEIR PIPE DIAMETER: -

COLLAR HEIGHT ABOVE ~~GROUND~~ CEMENT BLOCK: 0.23 PUMPING RATE: -

ORIFICE PLATE DIAMETER: - MANOMETER TUBE HEIGHT: -

START TIME: 6^{30}

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
28/10/88		1				10^{30}	240	24.78	5.89
		2		WITH PROBE PROBLEMS		11^{30}	300	24.95	6.06
		3				12^{30}	360	25.11	6.22
		4				1^{30}	420	25.23	6.34
		5				2^{30}	480	25.37	6.48
		6				3^{30}	540	25.47	6.58
		7				4^{30}	600	25.55	6.66
		8				5^{30}	660	25.62	6.73
		9				6^{30}	720	25.71	6.82
	6^{40}	10	21.01	2.12		7^{30}	780	25.81	6.92
	6^{45}	15	21.52	2.63		8^{30}	840	25.89	7.00
	6^{50}	20	21.86	2.97		9^{30}	900	25.93	7.04
	6^{55}	25	22.17	3.28		10^{30}	960	26.50	7.66
	7^{00}	30	22.34	3.45		11^{30}	1020	26.12	7.23
	7^{05}	35	22.52	3.63	29/10/88	12^{30}	1080	26.13	7.24
	7^{10}	40	22.72	3.83		1^{30}	1140	26.19	7.30
	7^{20}	50	22.91	4.02		2^{30}	1200	26.26	7.37
	7^{30}	60	23.13	4.24		3^{30}	1260	26.30	7.41
	7^{40}	70	23.35	4.46		4^{30}	1320	26.35	7.46
	7^{50}	80	23.49	4.60		5^{30}	1380	26.40	7.51
	8^{00}	90	23.66	4.77		6^{30}	1440	26.46	7.57
	8^{30}	120	23.98	5.09		7^{30}	1500	26.57	7.68
	9^{00}	150	24.24	5.35		8^{30}	1560	26.56	7.67
	9^{30}	180	24.47	5.58		9^{30}	1620	26.59	7.70
	10^{00}	210	24.68	5.79		10^{30}	1680	26.63	7.74

PTO...

48 Hour Constant Rate Pumping Test (cont.)

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Recovery Date	Recovery Elapsed Time Hours	Recovery Elapsed Time Minutes	Recovery Water Level m	Recovery Drawdown m
	11^{30}	1740	26.67	7.78	3/10/88		1	27.08	8.19
	12^{30}	1800	26.73	7.84			2	26.79	7.90
	1^{30}	1860	26.77	7.88			3	26.55	7.66
	2^{30}	1920	26.82	7.93			4	26.36	7.47
	3^{30}	1980	26.85	7.96			5	26.26	7.37
	4^{30}	2040	26.90	8.01			6	26.18	7.29
	5^{30}	2100	26.95	8.06			7	26.09	7.20
	6^{30}	2160	26.98	8.09			8	26.01	7.12
	7^{30}	2220	26.99	8.10			9	25.96	7.07
	8^{30}	2280	27.07	8.18			10	25.91	7.02
	9^{30}	2340	27.12	8.23			15	25.65	6.76
	10^{30}	2400	27.14	8.25			20	25.50	6.61
	11^{30}	2460	27.16	8.27			25	25.38	6.49
30/10/88	12^{30}	2520	27.15	8.26			30	25.24	6.35
	1^{30}	2580	27.18	8.29			35	–	–
	2^{30}	2640	27.21	8.32			40	25.05	6.16
	3^{30}	2700	27.24	8.35			50	25.03	6.14
	4^{30}	2760	27.28	8.39			60	24.72	5.83
	5^{30}	2820	27.38	8.49			70		
	6^{30}	2880	27.39	8.50			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

BOREFIELD MONITORING. PUMPED BORE : BW 1P

		BW. 8		BW 9		BW 2			
DATE	TIME (min)	WATER LEVEL (m)	DRAW DOWN (m)	WATER LEVEL (m)	DRAW DOWN (m)	WATER LEVEL (m)	DRAW DOWN (m)	WATER LEVEL (m)	DRAWDOWN (m)
W.L	0	24.41	–	24.32	–	–	–		
	1 HR	–	–	–	–	–	0.01		
	4 HRS	24.41	0	24.32	0	–	0.01		
	10 HRS	24.41	0	24.32	0	–	0.01		
	24 HRS	24.42	0.01	24.35	0.03	–	0.05		
	29 HRS	24.42	0.01	24.35	0.03	–	0.05		
	34 HRS	24.41	0	24.33	0.01	–	0.065		
	37 HRS	–	–	–	–	–	0.07		
	48 HRS	24.42	0.01	24.33	0.01	–	0.09		

Rockwater
PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

48 Hour Constant Rate Pumping Test

BORE NO: BW2P .... CLIENT: BILLITON ....

DATE: 10/11/88 .... PUMP INLET SETTING: 56m ....

S.W.L. (m below dip tube): 26.47 .... AVAILABLE DRAWDOWN: 29.53 m ....

DIP TUBE HEIGHT ABOVE COLLAR: .... WEIR PIPE DIAMETER: 3" ....

COLLAR HEIGHT ABOVE GROUND: 0.48 .... PUMPING RATE: 300 m³/d ....

ORIFICE PLATE DIAMETER: 2" .... MANOMETER TUBE HEIGHT: 11.6 ....

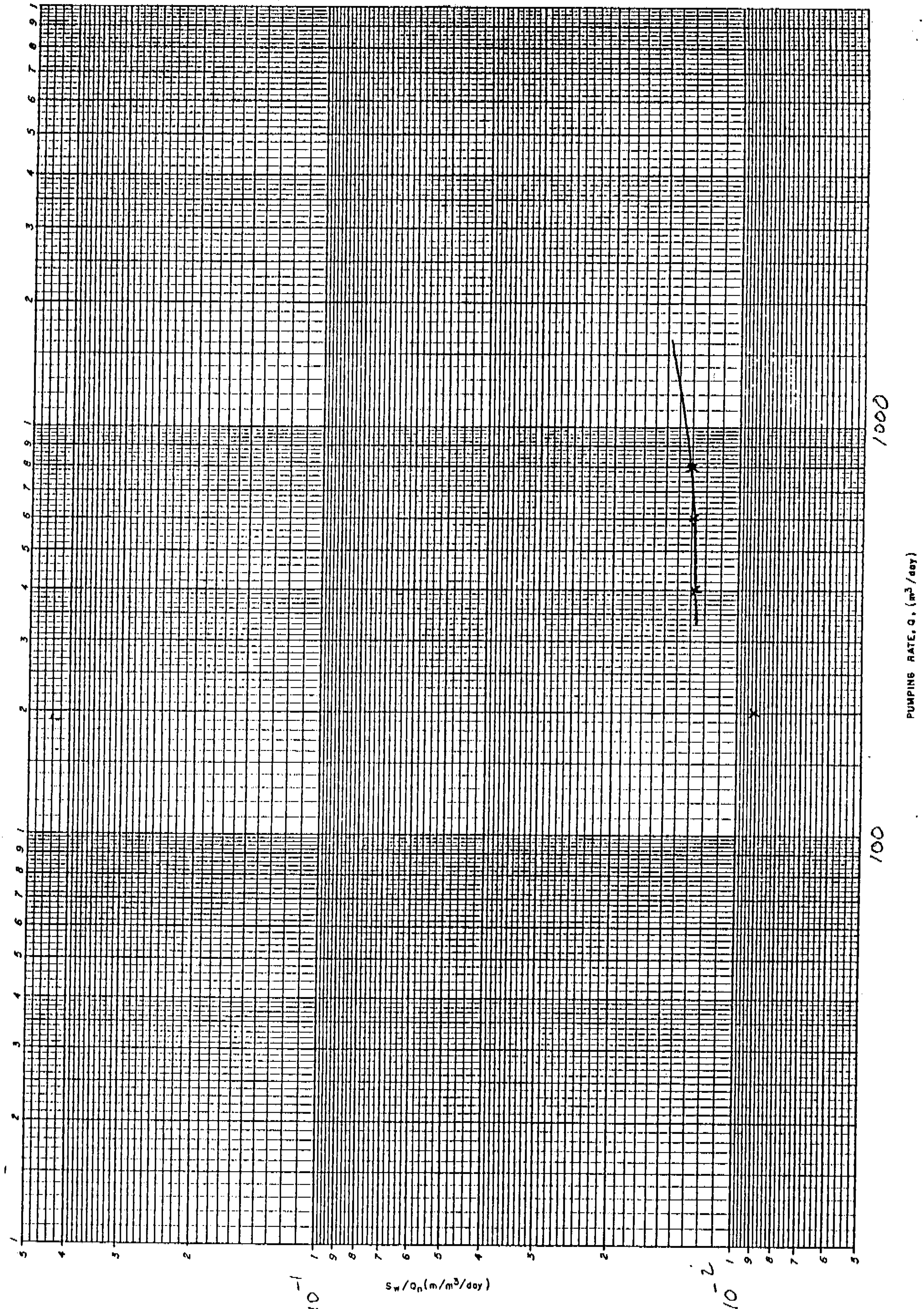
START TIME: 1.00 pm ....

3 FUEL PROBLEMS.

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
10/11/88		1	29.85	3.38		5^{00}	240	31.54	5.07
		2	30.29	3.82		6^{00}	300	31.79	5.32
		3	30.72	4.25		7^{00}	360	31.82	5.35
		4	30.91	4.44		8^{00}	420	31.83	5.36
		5	30.99	4.52		9^{00}	480	31.91	5.44
		6	31.10	4.63		10^{00}	540	31.96	5.49
		7	31.16	4.69		11^{00}	600	31.93	5.46
		8	31.22	4.75	11/11/88	12^{00}	660	31.91	5.44
		9	31.26	4.79		1^{00}	720	31.98	5.51
	1^{10}	10	31.34	4.87		2^{00}	780	32.03	5.56
	1^{15}	15	31.45	4.98		3^{00}	840	32.04	5.57
	1^{20}	20	31.32	4.85		4^{00}	900	32.04	5.57
	1^{25}	25	31.29	4.82		5^{00}	960	32.03	5.56
	1^{30}	30	31.31	4.84		6^{00}	1020	32.05	5.58
	1^{35}	35	31.36	4.89		7^{00}	1080	32.06	5.59
	1^{40}	40	31.43	4.96		8^{00}	1140	32.11	5.64
	1^{50}	50	31.35	4.88		9^{00}	1200	32.10	5.63
	2^{00}	60	31.32	4.85		10^{00}	1260	32.17	5.70
	2^{10}	70	31.47	5.00		11^{00}	1320	32.24	5.77
	2^{20}	80	31.41	4.94		12^{00}	1380	32.24	5.77
	2^{30}	90	31.43	4.96		1^{00}	1440	32.21	5.74
	3^{00}	120	31.38	4.91		2^{00}	1500	32.23	5.76
	3^{30}	150	31.46	4.99		3^{00}	1560	32.25	5.78
	4^{00}	180	31.55	5.08		4^{00}	1620	32.28	5.81
	4^{30}	210	31.64	5.17		5^{00}	1680	32.28	5.81

PTO...

FIGURE



Client : Billiton Australia P/L

Project: Batman Water Supply

Date : Dec 1988 Dwg. No.

Bore BW 8P.

Sheahan Analysis

$P = 2.0$

$S_w/Q_x = 0.023$

$Q_x = 6000$

48 Hour Constant Rate Pumping Test (cont.)

					Recovery				
Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
	6^{00}	1740	32.32	5.85			1	28.15	1.68
	7^{00}	1800	32.33	5.86			2	27.74	1.27
	8^{00}	1860	32.38	5.91			3	27.71	1.24
	9^{00}	1920	32.37	5.90			4	27.71	1.24
	10^{00}	1980	32.33	5.86			5	27.71	1.24
	11^{00}	2040	32.34	5.87			6	27.71	1.24
12/11/88	12^{00}	2100	32.46	5.99			7	27.69	1.22
	1^{00}	2160	32.35	5.88			8	27.68	1.21
	2^{00}	2220	32.35	5.88			9	27.68	1.21
	3^{00}	2280	32.20	5.82			10	27.65	1.18
	4^{00}	2340	32.28	5.81			15	27.59	1.12
	5^{00}	2400	32.33	5.86			20	27.56	1.09
	6^{00}	2460	32.34	5.87			25	27.52	1.05
	7^{00}	2520	32.45	5.98			30	27.52	1.05
	8^{00}	2580	32.50	6.03			35	27.46	0.99
	9^{00}	2640	32.44	5.97			40	27.46	0.99
	10^{00}	2700	32.40	5.93			50	27.42	0.95
	11^{00}	2760	32.40	5.93			60	27.40	0.93
	12^{00}	2820	32.39	5.92			70		
	1^{00}	2880	32.36	5.89			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

48 Hour Constant Rate Pumping Test

BORE NO: BW2 .... CLIENT: Billiton Gold

DATE: 19/11/88 .... PUMP INLET SETTING: –

S.W.L. (m below dip tube): 26.92 .... AVAILABLE DRAWDOWN: –

DIP TUBE HEIGHT ABOVE COLLAR: – .... WEIR PIPE DIAMETER: –

COLLAR HEIGHT ABOVE GROUND: .... PUMPING RATE: –

ORIFICE PLATE DIAMETER: – .... MANOMETER TUBE HEIGHT: –

START TIME: 1.00 pm

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
		1	27.44	0.52		5.00	240	28.32	1.40
		2	27.59	0.67		6.00	300	28.41	1.49
		3	27.75	0.83		7.00	360	28.43	1.51
		4	27.83	0.91		8.00	420	28.49	1.57
		5	27.86	0.94		9.00	480	28.51	1.59
		6	27.90	0.98		10.00	540	28.56	1.64
		7	27.93	1.01		11.00	600	28.58	1.66
		8	27.95	1.03		12.00	660	28.60	1.68
		9	27.97	1.05		1.00	720	28.63	1.71
	1.10	10	27.98	1.06		2.00	780	28.65	1.73
	1.15	15	–	–		3.00	840	28.68	1.76
	1.20	20	28.04	1.12		4.00	900	28.69	1.77
	1.25	25	28.05	1.13		5.00	960	28.71	1.79
	1.30	30	28.06	1.14		6.00	1020	28.75	1.83
	1.35	35	28.08	1.16		7.00	1080	28.75	1.83
	1.40	40	28.10	1.18		8.00	1140	28.77	1.85
	1.50	50	28.11	1.19		9.00	1200	28.77	1.85
	2.00	60	28.12	1.20		10.00	1260	28.80	1.88
	2.10	70	28.21	1.29		11.00	1320	28.84	1.92
	2.20	80	28.16	1.24		12.00	1380	28.83	1.91
	2.30	90	28.17	1.25		1.00	1440	28.84	1.92
	3.00	120	28.19	1.27		2.00	1500	28.85	1.93
	3.30	150	28.24	1.32		3.00	1560	28.86	1.94
	4.00	180	28.27	1.35		4.00	1620	28.87	1.95
	4.30	210	28.32	1.40		5.00	1680	28.89	1.97

PTO...

48 Hour Constant Rate Pumping Test (cont.)

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Recovery Elapsed Time Hours	Recovery Elapsed Time Minutes	Water Level m	Drawdown m
	6^{00}	1740	28.91	1.99			1	28.69	1.77
	7^{00}	1800	28.93	2.01			2	28.32	1.40
	8^{00}	1860	28.96	2.04			3	28.19	1.27
	9^{00}	1920	28.97	2.05			4	28.21	1.29
	10^{00}	1980	28.98	2.06			5	28.15	1.23
	11^{00}	2040	29.00	2.08			6	28.11	1.19
	12^{00}	2100	29.04	2.12			7	28.12	1.20
	1^{00}	2160	29.01	2.09			8	28.11	1.19
	2^{00}	2220	29.01	2.09			9	28.10	1.18
	3^{00}	2280	29.01	2.09			10	28.09	1.17
	4^{00}	2340	29.02	2.10			15	28.06	1.14
	5^{00}	2400	29.03	2.11			20	27.98	1.06
	6^{00}	2460	29.04	2.[illegible]			25	27.95	1.03
	7^{00}	2520	29.08	2.16			30	27.95	1.03
	8^{00}	2580	29.10	2.18			35	27.92	1.00
	9^{00}	2640	29.12	2.20			40	27.90	0.98
	10^{00}	2700	29.11	2.19			50	27.86	0.94
	11^{00}	2760	29.10	2.18			60	27.85	0.93
	12^{00}	2820	29.09	2.17			70		
	1^{00}	2880	29.09	2.17			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

48 Hour Constant Rate Pumping Test

BORE NO: BW 6P CLIENT: BILLITON

DATE: 31/10/88 PUMP INLET SETTING: 57m

S.W.L. (m below dip tube): 19.21 AVAILABLE DRAWDOWN:

DIP TUBE HEIGHT ABOVE COLLAR: 0.03 WEIR PIPE DIAMETER: 3"

COLLAR HEIGHT ABOVE GROUND: 0.42 PUMPING RATE: 350

ORIFICE PLATE DIAMETER: 2" MANOMETER TUBE HEIGHT: 15.8"

START TIME: 06^{30}

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
31/10/88		1	22.54	3.33		10^{30}	240	25.79	6.58
		2	23.25	4.04		11^{30}	300	25.94	6.73
		3	23.76	4.55		12^{30}	360	26.10	6.89
		4	24.08	4.87		1^{30}	420	26.23	7.02
		5	24.33	5.12		2^{30}	480	26.30	7.09
		6	24.45	5.24		3^{30}	540	26.38	7.17
		7	24.55	5.34		4^{30}	600	26.47	7.26
		8	24.64	5.43		5^{30}	660	26.56	7.35
		9	24.70	5.49		6^{30}	720	26.62	7.41
	06^{40}	10	24.75	5.54		7^{30}	780	26.70	7.49
	06^{45}	15	24.90	5.69		8^{30}	840	26.77	7.56
	06^{50}	20	24.99	5.78		9^{30}	900	26.87	7.66
	06^{55}	25	25.04	5.83		10^{30}	960	26.91	7.75
	07^{00}	30	25.09	5.88		11^{30}	1020	26.98	7.74
	07^{05}	35	25.11	5.90	1/11/88	12^{30}	1080	26.99	7.78
	07^{10}	40	25.13	5.92		1^{30}	1140	27.05	7.84
	07^{20}	50	25.14	5.93		2^{30}	1200	27.08	7.87
	07^{30}	60	25.20	5.99		3^{30}	1260	27.11	7.90
	07^{40}	70	25.24	6.03		4^{30}	1320	27.14	7.93
	07^{50}	80	25.30	6.09		5^{30}	1380	27.20	7.99
	08^{00}	90	25.31	6.10		6^{30}	1440	27.30	8.09
	08^{30}	120	25.46	6.25		7^{30}	1500	27.34	8.13
	09^{00}	150	25.53	6.32		8^{30}	1560	27.38	8.17
	09^{30}	180	25.62	6.41		9^{30}	1620	27.41	8.20
	10^{00}	210	25.69	6.48		10^{30}	1680	27.44	8.23

PTO...

48 Hour Constant Rate Pumping Test (cont.)

Recovery

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
	11 30	1740	27.47	8.26	3/11/88		1	23.57	4.36
	12 30	1800	27.50	8.29			2	23.20	3.99
	1 30	1860	27.50	8.29			3	23.00	3.79
	2 30	1920	27.56	8.35			4	22.90	3.69
	3 30	1980	27.58	8.37			5	22.87	3.66
	4 30	2040	27.65	8.44			6	22.81	3.60
	5 30	2100	27.64	8.43			7	22.78	3.57
	6 30	2160	27.67	8.46			8	22.76	3.55
	7 30	2220	27.80	8.59			9	22.74	3.53
	8 30	2280	27.80	8.59			10	22.70	3.49
	9 30	2340	27.80	8.59			15	22.63	3.42
	10 30	2400	27.87	8.66			20	22.50	3.29
	11 30	2460	27.86	8.65			25	22.43	3.22
2/11/88	12 30	2520	27.86.	8.65			30	22.35	3.14
	1 30	2580	27.91	8.70			35	22.27	3.06
	2 30	2640	27.90	8.69			40	22.22	3.01
	3 30	2700	27.92	8.71			50	22.08	2.87
	4 30	2760	27.93	8.72			60	22.00	2.79
	5 30	2820	27.98	8.77			70		
	6 30	2880	28.04	8.83			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater PROPRIETARY LIMITED **GROUNDWATER CONSULTANTS**

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

48 Hour Constant Rate Pumping Test

BORE NO: CABW 6 .... CLIENT: [illegible]

DATE: 31/10/88 .... PUMP INLET SETTING: ......

S.W.L. (m below dip tube): 19.46 ...... AVAILABLE DRAWDOWN: ......

DIP TUBE HEIGHT ABOVE COLLAR: – ...... WEIR PIPE DIAMETER: – ......

COLLAR HEIGHT ABOVE GROUND: 0.59 ...... PUMPING RATE: ......

ORIFICE PLATE DIAMETER: – ...... MANOMETER TUBE HEIGHT: – ......

START TIME: 06^{30} ......

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
31/10/88		1	19.66	0.20		10^{30}	240	21.02	1.56
		2	19.80	0.34		11^{30}	300	21.14	1.68
		3	19.91	0.45		12^{30}	360	21.30	1.84
		4	19.99	0.53		1^{30}	420	21.34	1.88
		5	20.05	0.59		2^{30}	480	21.43	1.97
		6	20.10	0.64		3^{30}	540	21.52	2.06
		7	20.15	0.69		4^{30}	600	21.61	2.15
		8	20.17	0.71		5^{30}	660	21.67	2.21
		9	20.20	0.74		6^{30}	720	21.72	2.26
	06^{40}	10	20.21	0.75		7^{30}	780	21.84	2.38
	06^{45}	15	20.30	0.84		8^{30}	840	[illegible]	2.41
	06^{50}	20	20.34	0.88		9^{30}	900	21.87	2.45
	06^{55}	25	20.37	0.91		10^{30}	960	21.91	2.50
	07^{00}	30	20.41	0.95		11^{30}	1020	21.96	2.57
	07^{05}	35	20.44	0.98	1/11/88	12^{30}	1080	22.03	2.63
	07^{10}	40	20.46	1.00		1^{30}	1140	22.09	2.57
	07^{20}	50	20.50	1.04		2^{30}	1200	22.03	2.70
	07^{30}	60	20.54	1.08		3^{30}	1260	22.16	2.82
	07^{40}	70	20.59	1.13		4^{30}	1320	22.22	2.86
	07^{50}	80	20.62	1.16		5^{30}	1380	22.32	2.84
	08^{00}	90	20.64	1.18		6^{30}	1440	22.30	2.85
	08^{30}	120	20.74	1.28		7^{30}	1500	22.31	2.89
	09^{00}	150	20.82	1.36		8^{30}	1560	22.35	2.92
	09^{30}	180	20.89	1.43		9^{30}	1620	22.38	2.95
	10^{00}	210	20.95	1.49		10^{30}	1680	22.41	2.96

PTO...

48 Hour Constant Rate Pumping Test (cont.)

Recovery

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown
	11^{30}	1740	22.42	~~22.46~~ 3.00	2/11/88		1	22.70	3.24
	12^{30}	1800	[illegible]	3.02			2	22.63	3.17
	1^{30}	1860	[illegible]	3.09			3	22.52	3.06
	2^{30}	1920	[illegible]	3.08			4	22.51	3.05
	3^{30}	1980	[illegible]	3.14			5	22.49	3.03
	4^{30}	2040	[illegible]	3.14			6	22.47	3.01
	5^{30}	2100	22.66	3.20			7	22.46	3.00
	6^{30}	2160	22.65	3.19			8	22.45	2.99
	7^{30}	2220	22.72	3.26			9	22.43	2.97
	8^{30}	2280	22.72	3.26			10	—	—
	9^{30}	2340	22.76	3.30			15	22.39	2.93
	10^{30}	2400	22.77	3.31			20	22.35	2.89
	11^{30}	2460	22.80	3.34			25	22.31	2.85
2/11/88	12^{30}	2520	22.80	3.34			30	22.29	2.83
	1^{30}	2580	22.85	3.39			35	22.25	2.79
	2^{30}	2640	22.87	3.41			40	22.20	2.74
	3^{30}	2700	22.90	3.44			50	22.15	2.69
	4^{30}	2760	22.91	3.45			60	22.09	2.63
	5^{30}	2820	22.92	3.46			70		
	6^{30}	2880	22.98	3.52			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

94 ROKEBY ROAD, SUBIACO, WESTERN AUSTRALIA 6008. TELEPHONE (09) 382 4922

HOUR CONSTANT RATE PUMPING TEST

CLIENT : BILLITON GOLD

DATE : 14/11/88

BORE NO : BLO8P

BORE TYPE: PRODUCTION/~~OBSERVATION~~

A. DEPTH TO WATER (m below dip tube): 24.50

D. PUMP INLET SETTING: 56m

~~B. HEIGHT OF DIP TUBE ABOVE GROUND LEVEL:~~

AVAILABLE DRAWDOWN (D-C): 31.50

C. STATIC WATER LEVEL (A-B) (mbgl):

WEIR PIPE DIAMETER: 3"

HEIGHT OF CASING ABOVE GROUND: 0.80

MANOMETER TUBE HEIGHT: 5"

START TIME: 7:10AM

PUMPING RATE: 400 m³/d

ORIFICE PLATE: 2½"

DATE	ELAPSED TIME - TIME	ELAPSED TIME - MINUTES	WATER LEVEL (m)	DRAWDOWN (m)	DATE	ELAPSED TIME - TIME	ELAPSED TIME - MINUTES	WATER LEVEL (m)	DRAWDOWN (m)
14/11/88		1	27.10	2.60		11:10	240	30.08	5.58
		2	27.59	3.09		12:10	300	30.18	5.68
		3	28.33	3.83		1:10	360	30.28	5.78
		4	–	–		2:10	420	30.38	5.88
		5	28.80	4.30		3:10	480	30.46	5.96
		6	28.90	4.40		4:10	540	30.55	6.05
		7	28.94	4.44		5:10	600	30.65	6.15
		8	29.07	4.57		6:10	660	30.69	6.19
		9	29.17	4.67		7:10	720	30.75	6.25
	7:20	10	29.27	4.77		8:10	780	30.79	6.29
	7:25	15	29.38	4.88		9:10	840	30.86	6.36
	7:30	20	29.48	4.98		10:10	900	30.93	6.43
	7:35	25	29.55	5.05		11:10	960	31.03	6.53
	7:40	30	29.60	5.10		12:10	1020	31.04	6.54
	7:45	35	29.60	5.10		1:10	1080	31.10	6.60
	7:50	40	29.64	5.14		2:10	1140	31.21	6.71
	8:00	50	29.68	5.18		3:10	1200	31.23	6.73
	8:10	60	29.70	5.20		4:10	1260	31.29	6.79
	8:20	70	29.73	5.23		5:10	1320	31.35	6.85
	8:30	80	29.76	5.26		6:10	1380	31.39	6.89
	8:40	90	29.79	5.29		7:10	1440	31.43	6.93
	8:55	105	29.82	5.32		8:10	1500	31.50	7.00
	9:10	120	29.88	5.38		9:10	1560	31.55	7.05
	9:40	150	29.98	5.48		10:10	1620	31.52	7.02
	10:10	180	30.00	5.50		11:10	1680	31.56	7.06
	10:40	210	30.06	5.56	15/11/88	12:10	1740	31.63	7.13

Day Constant Rate Pumping Test (cont.)

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Recovery Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
	1^{10}	1800	31·62	7·12			1	29·32	
	2^{10}	1860	31·65	7·15			2		
	3^{10}	1920	31·69	7·19			3		
	4^{10}	1980	31·74	7·24			4		
	5^{10}	2040	31·78	7·28			5		
	6^{10}	2100	31·83	7·33			6		
	7^{10}	2160	31·85	7·35			7		
	8^{10}	2220	31·90	7·40			8		
	9^{10}	2280	31·93	7·43			9		
	10^{10}	2340	31·97	7·47			10		
	11^{10}	2400	32·01	7·51			15		
16/11/88	12^{10}	2460	32·04	7·54			20		
	1^{10}	2520	32·06	7·58			25		
	2^{10}	2580	32·11	7·61			30		
	3^{10}	2640	32·15	7·65			35		
	4^{10}	2700	32·21	7·71			40		
	5^{10}	2760	32·23	7·73			50		
	6^{10}	2820	32·26	7·76			60		
	7^{10}	2880	32·28	7·78			70		
							80		
							90		
							120		
							150		
							180		

Rockwater PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
94 ROKEBY ROAD, SUBIACO, WESTERN AUSTRALIA 6008. TELEPHONE (09) 382 4922

___ HOUR CONSTANT RATE PUMPING TEST

CLIENT : BILLITON GOLD

DATE : 13/11/88

BORE NO : BW8

BORE TYPE: ~~PRODUCTION~~/OBSERVATION

A. DEPTH TO WATER (m below ~~dip tube~~ casing): 25·01

D. PUMP INLET SETTING: –

B. HEIGHT OF DIP TUBE ABOVE GROUND LEVEL: –

AVAILABLE DRAWDOWN (D-C): –

C. STATIC WATER LEVEL (A-B) (mbgl): 24·65

WEIR PIPE DIAMETER: –

HEIGHT OF CASING ABOVE GROUND: 6" 0·36 2" 0·33

MANOMETER TUBE HEIGHT: –

START TIME:

PUMPING RATE: –

DATE	ELAPSED TIME – TIME	ELAPSED TIME – MINUTES	WATER LEVEL (m)	DRAWDOWN (m)	DATE	ELAPSED TIME – TIME	ELAPSED TIME – MINUTES	WATER LEVEL (m)	DRAWDOWN (m)
13/11/88	7·10	1	~~25.31~~	–		11·10	240	26·90	1·89
		2	25·31	0·30		12·10	300	26·90	1·89
		3	25·37	0·36		1·10	360	27·24	2·23
		4	25·42	0·41		2·10	420	27·40	2·39
		5	25·45	0·44		3·10	480	27·54	2·53
		6	25·49	0·48		4·10	540	27·66	2·65
		7	25·52	0·51		5·10	600	27·74	2·73
		8	25·57	0·56		6·10	660	27·83	2·82
		9	25·57	0·56		7·10	720	27·91	2·90
		10	25·59	0·58		8·10	780	27·98	2·97
		15	25·71	0·70		9·10	840	28·04	3·03
		20	25·77	0·76		10·10	900	28·13	3·12
		25	25·85	0·84		11·10	960	28·20	3·19
		30	25·90	0·89		12·10	1020	28·25	3·24
		35	25·97	0·96		1·10	1080	28·30	3·29
		40	26·03	1·02		2·10	1140	28·35	3·34
		50	26·08	1·07		3·10	1200	28·41	3·40
		60	26·15	1·14		4·10	1260	28·50	3·49
		70	26·23	1·22		5·10	1320	28·51	3·50
		80	26·26	1·25		6·10	1380	28·54	3·53
		90	26·32	1·31		7·10	1440	28·58	3·57
		105	26·42	1·41		8·10	1500	28·66	3·65
	9·10	120	26·54	1·53		9·10	1560	28·68	3·67
	9·40	150	26·62	1·61		10·10	1620	28·77	3·76
	10·10	180	26·73	1·72		11·10	1680	28·85	3·84
	10·40	210	26·83	1·82		12·10	1740	28·92	3·91

BOREFIELD MONITORING. PUMPED BORE:

DATE	TIME (min)	WATER LEVEL (m)	DRAW DOWN (m)	WATER LEVEL (m)	DRAW DOWN (m)	RECOVERY WATER LEVEL (m)	DRAW DOWN (m)	WATER LEVEL (m)	DRAWDOWN (m)
	1800	28.94	3.93		1	29.07	4.06		
	1860	28.91	3.90		2	28.20	3.69		
	1920	29.00	3.99		3	28.42	3.41		
	1980	29.9	4.00		4	28.29	3.28		
	2040	29.04	4.03		5	28.20	3.19		
	2100	29.09	4.08		6	28.13	3.12		
	2160	29.12	4.11		7	28.09	3.08		
	2220	29.19	4.18		8	28.06	3.05		
	2280	29.19	4.18		9	28.03	3.02		
	2340	29.23	4.22		10	28.00	2.99		
	2400	29.26	4.25		15	27.97	2.96		
	2460	29.29	4.28		20	27.92	2.91		
	2520	29.34	4.33		25	27.84	2.83		
	2580	29.40	4.37		30	27.83	2.82		
	2640	29.39	4.38		35	27.80	2.79		
	2700	29.48	4.47		40	27.80	2.79		
	2760	29.52	4.51		45	27.78	2.77		
	2820	29.60	4.59		50	27.79	2.78		
	2880	29.53	4.52		55	27.74	2.73		
					60				

Rockwater
PROPRIETARY LIMITED
GROUNDWATER CONSULTANTS
ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

48 Hour Constant Rate Pumping Test

BORE NO: BW1OP CLIENT: BILLITON AUST

DATE: 3/11/88 PUMP INLET SETTING: 56m

S.W.L. (m below dip tube): 22.52 AVAILABLE DRAWDOWN: 33.48m

DIP TUBE HEIGHT ABOVE COLLAR: 0.02 WEIR PIPE DIAMETER: 3"

COLLAR HEIGHT ABOVE GROUND: 0.36 PUMPING RATE: 250 m³/d

ORIFICE PLATE DIAMETER: 2" MANOMETER TUBE HEIGHT: 8.1"

START TIME: 8.00pm

Date	Elapsed Time Hours	Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Minutes	Water Level m	Drawdown m
3/11/88		1	24.50	1.98	4/11/88	12.00	240	38.71	16.19
		2	25.03	2.51		1.00	300	39.20	16.68
		3	25.60	3.08		2.00	360	39.45	~~18.73~~ 16.9
		4	26.18	3.66		3.00	420	39.78	17.26
		5	26.69	4.17		4.00	480	39.89	17.37
		6	27.09	4.57		5.00	540	39.93	17.41
		7	27.25	4.73		6.00	600	40.14	17.62
		8	27.47	4.95		7.00	660	40.27	17.75
		9	27.69	5.17		8.00	720	40.48	17.96
	8.10pm	10	27.87	5.35		9.00	780	40.53	18.01
	8.15	15	28.60	6.08		10.00	840	40.59	18.07
	8.20	20 ↑	29.05	6.53		11.00	900	40.70	18.18
	8.25	25	29.95	7.43	~~4/11/88~~	12.00	960	40.88	18.36
	8.30	30	30.90	8.38		1.00	* 1020 ↑	40.87	18.35
	8.35	35	31.57	9.05		2.00	1080	41.13	18.61
	8.40	40 ↑	32.04	9.52		3.00	1140	41.18	18.66
	8.50	50	32.94	10.42		4.00	1200	41.30	18.78
	9.00	60	33.72	11.20		5.00	1260	41.40	18.88
	9.10	70	34.26	11.74		6.00	1320	41.46	18.94
	9.20	80	34.75	12.23		7.00	1380	41.57	19.05
	9.30	90	35.13	12.61		8.00	1440	41.63	19.11
	10.00	120	36.26	13.74		9.00	1500	41.63	19.11
	10.30	150	36.99	14.47		10.00	1560	41.73	19.21
	11.00	180	37.62	15.10		11.00	1620	41.72	19.20
	11.30	210	38.13	15.61	5/11/88	12.00	1680	41.55	19.03

* NOTE - MOTOR REVS FLUCTUATING - NOT SURE WHY?... SLIGHTLY HOT.

48 Hour Constant Rate Pumping Test (cont.)

					Recovery				
Date	Elapsed Time Hours	Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Minutes	Water Level m	Drawdown m
5/11/88	1^{00}	1740	41.68	19.16			1		
	2^{00}	1800	41.87	19.35			2		
	3^{00}	1860	41.88	19.36			3		
	4^{00}	1920	41.73	19.21			4	~~38.87~~	38.87
	5^{00}	1980	41.83	19.31			5	38.25	
	6^{00}	2040	42.08	19.56			6		
	7^{00}	2100	42.18	19.66			7	36.79	
	8^{00}	2160	42.14	19.62			8		
	9^{00}	2220	42.05	19.53			9	36.45	
	10^{00}	2280	42.16	19.64			10	36.28	
	11^{00}	2340	~~[illegible]~~	~~[illegible]~~			15	35.95	
	12^{00}	2400					20	35.58	
	1^{00}	2460					25	35.20	
	2^{00}	2520					30	34.90	
	3^{00}	2580					35	34.62	
	4^{00}	2640					40	34.38	
	5^{00}	2700					50	33.94	
	6^{00}	2760					60	33.57	
	7^{00}	2820					70		
	8^{00}	2880					80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater
PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

37.42

48 Hour Constant Rate Pumping Test

BORE NO:....BW.10.................... CLIENT:...Billiton Aust.........

DATE:...3/11/88.................... PUMP INLET SETTING:...–..........

S.W.L. (m below dip tube):..23.28....... AVAILABLE DRAWDOWN:...–..........

DIP TUBE HEIGHT ABOVE COLLAR:............ WEIR PIPE DIAMETER:...–..........

COLLAR HEIGHT ABOVE GROUND:.............. PUMPING RATE:...–..........

ORIFICE PLATE DIAMETER:...–.............. MANOMETER TUBE HEIGHT:...–..........

START TIME:...8:00pm...............

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	
3/11/88		1	23.51m	0.23	4/11/88	12.00am	240	36.53	13.25	
		2	24.10	0.82		1.00	300	37.04	13.76	
		3	24.41	1.13		2.00	360	37.42	14.14	
		4	24.96	1.68		3.00	420	37.70	[illegible]	14.9
		5	25.19	1.91		4.00	480	37.82	[illegible]	14.5
		6	25.45	2.17		5.00	540	[illegible]	[illegible]	
		7	25.72	2.44		6.00	600	38.01	[illegible]	14.7
		8	25.79	2.51		7.00	660	38.15	14.87	
		9	25.83	2.55		8.00	720	38.33	15.05	
	8.10pm	10	25.83	2.55		9.00	780	38.51	15.23	
	8.15	15	26.02	2.74		10.00	840	38.56	15.28	
	8.20	20	26.40	3.12		11.00	900	38.78	15.50	
	8.25	25	28.15	4.87		12.00pm	960	38.81	15.53	
	8.30	30	29.06	5.78		1.00	1020	38.90	15.62	
	8.35	35	29.73	6.45		2.00	1080	39.09	15.81	
	8.40	40	30.16	6.88		3.00	1140	39.23	15.95	
	8.50	50	31.05m	7.77		4.00	1200	39.32	16.04	
	9.00	60	31.80	8.52		5.00	1260	39.40	16.12	
	9.10	70	32.27	8.99		6.00	1320	39.49	16.21	
	9.20	80	32.81	9.53		7.00	1380	39.57	16.29	
	9.30	90	33.31	10.03		8.00	1440	39.63	16.35	
	10.00	120	34.31	11.03		9.00	1500	39.67	16.39	
	10.30	150	35.00	11.72		10.00	1560	39.74	16.46	
	11.00	180	35.72	12.44		11.00	1620	39.73	16.45	
	11.30	210	36.13	12.85	5 Nov 88	12.00am	1680	39.67	16.39	

PTO...

48 Hour Constant Rate Pumping Test (cont.)

					Recovery				
Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
5/10/88	1.00am	1740	39.77 ~~39.88~~	16.49			1		
	2.00	1800	39.83	16.55			2		
	3.00	1860	39.64	16.36			3		
	4.00	1920	39.82	16.54			4	38.10	
	5.00	1980	39.91	16.63			5	37.56	
	6.00	2040	40.07	16.79			6	~~37.2~~	
	7.00	2100	40.14	16.86			7	37.20	
	8.00	2160	40.15	16.87			8		
	9.00	2220	40.12	16.84			9	37.01	
	10.00	2280	40.19	16.91			10	37.83	
	11.00	2340	1				15	36.56	
	12.00	2400					20	36.10	
	1300	2460					25	35.75	
	1400	2520					30	35.50	
	1500	2580					35	35.21	
	1600	2640					40	34.99	
	1700	2700					50	34.59	
	1800	2760					60	34.21	
	1900	2820					70		
	2000	2880					80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

Rockwater
PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

ADDRESS: 94 ROKEBY ROAD, SUBIACO WA 6008. TEL: 382 4922

48 Hour Constant Rate Pumping Test

BORE NO: RW.10P CLIENT: BILLITON AUST

DATE: 7/11/88 PUMP INLET SETTING: 56m

S.W.L. (m below dip tube): 23.03 AVAILABLE DRAWDOWN: 32.97

DIP TUBE HEIGHT ABOVE COLLAR: } 0.38 WEIR PIPE DIAMETER: 3"

COLLAR HEIGHT ABOVE GROUND: } 0.38 PUMPING RATE: 250 m³/d

ORIFICE PLATE DIAMETER: 2" MANOMETER TUBE HEIGHT: 8.10"

START TIME: 16:30

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
7NOV88	1631 ~~1630~~	1	25.10	2.07	7NOV88	2030	240	36.28	13.25
	1632	2	25.78	2.75		2130	300	36.96	13.93
	1633	3	26.40	3.37		2230	360	37.48	14.45
	1634	4	26.87	3.84		2330	420	38.10	15.07
	1635	5	26.72	3.69	8NOV88	0030	480	38.42	15.39
	1636	6	27.05	4.02		0130	540	38.73	15.70
	1637	7	27.45	4.42		0230	600	39.03	16.00
	1638	8	27.80	4.77		0330	660	39.46	16.43
	1639	9	28.02	4.99		0430	720	39.68	16.65
	1640	10	28.25	5.22		0530	780	39.86	16.83
	1645	15	28.77	5.74		0630	840	40.06	17.03
	1650	20	29.15	6.12		0730	900	40.23	17.20
	1655	25	29.64	6.61		0830	960	40.40	17.37
	1700	30	30.33	7.30		0930	1020	40.67	17.64
	1705	35	30.73	7.70		1030	1080	40.84	17.81
	1710	40	31.22	8.19		1130	1140	40.95	17.92
	1720	50	31.75	8.72		1230	1200	41.02	17.99
	1730	60	32.15	9.12		1330	1260	41.09	18.06
	1740	70	32.43	9.40		1430	1320	41.22	18.19
	1750	80	32.77	9.74		1530	1380	41.26	18.23
	1800	90	32.91	9.88		1630	1440	41.69	18.66
	1830	120	33.04	10.01		1730	1500	41.69	18.66
	1900	150	34.73	11.70		1830	1560	41.77	18.74
	1930	180	34.90	11.87		1930	1620	41.77	18.74
	2000	210	35.80	12.77		2030	1680	41.94	18.91

PTO...

					Recovery				
Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
	21^{30}	1740	41.95	18.92			1	39.75	
	22^{30}	1800	41.99	18.96			2	38.88	
	23^{30}	1860	42.04	19.01			3	38.66	
	00^{30}	1920	42.01	18.98			4	38.37	
	01^{30}	1980	42.10	19.07			5	38.15	
	02^{30}	2040	42.12	19.09			6	37.93	
	03^{30}	2100	42.21	19.18			7	37.[illegible]	
	04^{30}	2160	42.15	19.12			8	37.63	
	05^{30}	2220	42.14	19.11			9	37.47	
	06^{30}	2280	42.23	19.20			10	37.36	
	07^{30}	2340	42.28	19.25			15	36.90	
	08^{30}	2400	42.36	19.33			20	36.77	
	09^{30}	2460	42.39	19.36			25	36.22	
	10^{30}	2520	42.43	19.40			30	36.00	
	11^{30}	2580	42.44	19.41			35	35.70	
	12^{30}	2640	42.44	19.41			40	35.48	
	13^{30}	2700	42.48	19.45			50	35.15	
	14^{30}	2760	42.53	19.50			60	34.81	
	15^{30}	2820	42.53	19.50			70		
	16^{30}	2880	42.64	19.61			80		
							90		
							120		

SALINITIES

SAMPLE NO.	TIME TAKEN	EC	TEMP.	TDS

PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

94 ROKEBY ROAD, SUBIACO, WESTERN AUSTRALIA 6008. TELEPHONE (09) 382 4922

48 HOUR CONSTANT RATE PUMPING TEST

CLIENT : BILLITON

DATE : 7NOV88

BORE NO : BW10

BORE TYPE: ~~PRODUCTION~~/OBSERVATION

A. DEPTH TO WATER (m below dip tube): ________

D. PUMP INLET SETTING: —

B. HEIGHT OF DIP TUBE ABOVE GROUND LEVEL: ________

AVAILABLE DRAWDOWN (D-C): ________

C. STATIC WATER LEVEL (A-B) (mbgl): 23.76

WEIR PIPE DIAMETER: ________

HEIGHT OF CASING ABOVE GROUND: ________

MANOMETER TUBE HEIGHT: —

START TIME: 4.30pm

PUMPING RATE: —

DATE	-ELAPSED TIME- TIME	MINUTES	WATER LEVEL (m)	DRAWDOWN (m)	DATE	-ELAPSED TIME- TIME	MINUTES	WATER LEVEL (m)	DRAWDOWN (m)
7NOV88	1631	1	24.35	0.59	7NOV88	2030	240	34.25	10.49
	1632	2	24.70	0.94		2130	300	34.98	11.22
	1633	3	24.98	1.22		2230	360	35.52	11.76
	1634	4	25.20	1.44		2330	420	36.10	12.34
	1635	5	25.55	1.79		0030	480	36.50	12.74
	1636	6	25.64	1.88		0130	540	36.75	12.99
	1637	7	25.69	1.93		0230	600	37.09	13.33
	1638	8	25.76	2.00		0330	660	37.50	13.74
	1639	9	25.80	2.04		0430	720	37.73	13.97
	1640	10	25.82	2.06		0530	780	37.91	14.15
	1645	15	26.12	2.36		0630	840	38.13	14.37
	1650	20	26.44	2.68		0730	900	38.31	14.55
	1655	25	27.54	3.78		0830	960	38.39	14.63
	1700	30	28.39	4.63		0930	1020	38.75	14.99
	1705	35	29.02	5.26		1030	1080	38.98	15.22
	1710	40	29.39	5.63		1130	1140	39.02	15.26
	1720	50	29.85	6.09		1230	1200	39.18	15.42
	1730	60	30.23	6.47		1330	1260	39.20	15.44
	1740	70	31.43	7.67		1430	1320	39.37	15.61
	1750	80	31.67	7.91		1530	1380	39.36	15.60
	1800	90	31.91	8.15		1630	1440	39.70	15.94
	1815	105	—	—		1730	1500	39.77	16.01
	1830	120	32.08	8.32		1830	1560	39.84	16.08
	1900	150	32.66	8.90		1930	1620	39.87	16.11
	1930	180	33.27	9.51		2030	1680	39.99	16.23
	2000	210	33.63	9.87		2130	1740	40.05	16.29

48 HOUR CONSTANT RATE PUMPING TEST (Cont.)

DATE	-ELAPSED TIME-		WATER LEVEL (m)	DRAWDOWN (m)	DATE	RECOVERY -ELAPSED TIME-		WATER LEVEL (m)	DRAWDOWN (m)
	TIME	MINUTES				TIME	MINUTES		
	22^{30}	1740	40.11	16.35			1	39.76	
	23^{30}	1800	40.13	16.37			2	39.42	
	00^{30}	1860	40.14	16.38			3	39.00	
	01^{30}	1920	40.17	16.41			4	38.80	
	02^{30}	1980	40.18	16.42			5	38.63	
	03^{30}	2040	40.15	16.39			6	38.44	
	04^{30}	2100	40.22	16.46			7	38.30	
	05^{30}	2160	40.24	16.48			8	38.17	
	06^{30}	2220	40.33	16.57			9	38.06	
	07^{30}	2280	40.40	16.64			10	37.95	
	08^{30}	2340	40.49	16.73			15	37.50	
	09^{30}	2400	40.51	16.75			20	37.19	
	10^{30}	2460	40.50	16.74			25	36.89	
	11^{30}	2520	40.50	16.74			30	36.56	
	12^{30}	2580	40.52	16.76			35	36.29	
	13^{30}	2640	40.56	16.80			40	36.08	
	14^{30}	2700	40.60	16.84			50	35.77	
	15^{30}	2760	40.62	16.86			60	35.43	
	16^{30}	2820	40.68	16.92			70		
		2880					80		
							90		
							120		

SALINITIES

SAMPLE NUMBER	TIME TAKEN	EC	TEMP	TDS

BP 70p00

Rockwater PROPRIETARY LIMITED

GROUNDWATER CONSULTANTS

94 ROKEBY ROAD, SUBIACO, WESTERN AUSTRALIA 6008. TELEPHONE (09) 382 4922

HOUR CONSTANT RATE PUMPING TEST

CLIENT : ~~BP70~~ BILLITON

DATE : 13/11/88

BORE NO : BP70

BORE TYPE: PRODUCTION/~~OBSERVATION~~

A. DEPTH TO WATER (m below dip tube): 18.00

D. PUMP INLET SETTING: 35m

B. HEIGHT OF DIP TUBE ABOVE GROUND LEVEL: –

AVAILABLE DRAWDOWN (D-C): 17m

C. STATIC WATER LEVEL (A-B) (mbgl): 17.71

WEIR PIPE DIAMETER: –

HEIGHT OF CASING ABOVE GROUND: 0.29

MANOMETER TUBE HEIGHT: –

START TIME: 700am

PUMPING RATE: –

DATE	ELAPSED TIME – TIME	ELAPSED TIME – MINUTES	WATER LEVEL (m)	DRAWDOWN (m)	DATE	ELAPSED TIME – TIME	ELAPSED TIME – MINUTES	WATER LEVEL (m)	DRAWDOWN (m)
		1		0.65		11.00	240	|	1.60
		2		0.72		12.00	300		1.63
		3		0.78		1.00	360		1.58 *
		4		0.81		2.00	420		1.56
		5		0.83		3.00	480		1.56
		6		0.86		4.00	540		1.60
		7		0.88		5.00	600		1.86 *
		8		0.90		6.00	660		2.00
		9		0.91		7.00	720		1.99
	7.10	10		0.93		8.00	780		1.87
	7.15	15		0.97		9.00	840		1.88
	7.20	20		0.99		10.00	900		1.91
	7.25	25		1.03		11.00	960		1.94
	7.30	30		1.06	14/11/88	12.00	1020		1.95
	7.35	35		1.10		1.00	1080		1.97
	7.40	40		1.13		2.00	1140		2.06
	7.50	50		1.16		3.00	1200		2.12
	8.00	60		1.20		4.00	1260		2.12
	8.10	70		1.24		5.00	1320		2.13
	8.20	80		1.26		6.00	1380		2.40 *
	8.30	90		1.28		7.00	1440		2.27
	8.45	105		1.35		8.00	1500		2.32
	9.00	120		1.36		9.00	1560		
	9.30	150		1.44		10.00	1620		
	10.00	180		1.50		11.00	1680		
	10.30	210		1.56		12.00	1740		

* 7.2sec/K 120m³/d

* DRILLERS USING SHOWER. (LESS HEAD, MORE WATER)

* DRILLERS ALTERED PIPELINE ROUTE UP TO DRILLING RIGS. THIS CHANGE THE HEAD OF THE PUMP THUS REDUCING PUMPING RATE.

Day Constant Rate Pumping Test (cont.)

Recovery

Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m	Date	Elapsed Time Hours	Elapsed Time Minutes	Water Level m	Drawdown m
	1^{00}	1800					1		
	2^{00}	1860					2		
	3^{00}	1920					3		
	4^{00}	1980					4		
	5^{00}	2040					5		
	6^{00}	2100					6		
	7^{00}	2160					7		
	8^{00}	2220					8		
	9^{00}	2280					9		
	10^{00}	2340					10		
	11^{00}	2400					15		
15/11/88	12^{00}	2460					20		
	1^{00}	2520					25		
	2^{00}	2580					30		
	3^{00}	2640					35		
	4^{00}	2700					40		
	5^{00}	2760					50		
	6^{00}	2820					60		
	7^{00}	2880					70		
							80		
							90		
							120		
							150		
							180		

SECTION 3

FIELD NOTEBOOK DATA

Billiton Aust

Batman G/W Study

13-10-88 Fly from Perth to Darwin. Travel to Noonamah.

14-10-88 Travel from Noonamah to Katherine. Meet with Keith Kenny from Billiton. Discuss geology and obtain drill hole location plan etc.

15-10-88 Travel to site to get familiar with Coco City.

16-10-88 Travel to site. Phone for drillers. Wait for drillers to arrive. Travel to site and set up camp.

17-10-88 Set up on first site BW1

BW1.

C 17-10-88 F 17-10-88

Located near RP001 at Robin Prospect
Approx 9900mN 9800mE
9876N 9805E

LITHOLOGY

0 - 18 GREYWACKE grey brown highly weathered fine even grained becoming fresher

18 - 27 GREYWACKE grey slightly weathered fine even grained

27 - 64 GREYWACKE black fresh fine even grained

(EOH)

27/10/88 SWL - 18.37 mbtc
Collar Ht - 0.23m

DRILLING DETAILS

0 - 5.5 - 203mm DHH

5.5 - 64.7 192mm DDH

CASING DETAILS

6mm 155mm ID Class 9 UPVC
Removed and replaced with 1m .8" Steel

64.46mm 50mm Class 9 PVC (36m Slotted)

155mm +0.50 - 5.50m

50mm +0.60 - 63.84m (27.84-63.84m Slotted)

HYDRO DETAILS

Seepage at 29m

DEPTH	TIME(10L)	Q	EC	TEMP	TDS
39	7.6	114	210	35	81
45	4.0	216	210	35	81
58	1.50	576	250	34	100
57	1.50	576	210	35	81
63	1.5	576	250	31	107

BW2

C 18-10-88 F 18-10-88

Located about 50 metres west of PH043 and northernmost section line.
Approx 10500mN 9700mE
10492N 9684E

Lithology

0-6 Greywacke light brown highly weathered fine even grained, well foliated

6-30 Greywacke as above moderately to slightly weathered

30-76.9m Greywacke black grey fresh fine even grained

(EOH)

Drilling Details

0-76.9 7½" Diam

Casing Details

1m 8" Steel Removed

6m of 155mm ID Class 9 UPVC 0.22m

78m of 50mm Class 9 UPVC (36m slotted)
cover 0.27m

Hydro Details

Water Struck at 39m

Depth	Time(10l)	Q	EC	Temp	TDS
45	15	.57			
51	17	50	340	31	148
57	4	216	350	31	153
63	2.40	360	380	38	143
69	2.40	360	350	33	142
76	2.30	376	400	37	152

BW3

C 18-10-88 F 19-10-88

Located near on the edge of the main track

LITHOLOGY

0-3 ALLUVIUM - light brown fine grained silt

3-27 GREYWACKE light brown moderately to slightly weathered fine even grained

27-71m GREYWACKE dark grey black fresh fine even grained

DRILLING DETAILS

0 - 71 m 192mm DHH

Hole back filled

Dry Hole

52 43 40

BW4

C 19-10-88 F 19-10-88

Located on southern boundary of EL 2935 along the old EDITH FALLS dirt road

LITHOLOGY

0-3 ALLUVIUM / WEATHERED HORNFELS light brown fine grained silt and light brown highly to moderately weathered hornfels

3-52.5 HORNFELS dark grey black fresh fine grained

DRILLING DETAILS

0- 52.5 192mm DOTH

Hole abandoned, backfilled

BWS

C 19-10-88 F 20-10-88

Located an approx 10600mN 9000mE.

0-3 ALLUVIUM grey brown silt and clay

3-24 GREYWACKE grey brown to dark grey moderately to slightly weathered

24-30 GREYWACKE (HORNFELSED) grey to dark grey, slightly weathered / fine even grained

30-64.7m GREYWACKE (HORNFELSED) dark grey blue? fresh fine even grained

SWL 18.27m

DRILLING DETAILS

0 - 64.7 7½" DHH.

~~[illegible]~~

CASING DETAILS

6m of 155mm Class 9 UPVC 0.38m

65.39m of 50mm Class 9 UPVC (3mm Slotted) 0.63m

HYDRO DETAILS

First moisture 39m

Depth	Time(100)	Q	EC	TEMP	TDS
45m	10	86.4	—	—	—
57	20	43.2	250	33	100
57	21	181.4	250	32	98.

BWG

C - 20-10-88 F - 20-10-88.
8955N 10365E
10360mE

Located approx 8960mN ~~8960mE~~
at the bottom of a long
clear valley.

0-3m ALLUVIUM grey brown
silt and clay

3-6m GREYWACKE light grey
clay after completely
weathered greywacke

6-21 GREYWACKE dark to dark
grey moderately to
slightly weathered fine
even grained schistose

21-30 GREYWACKE dark grey
as above

30-70.8 GREYWACKE dark grey black
fresh fine even grained
with calcite veining quartz
from 33 to 60 metres.

72
70.78

DRILLING DETAILS

0-2m 7 5/8" DHH

2-7m Finger bit

8 - 70.8m 7 5/8" DHH

CASING DETAILS

6m of 155mm Class 9 UPVC
+0.30m - 5.70m

70.78 m of 50mm Class 9 UPVC
(42m Slotted)

+0.72m - 70.06m (Slotted -
28.06 - 70.06)
COLLAR - 0.48
SWL - 14.60

HYDRO DETAILS

Water first cut at 30m

DEPTH	TIME	Q	EC	TEMP	TDS
33m	20	43.5	700	34	299
39m	8	155.5 / 75	710	37	283
45	8	186	710	34	303
51	5.5 / 11.5	252	1000	32	430
57	5.5 / 85	258	700	30	330.

DEPTH	TIME	Q	EC	TEMP	TDS
63	5.5 8.5	258	710	29	305
70.8	≈ 3 l/sec	~259	710	31	280

BW7

C + F 21-10-88

Located approx 9450mN 11000mE

0 – 12 HORNFELS light brown grey
Moderately weathered fine
even grained

12 - 18 HORNFELS grey brown
slightly weathered with
minor ferruginous stain
on joints.

18 - 36 HORNFELS dark grey - black
fine even grained fresh
massive

DRILLING DETAILS

0-5.8m 4 5/8" DHH

5.8 - 36 6 DHH

Hole Abandoned - Dry - Backfilled

Minor water on W/Fresh
rock contact at 18m

BW8

C 21-10-88 F 22-10-88

Located 9600mN 10250mE
9598N 10253E

0-27m GREYWACKE & light brown
Completely to highly
weathered - clay

27-39 GREYWACKE dark grey
slightly weathered fine
even grained

39-67.7 GREYWACKE dark grey
black fresh fine even
grained with minor
quartz veins.

(EOH)

27/10/88 SWL 24.41 mbtc
Collar Ht 0.55m

DRILLING DETAILS
0 - 5.5m 7 5/8" DAH 192mm
5.5 - 67.7 152mm DAH

CASING DETAILS
6m of 155mm ID Class 9 UPVC
+0.47 - 5.53m
37.7m of 50mm Class 9 UPVC (30m Slotted)

+0.54 - 37.16m (Slotted 7.16 - 37.16m)

HYDRO DETAILS
First water 36m

DEPTH	Time(100)	Q	EC	TEMP	TDS
49m	6	144	570	28	276
56	5	172	600	33	260
61	4.5	192	610	34	258
	1400	216	640	34	272

BW9

C - 22-10-88 F 23-10-88

Located at 10150mN 10200mE
10146N 10200E

0 - 9 Greywacke light brown
Grey fine grained highly
weathered

9 - 27 Greywacke as above
becoming fresher

27 - 64.4 Greywacke black fresh
(EOH) fine even grained

SWL 24.36m

DRILLING DETAILS
0 - 64.7 192mm DTH

CASING DETAILS
6m of 155mm Class 9 UPVC
+0.45 — 5.55 0.35m

65.3m of 50mm Class 9 UPVC (3mm Slotted)
+0.75 - 64.55 (Slotted 34.55-64.55)
0.60m

HYDRO DETAILS

Depth	Time (vol)	Q	EC	Temp	TDS
33	10m	86.4	530	29	249
39	6.3	137	550	30	253
45	5	172	570	32	251
58	5	172	580	32	256
57	4.5	192	580	32	256
64	4.7	192	580	32	256

BW10

C 23-10-88

9163N 9960E

Located at approx 9400mN 9900mE

0-15m CLAY light brown clay
dry after completely
weathered greywacke

15-30 GREYWACKE as above
but highly to slightly
weathered

30-70.8 GREYWACKE
(EOH)

Collar - 0.54
SWL - 23.70

Greywacke grey black fine
even grained fab with
minor limonite staining on
joint surfaces especially
30-40 metres.

DRILLING DETAILS

0 - 70.8 192mm DHH

CASING DETAILS

6m of 155mm Class 9 UPVC
+0.76 - 5.24m

70.38m of 50mm Class 9 UPVC (36m Slot)
+0.88 - 69.

HYDRO DATA

DEPTH	Time(102)	Q	EC	TEMP	TDS
52	23	37	Sample full of foam.		
58	7	123			
64	33	261	230	34	91
70.8	Approx 3	288	200	31	84

BW 1 P PRODUCTION BORE

C 24-10-88 + 25-10-88 :

Located about 10 m North of BW 1 9906N 9806E

Lithology - as for BW1

S.W.L. 27/10/88 - 18.40 m btc
COLLAR HT - 0.10 m

DRILLING DETAILS
0 - 5.5 12¼" Roller bit
5.5 - 62m 254mm DHH.

CASING DETAILS
6m of 272mm OD 260mm ID Steel
+0.25 - 5.75m

62.8m of 155mm Class 9 UPVC (36m Slotted)
+0.2 - 62.6m (Slotted 26 - 62.60)

DEPTH	TIME (10L)	Q	EC	TEMP	TDS
35m	10sec	86.4			
41	7.5	115	200	29	89
47			210	30	91
52	7/5	296	220	29	98
58	6/6	288	220	31	94
62	5/6	316	250	33	102

DEVELOPMENT DETAILS

DEPTH	TIME (10L)	Q
56	2.6	432
50	2.3	376
44	2.8	308
38	3.1	279
32	5.0	172

Running Casing 1 Hour.

Development 1 Hour

BW6R

8984N 10362E

Located about 29 m North of BW6

C 25-10-88 F 26/10/88

DEVELOPING. 1HR

27/10/88 SWL - 18.98 m btc
COLLAR HT - 0.30 m

DRILLING DETAILS
0 - 2.5 m 12¼" Roller bit
2.5 - 70 m 254 mm DHH

CASING DETAILS.
3 m of 272 mm OD 260 mm ID Steel
+0.30 - 2.70 m

m of 155 mm Class 9 UPVC
+0.30 -

42 m SLOTS.

HYDRO DETAILS

DEPTH	(TIME(SEC)	Q	EC	TEMP	TDS
39	3.5	247	670	32°C	299
45	3.3	262	680	32°C	304
51	2.8	308	680	32°C	304
57	3.0	288	680	32°C	304
63	2.3	375	700	33°C	306
70	2.3	375	690	33°C	302

BW 10P PRODUCTION BORE.

LOCALITY - 9162N 9982E

START - 26/10/88 FINISH - 1/11/88

DRILLING DETAILS -
ROLLER: 0-5.5 m 311 mm DIA HOLE
HAMMER: 5.5 - ~~71.0~~ 71.5 M 254mm DIA HOLE
0 - 33m 311 mm DIA HOLE.

CASING DETAILS -
+0.33 - 32.09m 272 mm OD
260 mm ID STEEL SURFACE CASING.
+0.25 - 69.9m 168 mm OD
155 mm ID CLASS 9 UPVC
29.9 - 69.9 m SLOTTED.

HYDRO DATA
DEPTH FLOW EC TEMP TDS
36m CUT WATER
FOAM / QUICKMUD INJECTION
THROUGHOUT DRILLING.
NO FLOW RATES POSSIBLE
AS CAVING OCCURRED ONCE FOAM
WASHED OUT.

LITHOLOGY AS FOR BW 10.
RUNNING CASING.
WASH CASING DOWN.
PROBLEMS WITH FORMATION.
PULL OUT PVC - 28m
REMOVED COLLAR. - 6m
DRILL 0 - 36m 311 mm DIA HOLE
RUN 33m 10" STEEL TO CASE
OFF CLAYS. CEMENT GROUT.
RE-DRILL TO 71.5m.
RUN PVC.

DEVELOPMENT

66 m	10 min	1.5 sec/10L	576 m^3/d
		200 AT 32 °C	83 mg/L
	30 min	2.5 sec/10L	346 m^3/d
60 m	20 min	3.0 sec/10L	288 m^3/d
		200 AT 32 °C	83 mg/L
54m	30 min	4.0 sec/10L	216 m^3/d

COLLAR - 0.15
SWL - 23.4

BW 2P PRODUCTION BORE

LOCATION - 10482N 9689E

START - 2/11/88* FINISH - 5/11/88

DRILLING DETAILS - 0-5.5m 311mm DIA HOLE
5.5-72 m 254mm DIA HOLE

CASING DETAILS - +0.60 - 5.4 m 272mm OD
260mm ID STEEL SURFACE CASING.
+0.60 - 70.9 m 155mm ID
168mm OD uPVC CLASS 9.
34.9 - 70.9 m SLOTTED

HYDRO DATA

DEPTH	FLOW		EC	TEMP	TDS
33 m		MOIST.			
33 m	FOAM	WATER	INJECTION		
48m	6SEC/10L	144 m³/d	350	33°C	147
54m	4SEC/10L	216 m³/d	380	32°C	154
60m	3SEC/10L	288 m³/d	350	33°C	147
66m	2SEC/10L	432 m³/d	350	33°C	147
72m	1.5SEC/10L	576 m³/d	360	33°C	151

LITHOLOGY AS FOR BW 2.

* NB - RIG BRAKE DOWN - REPAIRED IN KATHERINE
ARRIVED BACK ON SITE ON 5/11/88.

DEVELOPMENT

DEPTH	FLOW		
66m	2SEC/10L	432 m³/d	1HR 15min.
60m	2.25SEC/10L	372 m³/d	
54m	2.4SEC/10L	360 m³/d	
48m	3SEC/10L	288 m³/d	

COLLAR - 0.48m

BW8P PRODUCTION BORE.

LOCATION - 9615N 10266E

START - 5/11/88 FINISH - 6/11/88

DRILLING DETAILS - 0-28 m 254mm DIA HOLE
REAMED: 0-28m 311mm DIA HOLE
28-60m 254mm DIA HOLE

CASING DETAILS - +0.8 - 27.84m 272mm OD
260mm ID STEEL. 30m - 1.36 -
+0.8 - 57.8m 168mm OD
155mm ID UPVC CLASS 9.
30 - 27.83 - 57.8m SLOTTED.
- 1.37 - 8c

HYDRO DATA

DEPTH	FLOW	EC	TEMP	TDS.
30m	FOAM/WATER INJECTION			
45m	3SEC/10L 288m³/d			
54m	2.5SEC/10L 345m³/d			
59.9m	1SEC/10L 864m³/d			

LITHOLOGY AS FOR BW 8.

DEVELOPMENT.

DEPTH	FLOW	TIME SPENT.	
54m	0.6SEC/10L	5min	640 AT 37°C
	0.8SEC/10L	20 min	
48m	1SEC/10L	35min	
AAZAA			

BILLITON AUSTRALIA GOLD PTY. LTD.
MT. TODD JOINT VENTURE
Annual Report for Period Ending 28 January, 1989

APPENDIX XI
Metallurgical Testing of Mt. Todd Gold Ores
Beneficiation
- Amdel

amdel
technology and enterprise

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2 November 1988

OD 3/114/0-06780

Billiton Australia
PO Box 872K
MELBOURNE VIC. 3001

Attention: Mr. M. Grier

Dear Mr. Grier,

REPORT NO. 06780/89
GATOR (MT. TODD) GOLD ORES
BENEFICIATION TESTWORK

The report which is enclosed describes the results of recent beneficiation testwork, comprising Part One of the Metallurgical Test Programme on Mount Todd Ores.

Thank you for the opportunity of conducting this testwork. We look forward to being of continuing service in the future.

Yours sincerely,

Peter M. Cameron
General Manager, International Operations Group

JKE:yvu
JKE14/sr06780.doc

amdel
technology and enterprise

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PO Box 872K
MELBOURNE VIC. 3001

Attention: Mr. M. Grier

REPORT NO. 06780/89
METALLURGICAL TESTING OF GATOR
(MOUNT TODD) GOLD ORES
PART ONE - BENEFICIATION TESTWORK

YOUR REFERENCE:		BOD:MFG:SMS
ORDER NOS:		81273, 81274
SAMPLES TESTED:	(1)	Weathered Ore: Composite Nos B1WB1-1D, B1WB1-1P, HL1WB1-1D, HLWB1-1P
	(2)	Transitional Ore: Composite Nos B1TB1-1D, B1TB1-1P, HL1TB1-1D, HL1TB1-1P
	(3)	Primary Ore: Composite Nos B1PB1-1D, B1PB1-1P, HL1PB1-1D, HL1PB1-1P
DATE RECEIVED:		29 July 1988
WORK REQUESTED:		Head Assays, Scrubbing, Gravity Concentration, Flotation, Magnetic Separation Testing, Visual Sorting

Investigation and Report by: R. Phillips, J.K.W. Ellis

General Manager, International Operations Group: Peter M. Cameron

CONTENTS

SUMMARY

Metallurgical Testing of Gator (Mt. Todd) Gold Ores, Part One - Beneficiation

In July 1988 Amdel was invited by Billiton Australia Gold Pty Ltd to submit a proposal to conduct heap leaching and beneficiation testwork on samples of ore from their Gator Prospect, near Katherine, Northern Territory.

The test programme had the following objectives:

(a) Heap (column) leaching: to establish the amenability of the ore by crushed sizes to percolation and cyanide extraction of gold by heap leaching.

(b) Beneficiation: to define the response of the ore to alternate metallurgical processing by laboratory scale testing of scrubbing, gravity concentration, flotation, magnetic separation and agitation cyanide leaching.

The report which follows describes the results of beneficiation testwork, which were summarised as follows:

The samples submitted were drill core and percussion drill chippings from three ore zones; weathered, transition and primary ore.

Drill Core

Drill core samples were composited, crushed to -12.5 mm and portions submitted for multi-element scan analysis, sizing and gold assay, and scrubbing tests.

Elemental Analysis

The ores were of similar composition in terms of their major constituents namely silica, alumina, and potassium minerals, with very minor amounts of copper (below 0.25%). The primary ore contained 0.49% sulphur, compared to less than 0.02% in the weathered and transition ore.

Gold Distribution and Scrubbing Tests

The ores were hard and of dense texture, and it was felt that crushing of drill core did not produce a natural size distribution. Sizing and assay of fractions indicated that although gold grades were higher in the finer fractions, the coarse size distribution of the crushed products meant that gold was mainly distributed in the coarse fractions. Scrubbing gave very little increase in the amount of gold contained in the finer fractions.

Head Assays

Head assays determined from the above tests on drill-core were as follows:

Weathered Ore	0.37 g/t Au
Transition Ore	1.36 g/t Au
Primary Ore	1.04 g/t Au

Percussion Drill Chips

Percussion drill chip samples were submitted for mineralogical examination, gravity concentration, flotation and magnetic separation tests.

Mineralogical Examination

X-Ray diffraction analysis of head samples showed the major minerals in all three ore types to be quartz and muscovite. In the primary ore, the next most dominant iron mineral was chlorite, with small amounts of pyrite and sphalerite. The transistion ore had a larger amount of chlorite, with kaolinite, and goethite. Chlorite was not reported in the weathered ore.

Deslimed samples of drill chippings were tested by heavy liquid separation at S.G. 3.3. The heavy fractions were of higher gold grade than the light fraction, but only contained 10-20% of the total gold in the ore.

Gravity Concentration

Table concentration at 100% minus 1 mm gave poor results with gold recoveries in the range 10-15%, to concentrates grading approximately twice head grade, suggesting that gold or gold bearing minerals were very poorly liberated at 100% - 1 mm, or S.G. differences were insufficient, or that the gold ore was softer than the surrounding rock.

Flotation

Sulphide flotation tests were conducted on ore at 80% minus 75 microns, using four rougher and one cleaner flotation stages. Gold recovery from the weathered and transition ore were poor; however quite good upgrading was achieved for the transition ore. The primary ore was more amenable to flotation, providing gold recovery of 71.2% to a concentrate grading 18.3 g/t Au from a head grade of 1.34 g/t. Further tests would be required to investigate whether finer grinding would improve gold liberation, and to optimise flotation conditions.

Magnetic Separation

Wet high intensity magnetic separation tests were conducted on ore at 80% minus 212 microns. All three ore types produced a magnetic product, but while the tests on weathered and transition ore resulted in gold concentration into the magnetic fraction, magnetic separation of the primary ore resulted in gold concentration into the non magnetic fraction. The primary ore contained a higher proportion of magnetic mineral, (chlorite), than the weathered and transitional ore. Their results may be explained by the association of gold with pyrite/chalcopyrite in the primary ore, and the higher magnetic susceptibility of the oxidised species of these minerals in the weathered and transitional zones. The tests indicated that magnetic separation could be used to achieve a bulk upgrading of the ore, however this would result in the rejection of a significant proportion of the gold in the ore.

Site Water Analysis

Two samples of site water were tested and found to be relatively low in impurity levels, and suitable for use in cyanide leaching.

Visual Sorting

Samples of weathered and primary ore at 100% - 2 mm were hand picked to separate the darker material from the weathered ore, and the veined particles from the primary ore. Fire assays showed the darker weathered ore (high iron oxide) graded 11.3 g/t Au, with the remaining ore 1.42 g/t Au. The veined primary ore, contained pyrite and galena, and had 8.05 g/t Au, versus 0.44 g/t in the remaining ore.

1.0 INTRODUCTION

In July 1988 Amdel Limited (Amdel) were invited by Billiton Australia Gold Pty Ltd (Billiton) to submit a proposal to conduct programmes of heap leaching and beneficiation testwork on samples of ore from their Gator Prospect, near Katherine, Northern Territory. The scope of work for the metallurgical testwork was defined by Billiton in a memorandum dated July 8, 1988, ref. Bod: 8807168.

The Gator ore body comprises three main zones from which corresponding ore types, both drill core and percussion chips, were to be tested, viz:

* Weathered Ore
* Transition Ore
* Primary Ore

The ore is of low gold grade and heap leaching was under consideration. The test programme had primary objectives to define the operating parameters and leaching efficiencies to a level upon which the owners could commit to proceed with the project. Some preliminary test work had indicated that the ore was slow leaching but good recoveries could be obtained at grind sizes of P_{80} = 75 micron. It also indicated that upon crushing gold values report to the finer fractions, and a criterion for sample preparation was that this latter phenomenon did not distort the testwork results. The testwork programme addressed two major process avenues for each of the ore types and had the following objectives:

(a) Beneficiation: to define the response of the ore to metallurgical processing by laboratory scale testing of scrubbing, gravity concentration, flotation and magnetic separation.

(b) Heap (column) leaching: to establish the amenability of the ore by crushed sizes to percolation and cyanide extraction of gold by heap leaching.

The beneficiation testwork is reported in the Part One Report which follows. The column leach testwork is reported separately.

2.0 SAMPLES RECEIVED

Samples of ore were received by Amdel on 31 August 1981.

Samples comprised diamond drill hole intersections and percussion drilling chips as listed below for three (3) ore types: weathered ore, transitional ore, and primary ore.

Ore Type	Testwork Programme	Composite Number	Sample Numbers and Marking
Weathered	Beneficiation	B1WB1-1D B1WB1-1P	BD2 - full core from 10 to 14m BP3 - chips from corresponding metreage (sample numbers 3011 to 3014).
	Column Leach	HL1WB1-1D HL1WB1-1P	BD3 - half core from 3 to 18m BP4 - chips from corresponding metreage (sample numbers 4004 to 4018).
Transitional	Beneficiation	B1TB1-1D	BD3 - full core from 31 to 32m; 38 to 39m; 57 to 58m.
		B1TB1-1P	BP4 - chips from corresponding metreage (sample numbers 4032, 4039, and 4058).
	Column Leach	HL1TB1-1D	BD3 - half core from 33 to 37m; 39 to 43m; 56 to 57m and 58 to 71m).
		HL1TB1-1P	BP4 - drill chips from corresponding metreage (sample numbers 4034 to 4037; 4040 to 4043; 4057; and 4059 to 4071).
Primary	Beneficiation	B1PB1-1D B1PB1-1D	BD2 - full core from 70 to 74m. BP3 - chips from corresponding metreage (sample numbers 3071 to 3074).
	Column Leach	HL1PB1-1D HL1PB1-1P*	BD1 - half core from 62 to 78m. BP2 - chips from corresponding metreage (sample numbers 2063 to 2078).

Approximate weights are shown in Section 3.1.

* Not received.

Note: The following metreages were removed from drill-core samples before further compositing:

BD2	12 - 13m
BD3	57 - 58m
BD2	72 - 74m

3.0 TESTWORK PROCEDURES

3.1 Sample Compositing and Preparation

3.1.1 Drill Core (Beneficiation Tests)

Weathered Ore: Composite No. B1WB1-1D

Approximately 45 kg of full drill core was received in one drum. After removal of retention sample BD2 12-13 m, the remaining drill-core was stage crushed to minus 12.5 mm and one 12 kg sub-sample riffled out.

Transitional Ore: Composite No. B1TB1-1D

Approximately 20 kg of full drill core was received in one drum. After removal of retention sample BD3 57-58 m, the remaining drill core was stage crushed to minus 12.5 mm and one 12 kg sub-sample riffled out.

Primary Ore: Composite No. B1PB1-1D

Approximately 30 kg of full drill core was received in two drums. After removal of retention sample BD2 72 - 74 m (Drum No. 41) the remaining drill-core was stage crushed to minus 12.5 mm and one 12 kg sub-sample riffled out.

3.1.2 Percussion Chips (Beneficiation Tests)

Weathered Ore: Composite No. B1WB1-1P

Approximately 20 kg of dry percussion chips were received in one drum. these were riffled mixed, then 15 x 1 kg charges riffled out.

Transitional Ore: Composite No. B1TB1-1P

Approximately 100 kg of wet percussion chips was received in five drums. The contents of each drum was dried, and riffled into two parts. One part was returned to its original drum, the other part was composited. The composite was riffle mixed, then 15 x 1 kg charges riffled out.

Primary Ore: Composite No. B1PB1-1P

Approximately 30 kg of dry percussion chips were received in one bag. These were riffle mixed then 15 x 1 kg charges riffled out. One 5 kg charge was riffled from the balance for column leach tests (refer Section 3.1.3) because samples for Composite No. HL1PB1-1P were not received.

3.2 Beneficiation Tests - Drill Core

The 12 kg sub-samples of minus 12.5 mm material from each ore type (refer Section 3.1) were tested as follows:

3.2.1 Size-Fraction Assays

A portion of approximately 2 kg was riffled out for screening on 6.70, 3.35, 1.4, 0.5 mm and 250, 150 and 75 micron screens. Each size fraction was submitted for mixer-mill pulverising and duplicate Au fire assay.

3.2.2 Multi-Element Analysis

A portion of approximately 0.5 kg was riffled out for multi-element ICP and XRF scans.

3.2.3 Scrubbing Test

The balance of the 12 kg sub-sample was rotary scrubbed at a solids content of 50% by weight for 30 minutes in a rubber lined cement mixer. The scrubber product was wet screened at 75 microns.

The plus 75 micron fraction was dried, weighed and screened on 6.70, 3.35, 1.40 and 0.50 mm, 250, 150 and 75 micron screens. The size fractions were submitted for mixer-mill pulverising and duplicate Au fire assay.

The minus 75 micron fraction from wet screening was dried, weighed and combined with the minus 75 micron fraction from the dry screening, then submitted for duplicate Au fire assay.

3.3 Beneficiation Tests - Percussion Chips

The fifteen 1 kg charges of percussion chips from each ore type (refer Section 3.1.2) were tested as follows:

3.3.1 Mineralogical Examination

(a) One 1 kg charge was riffled into two parts: one for XRD analysis, the other for mixer-mill pulverising and triplicate Au fire assay (head assay).

(b) One 1 kg charge was deslimed by wet screening at 38 micron.

The minus 38 micron fraction was dried, weighed and submitted for duplicate Au fire assay.

The plus 38 micron fraction was submitted for heavy liquid separation, with XRD analysis, mineralogical examination of polished sections, and Au fire assay of both the float and sink fractions.

3.3.2 Gravity Concentration

(a) One 1 kg charge was screened on 500, 250, 150, 106, 75, 53 and 38 micron screens. Each fraction was submitted for mixer-mill pulverising and duplicate Au fire assay.

(b) Five 1 kg charges were stage crushed to minus 1 mm and tabled to produce three products: concentrate, middlings and tailings, using a laboratory scale Wilfley Table.

Each product was dried, weighed and sub-samples submitted for mixer-mill pulverising and duplicate Au fire assay.

3.3.3 Flotation

Three 1 kg charges were used to establish the grinding time to achieve 80% passing 75 microns.

Two 1 kg charges were milled to 80% passing 75 microns and subjected to 4 stages of rougher flotation using 100 g/t of potassium amyl xanthate (PAX), 100 g/t of Cyanamid Aerofloat 238 and 25 g/t of frother (MIBC). Rougher concentrates were cleaned without further addition of reagents. Samples of cleaner concentrates, cleaner tails and rougher tails were submitted for gold analysis by duplicate fire assay.

3.3.4 Magnetic Separation

Three ~ 2.5 kg portions, one from each ore type, were rolls crushed to minus 1.7 mm then stage ground to 80% passing 212 microns. Approximately 500 g was removed as head sample for screening on 150 and 75 micron screens. The three size fractions were dried and submitted for duplicate Au fire assay.

The balance of the material at P80 212 microns was treated in a Jones Wet High Intensity Magnetic Separator, using two roughing passes at 30 amps (~ 17000 gauss) and one cleaner retreatment pass at 20 amps (~ 11300 gauss). Pass #1 middling and non-magnetics were retreated in Pass #2, leaving final middlings and non magnetic products.

Pass #1 and Pass #2 magnetics were retreated in Pass #3 at 20 amps, to produce a final magnetic concentrate, and a cleaner tailing consisting of the cleaner middlings and non-magnetics combined.

The four products, i.e.

cleaner concentrate
cleaner tailings
pass #2 middlings
pass #2 tailings

were screened on 150 and 75 microns, and the fractions dried and submitted for duplicate Au fire assay.

3.4 Site Water Analysis

Two samples of site water was submitted for standard water analysis.

3.5 Visual Sorting

Portions of weathered and primary ore drill chippings were crushed to -2 mm and visually hand sorted to separate darker material from the weathered ore and veined material from the primary ore. The products were fire assayed.

4. RESULTS AND DISCUSSION

4.1 Drill Core (-12.5 mm) Multi-Element Scan Assays

Multi-element analyses for the three ore types are shown in Table 1. The elemental compositions were similar, with predominant composition of silica (~ 66%), alumina (~ 15%), Fe_2O_3 (~ 8%) and potassium minerals (~ 4%). There were only minor amounts of other elements likely to have a significant effect on processing, with the possible exception of copper which varied from 0.101% (weathered ore), 0.141% (transition ore) to 0.253% (primary ore).

4.2 Drill Core (-12.5 mm). Gold Assays of Size Fractions

Gold distribution by size fractions for the three ore types crushed to -12.5 mm are shown in Tables 2, 3 and 4. The size distributions for all three ores were predominantly coarse, with less than 10% of the crushed ore being below 500 microns. The small amounts of finer material which were present, were of slightly higher grade than the coarser fractions. However, the bulk of the gold was contained in the coarse fractions.

4.3 Drill Core - Scrubbing Test Results

The gold distribution by size fractions of the scrubbed ore products are shown along with the feed sizings and assays, in Tables 2, 3 and 4. Each of the three ore types produced only a small increase in the gold contained in the fine fractions. This is shown in the following summary using 500 microns, as the top cut-off size.

	% of Au in -500 micron Fraction	
	Unscrubbed	Scrubbed
Weathered Ore	18.4	22.7
Transition Ore	11.1	16.5
Primary Ore	16.0	17.1

Calculated Head Grades

The assay of scrubbing test feed and product size fractions provided the following calculated head grades for the Drill Core Composites.

	Calculated Gold Assay g/t		
	Feed	Product	Average
Weathered Ore	0.33	0.40	0.37
Transition Ore	1.43	1.29	1.36
Primary Ore	0.91	1.16	1.04

4.4 Mineralogical Examination

The mineralogical report received from Amdel Geological Services follows:

4.5 Percussion Chippings - Gold Assays of Size Fractions

Gold distributions calculated for three percussion drill chippings composites are shown in Table 5. The head grades calculated from the gold distributions are summarised as follows:

Weathered Ore (Drill Chips)	1.61 g/t
Transition Ore (Drill Chips)	3.52 g/t
Primary Ore (Drill Chips)	1.25 g/t

It is noted that the sizing and assay of these samples was outside the original scope of work. The screen set nominated had a topsize of 500 micron which in fact retained a relatively high proportion of the drill chippings.

4.5 Percussion Chippings - Gravity Concentration

Metallurgical balances showing the result of table concentration testing of the three ore types (from percussion chippings crushed to 100% - 1 mm) are shown in Table 6.

The results showed that none of the three ore types had a satisfactory response to gravity concentration at 100% - 1.0 mm size. In each case, 60 - 65% of the gold remained in the gravity tailing which comprised 70 - 75% by weight of the ore feed. The results suggested either that gold, or the gold bearing dense mineral, was poorly liberated at the crush size of 1 mm topsize, or else that the combined S.G/size of the mineralised material was close to that of the of the gangue material.

4.7 Percussion Chippings - Sulphide Flotation

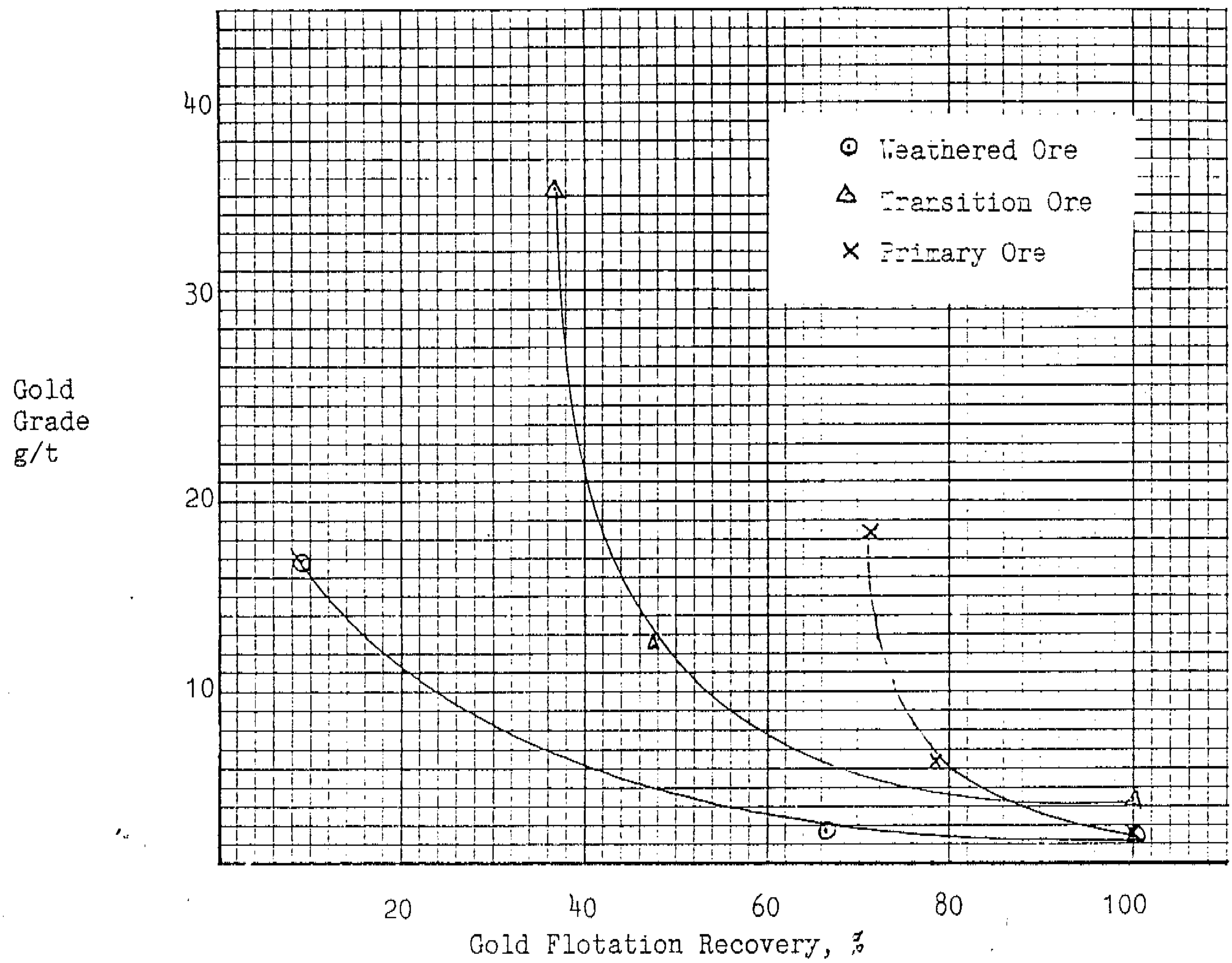
The metallurgical balances for rougher and cleaner flotation tests on the three ore types are shown in Table 7.

Flotation conditions were as outlined above (ref: Sect 3.3.3) and are shown in detail in Appendix I.

Grades and recoveries to final (cleaner conc) concentrate are summarised as follows:

	Au g/t	Recovery %
Weathered Ore	9.25	32.48
Transition Ore	35.50	36.69
Primary Ore	18.30	71.16

Grade-recovery plots for the three ores are shown below:



The tests, which were preliminary only, showed that the flotation response of the primary ore was significantly better than the response of the weathered and transition ores. Further testing of the primary ore would be recommended, to include investigation of the effect of finer grinding, and closer optimisation of flotation conditions with the objectives of improving both grade and recovery of gold to concentrate.

4.8 Percussion Drill Chippings - Magnetic Separation

4.8.1 Head Sample Size Analysis

The size distribution, gold by size fractions, and calculated head assays from sizings of head samples milled to 80% minus 212 microns are shown in Table 8.

4.8.2 Magnetic Separation Test Results

The metallurgical balances for the magnetic separation tests on weathered, transition and primary ore are shown in Tables 9, 10 and 11 respectively. The flowsheet showing the test sequence is shown in Figure 1. Note that the balance for the test on primary ore has been modified from that faxed to the client on 12 October 1988, following confirmation by repeat assaying, and a confirmatory magnetic separation test, that gold in the primary ore was concentrated in the non-magnetics, whereas in the weathered and transition ore, gold tended to concentrate in the magnetic fraction.

The separation results for the three ore types are summarised as follows:

	Weight %	Gold Assay g/t	Dist. %
Weathered Ore			
Magnetic	8.95	5.16	29.3
Non-Magnetic	91.05	1.23	70.7
Head	100.00	1.58	100.00
Transition Ore			
Magnetic	13.28	6.21	24.03
Non-Magnetic	86.72	2.91	75.37
Head	100.00	3.35	100.00
Primary Ore			
Magnetic	35.33	0.65	17.49
Non-Magnetic	64.67	1.67	82.51
Head	100.00	1.31	100.00

The magnetic separation performance data is plotted in Figure 2 which shows gold concentrate grade and recovery, versus weight percent recovered to concentrate.

Weathered Ore

Final products were as follows:

	Weight %	Gold Assay g/t	Dist. %
Cleaner Magnetics	8.95	5.16	29.30
Cleaner Tails (non-mag)	16.96	1.27	13.66
Rougher Middlings	9.50	1.17	7.05
Rougher Tails (non-mag)	64.00	1.22	49.99
Calculated Head	100.00	1.58	100.00

The metallurgical balances for the separate size fractions are shown in Table 9A. The results indicated that the magnetic separation had preferentially rejected fine material. Gold recovery was slightly higher at +75 micron than at -75 micron, and the gold distribution in concentrate was approximately 50% in the -75 micron fraction, compared to 60% in the weathered ore feed.

Transition Ore

Final products were as follows:

	Weight %	Gold Assay g/t	Dist. %
Cleaner Magnetics	13.28	6.21	24.63
Cleaner Tails (non-mag)	16.49	3.21	15.80
Rougher Middlings	10.53	2.85	8.96
Rougher Tails	59.70	2.84	50.61
Calculated Head	100.00	3.35	100.00

The metallurgical balances for the size fractions are shown in Table 10A. As for the weathered ore, the magnetic separation gave slightly preferential rejection of finer material, and better gold recovery from the -75 micron material in the feed. The gold recovery to magnetic concentrate was 24.63%, compared to 29.30% for the weathered ore. However the degree of upgrading, from feed grade to concentrate grade, was not as high as for the weathered ore. In the case of the transition ore, 13.28% of the feed weight was collected as magnetic product, compared to 8.95% of the feed weight of weathered ore.

Primary Ore

Final products were as follows:

	Weight %	Gold Assay g/t	Dist. %
Cleaner Magnetics	35.33	0.65	17.49
Cleaner Tails (non-mag)	21.89	0.81	13.65
Rougher Middlings	8.49	1.05	6.03
Rougher Tails (non-mag)	34.29	2.36	62.03
Calculated Head	100.00	1.31	100.00

The metallurgical balances for the size fractions are shown in Table 11A. As for the weathered and transition ores, the magnetic separation had preferentially rejected fine material. However, overall the ore was more highly magnetic than the weathered and transition ores, 35.33% by weight of the feed ore was collected as magnetic product which was relatively barren of gold compared to the non-magnetic product.

This appears to suggest that in the primary ore, gold is associated with pyrite (non-magnetic), with other magnetic iron minerals present (e.g. chlorite, pyrhottite etc). In the more oxidised zones, pyrite has oxidised to magnetic oxides, and some of the chlorite has weathered to less magnetic forms (e.g. hematite etc).

4.9 Site Water Analysis

Two drums of site water, identified as

5887/1	Drum 1
5887/2	Drum 2

were sampled and submitted for standard water analysis, and arsenic analysis. The results are shown in Tables 12 and 13.

The two waters were similar and were described as of good quality, in relation to possible future use in gold extraction processing. Magnesium content was fairly low, therefore excess lime requirement for neutralisation would not be anticipated. Hardness and total dissolved solids were also low, suggesting the water may not contribute to percolation difficulties. The waters were described as being of superior drinking quality to Adelaide tap water.

4.10 Visual Sorting

Colour differences were not great. The result of hand picking -2 mm ore particles was as follows:

	Wt (g)	Wt %	Au g/t	Au Dist. %
Weathered Ore				
Dark Ore	3.12	4.88	11.3	29.00
Paler Ore	60.83	95.12	1.42	71.00
Head	63.95	100.00	1.90	100.00
Primary Ore				
Veined Ore	9.16	10.77	8.05	68.83
Non-Veined Ore	76.02	89.23	0.44	31.17
Head	85.08	100.00	1.26	100.00

While hand sorting was reasonably effective for the primary ore, it was felt that machine sorting would not be feasible, in view of the small particle size, and indistinct colour separation.

5.0 CONCLUSIONS AND RECOMMENDATIONS

Earlier cyanidation testing (ref Amdel Report BA06600/2, dated 10 June 1988) of oxidised and primary ores (MTM1 and MTM2) had indicated satisfactory gold extraction and reagent consumptions at fairly fine grind sizes, although the primary ore, in particular, was slow leaching. The results of agitation cyanidation were summarised as follows:

Sample		MTM1	MTM2
Ore Type		Oxidised	Primary
Grind % -75 mm		85%	90%
Gold Extraction, %	(6h)	82.7	60.4
	(48h)	90.9	86.0
Reagent Used	(48h) kg/t		
	CaO	0.9	0.8
	NaCN	0.5	1.4
Calculated Head Grade Au		2.75 g/t	1.80 g/t

Subsequently, beneficiation testwork was conducted with the aim of upgrading the ores prior to gold extraction. Conclusions and recommendations from the results were as follows:

Elemental Analysis

The weathered, transition and primary ores were of similar overall composition. Alhtough galena, chalcopyrite and sphalerite had been identified visually in the veins within the primary ore, only low concentrations of base metals, arsenic, sulphur or other elements likely to affect processing e.g. by cyanidation were found. In conjunction with the availability of good qualitity water, the ore composition was evidently suitable for cyanide leaching.

Gold Distribution, Scrubbing of Crushed (12.5 mm) Ore

Crushed drill core was water scrubbed. Analysis of size fractions of scrubber feed and products indicated that the crushed ore contained relatively little fines (below 0.5 mm) and despite the slightly higher grade of the finer fractions, gold was mainly distributed in the coarser fractions.

These results tended to confirm the ores as hard and of dense texture. The small degree of size reduction from drill-core to 12.5 mm may have been insufficient to generate the "natural" percentage of fines which would arise from crushing run-of-mine ore. Therefore particular attention is required to the size-response of the ore to all metallurgical testing including cyanidation testing at laboratory scale.

Mineralogical Examination

The XRD analysis provided information which helped to explain the magnetic separation results. Heavy liquid separation tests were conducted using samples of drill chippings which were in fact significantly coarser than anticipated, and in the light of subsequent knowledge, of the nature of the ore, better results may have been obtained had the percussion chippings been further crushed. A more comprehensive mineralogical examination would be required to determine the heavy liquid separation efficiencies by size fractions. Such a study would be included in a liberation study which it is recommended should be conducted along with any further testing which may be contemplated.

Gravity Concentration

The results of table testing at 100% minus 1 mm were particularly poor, suggesting that:

- gold, or the minerals with which it is associated, were poorly liberated at 100% -1 mm, and/or -
- the gravity difference between the gold bearing ore and the background rock, was small and/or -
- the higher SG 'valuable' ore was slightly softer than the background rock.

The latter point is supported by the general tendency for finer fractions of ore to be of higher grade than the coarser fractions.

Flotation

Sulphide flotation tests on all three ore types gave poor results from the oxidised ores, but quite promising results from the primary ore. Further testing would be required if flotation was seen as the only available route, to include tests at finer grind sizes to improve liberation, and alternate reagent configuration, particularly for the oxidised ore. However, it is likely that once the ore was ground sufficiently finely for flotation, cyanidation would in fact be more effective on all ore types.

Magnetic Separation

All three ores were highly magnetic, however while the gold was associated with the magnetic portion of the weathered and transition ores, it was associated with the non-magnetic portion of the primary ore. Magnetic separation could be used to effect on bulk upgrading of the ore, but would result in the rejection of a significant proportion of the gold in the ore (unless the ore was finely ground). The "opposite" behaviour of oxidised and primary ore types may result in operational difficulties when treating actual mined ore.

Visual Sorting

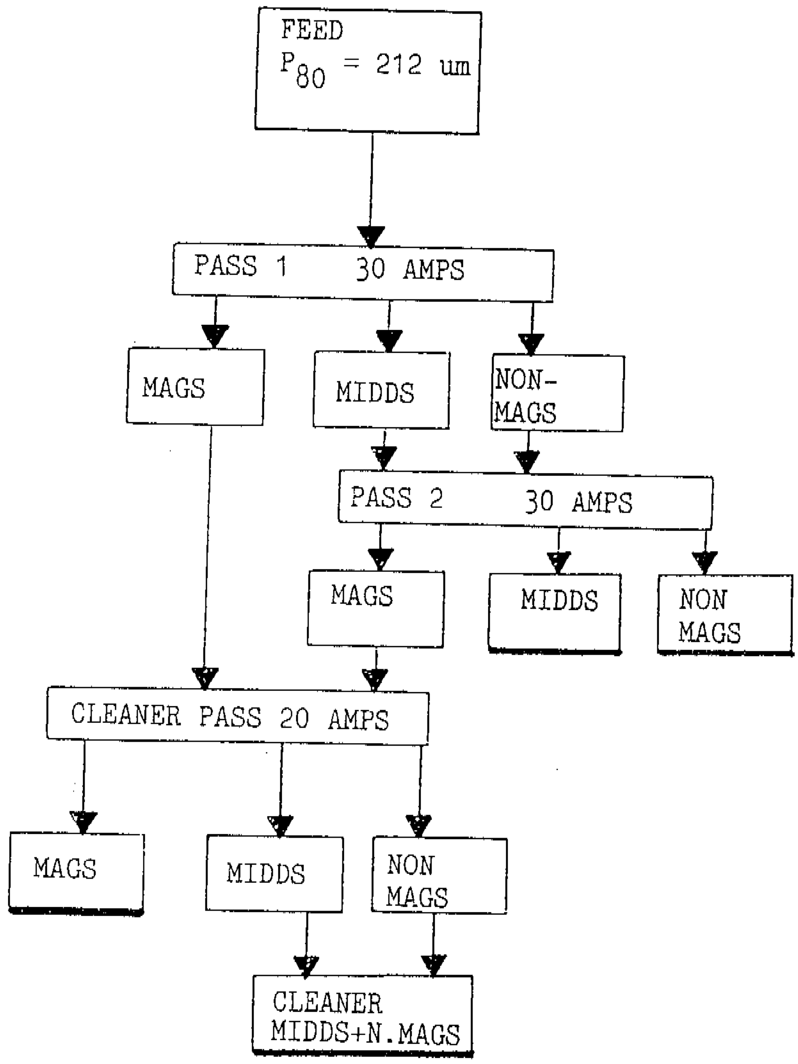
Samples of weathered and primary ore at -2 mm were hand sorted and in the case of the primary ore, quite good gold 'recovery' (68.8%) was obtained by visually separating "veined" or composite particles from the background uniform grey ore. However it was noted that the colour differences were not great, and the particle size (-2 mm + 1 mm) would normally be considered too fine to justify optical sorting of relatively low value ore. This does not rule out the possibility that some property of the composite veinlet particles may exist, which could be sensed and enable sorting to be carried out.

Summary

Although the beneficiation testwork was inconclusive, the findings from several sections of work have indicated the need for a liberation study of the ore, with objectives concerning the liberation (a) of vein material, and (b) of gold. These results will be required in order to fully interpret the results of the foregoing beneficiation tests, and the cyanidation investigations, and provide guidance as to the most appropriate process route.

FIGURE 1

MAGNETIC SEPARATION OF MT. TODD ORES



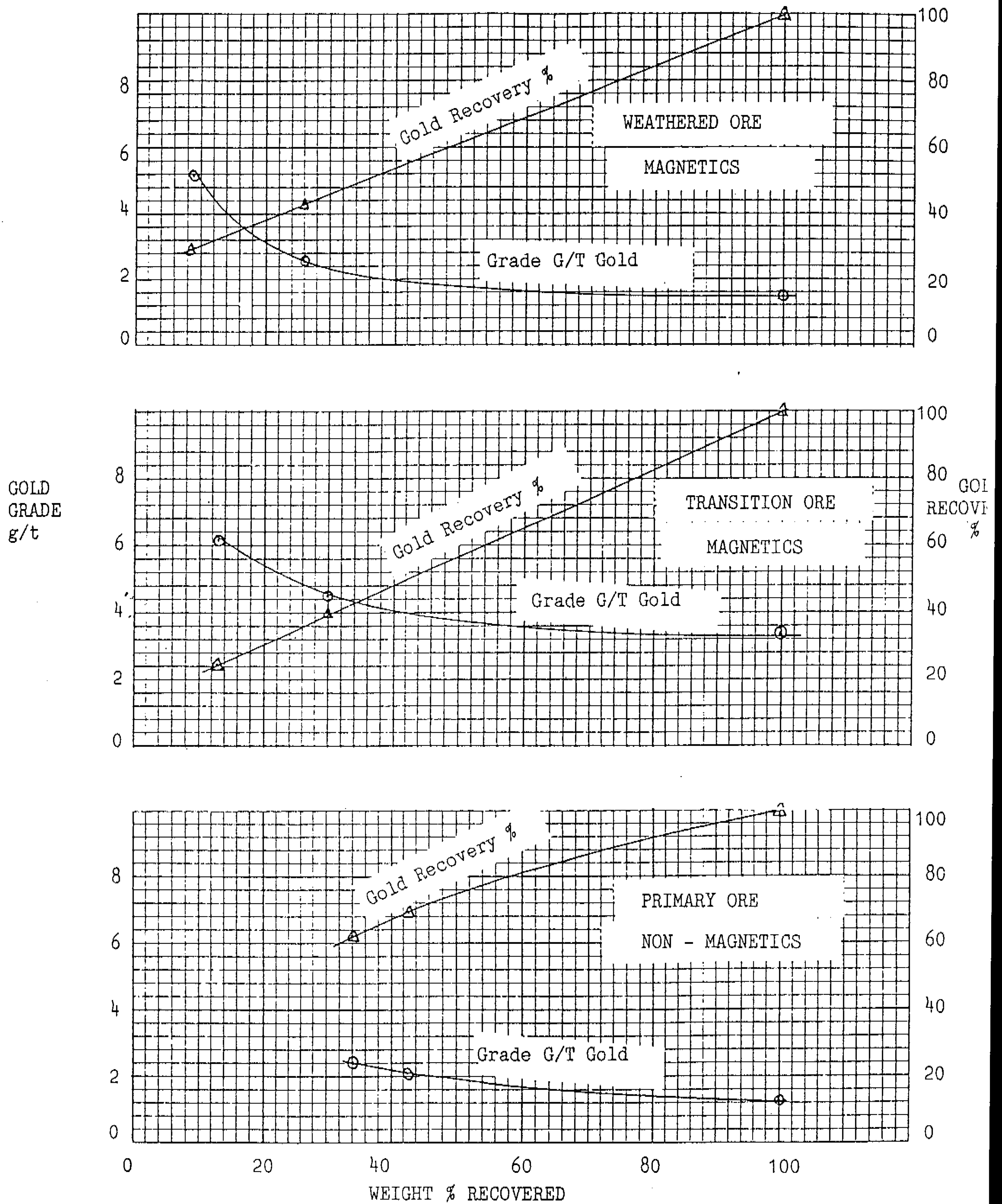


FIGURE 2

GOLD GRADES AND RECOVERIES VS Wt. % RECOVERED

TABLE 1: ICP SCANS OF COMPOSITE ORE SAMPLES, FULL DRILL CORE

Ore Type Composite No.	Weathered B1WB1-1D	Transitional B1TB1-1D	Primary B1PB1-1D
Al_2O_3%	16.4%	15.9%	13.6%
CaO	0.02	0.06	0.12
Fe_2O_3	8.45	7.90	7.90
K_2O	3.62	4.48	4.08
MgO	0.63	0.57	2.28
MnO	0.04	<0.01	0.04
Na_2O	0.20	0.18	0.17
P_2O_5	0.04	0.04	0.06
SiO_2	65.2	66.2	67.3
TiO_2	0.63	0.52	0.72
As	0.015	0.026	0.009
Ba	0.071	0.062	0.053
Cd	<0.002	<0.002	<0.002
Co	0.003	<0.002	0.003
Cr	0.008	0.010	0.015
Cu	0.101	0.141	0.253
La	<0.002	<0.002	<0.002
Mo	<0.002	<0.002	<0.002
Nb	<0.020	<0.020	<0.020
Ni	0.002	0.002	0.004
Pb	0.016	<0.005	0.005
Sn	<0.005	<0.005	<0.005
Sr	0.002	0.004	0.003
Ta	<0.020	<0.020	<0.020
V	0.010	0.007	0.011
W	<0.020	<0.020	<0.020
Y	0.003	0.004	0.002
Zn	0.030	0.028	0.036
Zr	0.016	0.019	0.021
S	0.007	0.017	0.490
Hg (ppm)	<0.01 ppm	<0.01 ppm	<0.01 ppm
Ag	70	3	3
Cs	<10	15	10
Ga	18	12	8
Ge	<4	<4	<4
In	<10	<10	<10
Rb	145	220	170
Tl	10	10	<10
Ce	50	80	70
Se	<2	3	<2
Sb	6	<4	<4
Te	<10	<10	<10
Th	14	30	20
U	<4	<4	<4

TABLE 2 : BENEFICIATION TESTS - FULL DRILL CORE

Ore Type : WEATHERED ORE
Composite No. : 31W31-1D

SCRUBBING TEST : minus 12.5mm feed

06780_1

Size Fraction micron		Product Wt g	Product Wt %	g/t Au	Au Dist'n %
	-6700	1.000	33.7	0.20	20.5
-6700	-3350	953	32.2	0.29	28.4
-3350	-1400	439	14.8	0.45	20.3
-1400	- 500	236	8.0	0.52	12.5
- 500	- 250	91	3.1	0.37	3.4
- 250	- 150	49	1.6	0.42	2.1
- 150	- 75	45	1.5	0.59	2.7
- 75		151	5.1	0.66	10.2
Calc head		2.963	100.0	0.33	100.0
Assay head					

SCRUBBING TEST : scrubber product

Products		Product Wt g	Product Wt %	g/t Au	Au Dist'n %
	-6700	3.608	33.3	0.27	22.4
-6700	-3350	3.271	30.1	0.40	29.7
-3350	-1400	1.597	14.7	0.47	17.2
-1400	- 500	594	5.5	0.59	8.1
- 500	- 250	209	1.9	0.51	2.5
- 250	- 150	99	0.9	0.48	1.1
- 150	- 75	76	0.7	0.73	1.3
- 75		1.397	12.9	0.56	17.8
Calc head		10.852	100.0	0.40	100.0
Assay head					

TABLE 3 : BENEFICIATION TESTS - FULL DRILL CORE

Ore Type : TRANSITIONAL ORE
Composite No. : B1TB1-1D

SCRUBBING TEST : minus 12.5mm feed

06780_2

Size Fraction		Product Wt		g/t	Au Dist'n
micron		g	%	Au	%
	-6700	837	36.2	1.37	34.7
-6700	-3350	755	32.7	1.37	31.3
-3350	-1400	332	14.4	1.52	15.3
-1400	- 500	176	7.6	1.42	7.6
- 500	- 250	68	3.0	1.32	2.7
- 250	- 150	34	1.5	1.52	1.5
- 150	- 75	30	1.3	1.77	1.6
- 75		78	3.4	2.26	5.3
Calc head		2.310	100.0	1.43	100.0
Assay head					

SCRUBBING TEST : scrubber product

Products		Product Wt		g/t	Au Dist'n
		g	%	Au	%
	-6700	3.373	35.4	1.12	30.5
-6700	-3350	2.910	30.5	1.28	30.2
-3350	-1400	1.317	13.8	1.50	16.0
-1400	- 500	526	5.5	1.55	6.6
- 500	- 250	216	2.3	1.62	2.8
- 250	- 150	133	1.4	1.50	1.6
- 150	- 75	110	1.1	1.95	1.7
- 75		943	9.9	1.36	10.4
Calc head		9.528	100.0	1.29	100.0
Assay head					

TABLE 4 : BENEFICIATION TESTS - FULL DRILL CORE

Ore Type : PRIMARY ORE
Composite No. : B1PB1-1D

SCRUBBING TEST : minus 12.5mm feed

o6780_3

Size Fraction micron		Product Wt g	Product Wt %	g/t Au	Au Dist'n %
	-6700	1.031	44.3	0.48	23.3
-6700	-3350	713	30.6	0.94	31.3
-3350	-1400	300	12.9	1.23	17.3
-1400	- 500	144	6.2	1.80	12.2
- 500	- 250	48	2.1	2.05	4.6
- 250	- 150	24	1.0	2.36	2.6
- 150	- 75	23	1.0	2.67	2.9
- 75		48	2.1	2.63	5.9
Calc head		2.330	100.0	0.91	100.0
Assay head					

SCRUBBING TEST : scrubber product

Products		Product Wt g	Product Wt %	g/t Au	Au Dist'n %
	-6700	5,040	47.2	0.92	37.3
-6700	-3350	2.971	27.8	1.00	24.1
-3350	-1400	1.311	12.3	1.37	14.5
-1400	- 500	492	4.6	1.75	7.0
- 500	- 250	169	1.6	1.91	2.6
- 250	- 150	97	0.9	2.03	1.6
- 150	- 75	80	0.7	2.38	1.5
- 75		518	4.8	2.72	11.4
Calc head		10.677	100.0	1.16	100.0
Assay head					

TABLE 5

WEATHERED ORE DRILL CHIPS - GOLD IN SIZINGS

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
+500um	70.68	1.63	71.66
+250um	9.45	1.45	8.52
+150um	5.27	1.43	4.70
+106um	2.52	1.92	3.02
+75um	2.02	1.43	1.80
+53um	1.26	1.50	1.18
+38um	1.17	1.67	1.22
(COARSE FRACTIONS)	(92.37)	(1.60)	(92.10)
-38um	7.63	1.66	7.90
Head(calc.)	100.00	1.61	100.00

TRANSITION ORE DRILL CHIPS - GOLD IN SIZINGS

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
+500um	67.33	3.63	69.44
+250um	9.45	3.40	9.14
+150um	5.66	2.75	4.43
+106um	2.71	2.60	2.00
+75um	2.33	2.75	1.82
+53um	1.63	3.05	1.41
+38um	1.81	3.03	1.56
(COARSE FRACTIONS)	(90.93)	(3.47)	(89.81)
-38um	9.07	3.95	10.19
Head(calc.)	100.00	3.52	100.00

PRIMARY ORE DRILL CHIPS - GOLD IN SIZINGS

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
+500um	91.41	1.13	82.48
+250um	4.67	2.92	10.91
+150um	1.61	2.30	2.96
+106um	0.53	2.10	0.89
+75um	0.34	2.60	0.71
+53um	0.17	2.60	0.36
+38um	0.16	2.35	0.31
(COARSE FRACTIONS)	(98.91)	(1.25)	(98.62)
-38um	1.09	1.58	1.38
Head(calc.)	100.00	1.25	100.00

TABLE 6

TABLE CONCENTRATION RESULTS

PRIMARY ORE DRILL CHIPS -1mm TABLING

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
CONCENTRATE	6.25	2.55	13.41
MIDDLINGS	17.80	1.60	23.97
(R. CONC.)	(24.05)	(1.85)	(37.38)
TAILINGS	75.95	0.98	62.62
Head(calc.)	100.00	1.19	100.00

TRANSITION ORE DRILL CHIPS -1mm TABLING

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
CONCENTRATE	6.83	4.70	10.40
MIDDLINGS	23.03	3.47	25.94
(R. CONC.)	(29.86)	(3.76)	(36.35)
TAILINGS	70.14	2.80	63.65
Head(calc.)	100.00	3.09	100.00

WEATHERED ORE DRILL CHIPS -1mm TABLING

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
CONCENTRATE	9.95	2.60	15.88
MIDDLINGS	17.48	1.82	19.53
(R. CONC.)	(27.43)	(2.10)	(35.41)
TAILINGS	72.57	1.45	64.59
Head(calc.)	100.00	1.63	100.00

TABLE 7

FLOTATION METALLURGICAL BALANCES

FLOTATION OF MT TODD WEATHERED ORE

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
CLEANER CONC	4.98	9.25	32.48
CLEANER TAIL	43.60	1.10	33.81
(ROUGHER CONC)	(48.58)	(1.94)	(66.29)
ROUGHER TAIL	51.42	0.93	33.71
Head(calc.)	100.00	1.42	100.00

FLOTATION OF MT TODD TRANSITION ORE

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
CLEANER CONC	3.44	35.50	36.69
CLEANER TAIL	10.08	3.52	10.64
(ROUGHER CONC)	(13.52)	(11.67)	(47.33)
ROUGHER TAIL	86.48	2.03	52.67
Head(calc.)	100.00	3.33	100.00

FLOTATION OF MT TODD PRIMARY ORE

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
CLEANER CONC	5.23	18.30	71.16
CLEANER TAIL	14.12	0.69	7.25
(ROUGHER CONC)	(19.35)	(5.45)	(78.41)
ROUGHER TAIL	80.65	0.36	21.59
Head(calc.)	100.00	1.34	100.00

File name: MTDF1.REP

TABLE 8

MAGNETIC SEPARATION OF
PERCUSSION CHIPPINGS: HEAD SIZINGS

WEATHERED ORE HEAD SIZING

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
+150uM	23.47	1.35	21.45
+75 uM	17.05	1.47	16.96
-75 uM	59.48	1.53	61.59
Head(calc.)	100.00	1.48	100.00

TRANSITION ORE HEAD SIZING

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
+150uM	14.07	2.60	10.66
+75 uM	26.72	3.10	24.13
-75 uM	59.21	3.78	65.21
Head(calc.)	100.00	3.43	100.00

PRIMARY ORE HEAD SIZING

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
+150uM	20.29	0.78	12.20
+75 uM	24.11	1.31	24.35
-75 uM	55.60	1.48	63.45
Head(calc.)	100.00	1.30	100.00

TABLE 9

MAGNETIC SEPARATION OF MT TODD WEATHERED ORE -212uM

Product		Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON-MAGS	+150	13.58	1.07	9.22
	+75	8.59	0.96	5.23
	-75	42.43	1.32	35.54
(PASS 2 NON-MAGS)		(64.60)	(1.22)	(49.99)
PASS 2 MIDDS	+150	2.34	1.21	1.80
	+75	2.11	1.10	1.47
	-75	5.05	1.18	3.78
(PASS 2 MIDDS)		(9.50)	(1.17)	(7.05)
(PASS 2 MIDDS+NMAG)		(74.09)	(1.21)	(57.04)
CLEANER MAGS	+150	2.44	4.83	7.48
	+75	2.32	4.80	7.06
	-75	4.19	5.55	14.75
(CLEANER MAGS)		(8.95)	(5.16)	(29.30)
CLNR MID+NMAGS	+150	5.18	1.20	3.94
	+75	4.14	1.06	2.79
	-75	7.64	1.43	6.93
(CLEANER MIDDS+NMAG)		(16.96)	(1.27)	(13.66)
(PASS 1+2 MAGS)		(25.91)	(2.61)	(42.96)
Head(calc.)		100.00	1.58	100.00

File name: WMAGS.REP

TABLE 9A: MAGNETIC SEPARATION BY SIZE FRACTIONS

WEATHERED ORE +150 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	57.69	1.07	41.09
PASS 2 MIDDS	9.94	1.21	8.01
(PASS 2 MIDDS+MAGS)	(67.63)	(1.09)	(49.10)
CLEANER MAGS	10.37	4.83	33.33
CLNR MIDD+NMAG	22.01	1.20	17.58
(PASS 1+2 MAGS)	(32.37)	(2.36)	(50.90)
Head(calc.)	100.00	1.50	100.00

WEATHERED ORE +75 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	50.06	0.96	31.61
PASS 2 MIDDS	12.30	1.10	8.90
(PASS 2 MIDDS+MAGS)	(62.35)	(0.99)	(40.50)
CLEANER MAGS	13.52	4.80	42.68
CLNR MIDD+NMAG	24.13	1.06	16.82
(PASS 1+2 MAGS)	(37.65)	(2.40)	(59.50)
Head(calc.)	100.00	1.52	100.00

WEATHERED ORE -75 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	71.44	1.32	58.14
PASS 2 MIDDS	8.54	1.18	6.21
(PASS 2 MIDDS+MAGS)	(79.99)	(1.31)	(64.35)
CLEANER MAGS	7.09	5.55	24.25
CLNR MIDD+NMAG	12.93	1.43	11.39
(PASS 1+2 MAGS)	(20.01)	(2.89)	(35.65)
Head(calc.)	100.00	1.62	100.00

File name: WSZ3.REP

TABLE 10

MAGNETIC SEPARATION OF MT TODD TRANSITION ORE -212uM

Product		Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON-MAGS	+150	7.76	1.96	4.54
	+75	13.48	2.97	11.95
	-75	38.47	2.97	34.11
(PASS 2 NON-MAGS)		(59.70)	(2.84)	(50.61)
PASS 2 MIDDS	+150	1.71	1.79	0.91
	+75	3.21	2.28	2.19
	-75	5.60	3.50	5.86
(PASS 2 MIDDS)		(10.53)	(2.85)	(8.96)
(PASS 2 MIDDS+NMAG)		(70.23)	(2.84)	(59.56)
CLEANER MAGS	+150	2.24	5.20	3.48
	+75	4.80	5.73	8.22
	-75	6.23	6.95	12.93
(CLEANER MAGS)		(13.28)	(6.21)	(24.63)
CLNR MID+NMAGS	+150	2.97	2.47	2.19
	+75	5.83	2.45	4.26
	-75	7.69	4.07	9.35
(CLEANER MIDDS+NMAG)		(16.49)	(3.21)	(15.80)
(PASS 1+2 MAGS)		(29.77)	(4.55)	(40.44)
Head(calc.)		100.00	3.35	100.00

File name: TMAGS.REP

TABLE 10A: MAGNETIC SEPARATION BY SIZE FRACTIONS

TRANSITION ORE +150 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	52.86	1.96	40.83
PASS 2 MIDDS	11.65	1.79	8.22
(PASS 2 MIDDS+MAGS)	(64.51)	(1.93)	(49.04)
CLEANER MAGS	15.26	5.20	31.27
CLNR MIDD+NMAG	20.23	2.47	19.69
(PASS 1+2 MAGS)	(35.49)	(3.64)	(50.96)
Head(calc.)	100.00	2.54	100.00

TRANSITION ORE +75 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	49.34	2.97	44.91
PASS 2 MIDDS	11.75	2.28	8.21
(PASS 2 MIDDS+MAGS)	(61.09)	(2.84)	(53.12)
CLEANER MAGS	17.57	5.73	30.85
CLNR MIDD+NMAG	21.34	2.45	16.02
(PASS 1+2 MAGS)	(38.91)	(3.93)	(46.88)
Head(calc.)	100.00	3.26	100.00

TRANSITION ORE -75 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	66.34	2.97	54.81
PASS 2 MIDDS	9.66	3.50	9.40
(PASS 2 MIDDS+MAGS)	(76.00)	(3.04)	(64.21)
CLEANER MAGS	10.74	6.95	20.77
CLNR MIDD+NMAG	13.26	4.07	15.01
(PASS 1+2 MAGS)	(24.00)	(5.36)	(35.79)
Head(calc.)	100.00	3.59	100.00

File name: TSZ3.REP

TABLE 11

MAGNETIC SEPARATION OF MT TODD PRIMARY ORE -212uM

Product		Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON-MAGS	+150	2.26	3.38	5.85
	+75	3.62	4.73	13.10
	-75	28.41	1.98	43.08
(PASS 2 NON-MAGS)		(34.29)	(2.36)	(62.03)
PASS 2 MIDDS	+150	2.25	0.45	0.78
	+75	2.29	0.78	1.37
	-75	3.95	1.55	4.69
(PASS 2 MIDDS)		(8.49)	(1.05)	(6.83)
(PASS 2 MIDDS+NMAG)		(42.78)	(2.10)	(68.86)
CLEANER MAGS	+150	10.14	0.37	2.87
	+75	11.99	0.59	5.42
	-75	13.20	0.91	9.20
(CLEANER MAGS)		(35.33)	(0.65)	(17.49)
CLNR MID+NMAGS	+150	5.50	0.43	1.81
	+75	6.48	0.78	3.87
	-75	9.90	1.05	7.96
(CLEANER MIDDS+NMAG)		(21.89)	(0.81)	(13.65)
(PASS 1+2 MAGS)		(57.22)	(0.71)	(31.14)
Head(calc.)		100.00	1.31	100.00

File name: PMAGS.REP

TABLE 11A: MAGNETIC SEPARATION BY SIZE FRACTIONS

PRIMARY ORE +150 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	11.22	3.38	51.72
PASS 2 MIDDS	11.17	0.45	6.86
(PASS 2 MIDDS+MAGS)	(22.38)	(1.92)	(58.58)
CLEANER MAGS	50.32	0.37	25.40
CLNR MIDD+NMAG	27.30	0.43	16.01
(PASS 1+2 MAGS)	(77.62)	(0.39)	(41.42)
Head(calc.)	100.00	0.73	100.00

PRIMARY ORE +75 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	14.85	4.73	55.17
PASS 2 MIDDS	9.39	0.78	5.75
(PASS 2 MIDDS+MAGS)	(24.24)	(3.20)	(60.92)
CLEANER MAGS	49.18	0.59	22.79
CLNR MIDD+NMAG	26.58	0.78	16.28
(PASS 1+2 MAGS)	(75.76)	(0.66)	(39.08)
Head(calc.)	100.00	1.27	100.00

PRIMARY ORE -75 UM

Product	Weight %	GOLD Assay g/t	GOLD Distribution %
PASS 2 NON MAG	51.23	1.98	66.35
PASS 2 MIDDS	7.12	1.55	7.22
(PASS 2 MIDDS+MAGS)	(58.35)	(1.93)	(73.57)
CLEANER MAGS	23.80	0.91	14.17
CLNR MIDD+NMAG	17.85	1.05	12.26
(PASS 1+2 MAGS)	(41.65)	(0.97)	(26.43)
Head(calc.)	100.00	1.53	100.00

File name: PSZ3.REP

TABLE 12

Water Analysis Report   Job No. 0676/89   Method W2/1 Page W1

Sample ID. 5887/1 DRUM 1

Chemical Composition

		mg/L	me/L
Cations			
Calcium	(Ca)	7.9	0.394
Magnesium	(Mg)	8.2	0.675
Sodium	(Na)	9.0	0.391
Potassium	(K)	5.1	0.130
Arsenic	(As)	<0.03	
Anions			
Hydroxide	(OH)		
Carbonate	(CO3)		
Bi-Carbonate	(HCO3)	96.6	1.583
Sulphate	(SO4)	5.3	0.110
Chloride	(Cl)	4	0.100
Nitrate	(NO3)	<0.1	

Other Analyses

Reaction - pH	6.3
Conductivity (E.C) (micro -S/cm at 25°C)	165
Resistivity Ohm.M at 25°C	60.606

Derived Data

	mg/L
Total Dissolved Solids	
A. Based on E.C.	87
B. Calculated (HCO3=CO3)	87
Total Hardness	53
Carbonate Hardness	53
Non-Carbonate Hardness	
Total Alkalinity	103
(Each as CaCO3)	

Totals and Balance

Cations (me/L)	1.6	Diff=	0.20
Anions (me/L)	1.8	Sum =	3.38

ION BALANCE (Diff*100/Sum) = 5.97%

Sodium / Total Cation Ratio 24.6%

Remarks

IMBALANCE UNKNOWN ALL RESULTS CHECKED AND VERIFIED

Note: mg/L = Milligrams per litre
me/L = MilliEqivs.per litre

TABLE 13

Water Analysis Report   Job No. 0676/89   Method W2/1 Page W2

Sample ID. 5887/2 DRUM 2

Chemical Composition

		mg/L	me/L
Cations			
Calcium	(Ca)	9.1	0.454
Magnesium	(Mg)	8.7	0.716
Sodium	(Na)	9.5	0.413
Potassium	(K)	5.4	0.138
Arsenic	(As)	<0.03	
Anions			
Hydroxide	(OH)		
Carbonate	(CO3)		
Bi-Carbonate	(HCO3)	105.5	1.730
Sulphate	(SO4)	5.6	0.117
Chloride	(Cl)	3	0.092
Nitrate	(NO3)	<0.1	

Other Analyses

Reaction - pH	6.5
Conductivity (E.C) (micro -S/cm at 25°C)	165
Resistivity Ohm.M at 25°C	60.606

Derived Data

	mg/L
Total Dissolved Solids	
A. Based on E.C.	87
B. Calculated (HCO3=CO3)	94
Total Hardness	59
Carbonate Hardness	59
Non-Carbonate Hardness	
Total Alkalinity	112
(Each as CaCO3)	

Totals and Balance

Cations (me/L)	1.7	Diff=	0.22
Anions (me/L)	1.9	Sum =	3.66

ION BALANCE (Diff*100/Sum) = 5.92%

Sodium / Total Cation Ratio 24.0%

Remarks

IMBALANCE UNKNOWN ALL RESULTS CHECKED AND VERIFIED

Note: mg/L = Milligrams per litre
me/L = MilliEqivs.per litre

APPENDIX 1

FLOTATION TESTING OF MT. TODD ORES

Flotation conditions used in preliminary flotation testing of weathered, transitional, and primary ore types.

Project No.: 06780

Test No.: WB/1

Sample: B1WB1-1P Weathered Ore

Date: 19 SEP 88

Test Object: Rougher and Cleaner Flotation

2 kg charge

(a) Grinding, Flotation Conditions and Reagents

Stage	Conditions				Reagent Addition, g/t															
	Cell Vol. ml	Time (min.)		pH		KAX		AF 238		MIBC					Temp °C		pH		eH, mV	
		Cond.	Flot.												Start	Finish	Start	Finish	Start	Finish
Grind		960 revs																		
Ro Flotation 1	4000	2	2			25		25		50							7.0			
Ro Flotation 2	(1000 rpm)	1	3			25		25		12½										
Ro Flotation 3		1	5			25		25		–										
Ro Flotation 4		1	5			25		25		–								7.6		
Cleaner Flotation	2000	1	8			–		–		–							7.5	7.4		
	(1200 rpm)																			

Project No.: 06780

Test No.: TB11

Sample: BITBI-1P Transitional Ore

Date: 19 SEP 88

Test Object: Rougher and Cleaner Flotation

2 kg charge

(a) Grinding, Flotation Conditions and Reagents

Stage	Conditions				Reagent Addition, g/t														
	Cell Vol. ml	Time (min.) Cond.	Time (min.) Flot.	pH		KAX		AF 238		MIBC				Temp °C Start	Temp °C Finish	pH Start	pH Finish	eH, mV Start	eH, mV Finish
GRIND		1000 revs															5.5		
Ro Flotation 1	4000	2	2			25		25		25						5.5			
Ro Flotation 2		1	3			25		25		—									
Ro Flotation 3		1	5			25		25		—									
Ro Flotation 4		1	5			25		25		—							5.7		
Cleaner Flotation	2000	1	7			—		—		—						5.9			
		1	3			25		—		—	Small amount floated						6.2		

Project No.: 06780

Test No.: PB/11

Sample: 81PB1-1P Primary Ore

Date: 19 SEP 88

Test Object: Rougher and Cleaner Flotation

2 kg charge

(a) Grinding, Flotation Conditions and Reagents

Stage	Conditions				Reagent Addition, g/t															
	Cell Vol. mℓ	Time (min.)		pH		KAX		AF 238		MIBC					Temp °C		pH		eH, mV	
		Cond.	Flot.												Start	Finish	Start	Finish	Start	Finish
Grind		1740 revs																		
																		7.4		
Ro Flotation 1	4000	2	2			25		25		50							7.3			
Ro Flotation 2		1	3			25		25		12½										
Ro Flotation 3		1	5			25		25		–										
Ro Flotation 4		1	5			25		25		–								7.8		
Cleaner Flotation	2000	1	6			–		–		–							7.6			
		1	3			12½		12½		Very small amount floated								7.7		

BILLITON AUSTRALIA GOLD PTY. LTD.
MT. TODD JOINT VENTURE
Annual Report for Period Ending 28 January, 1989

APPENDIX XII
Nedpac Report on Work Indices

METALLURGICAL INDICES
ON
MT TODD ORES
FOR
BILLITON AUSTRALIA GOLD PTY LTD

Prepared by:
NEDPAC ENGINEERING PTY LTD
Job No: 8315
December 1988

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PGP:KA

23rd January, 1989

Billiton Australia Gold Pty. Ltd.,
31st Floor,
570 Bourke Street,
MELBOURNE. VIC. 3000

Attn: Mr. M.F. Grier

Dear Martin,

Metallurgical Indices on Mt Todd Ores

Enclosed are two copies of our Report No. 8315 which formally confirms the results provided earlier by fax. I am pleased to have been able to carry out the testwork for you and trust that the report meets your requirements.

Please do not hesitate to contact me if there are any queries or if we can be of any further assistance.

Yours faithfully

PGPearson

P G PEARSON
Laboratory Manager

Enc.

This report has been prepared for Billiton Australia Gold Pty Ltd by Nedpac Engineering Pty Ltd. Other parties may be given access to the report or receive copies of the report, but only in full, including this page, the title page and appendices.

While Nedpac Pty Ltd has taken all reasonable care to ensure that the facts and opinions expressed in this report are accurate, it does not accept any legal responsibility for any loss or damage suffered resulting from use of this report howsoever caused, and whether by breach of contract, negligence or otherwise.

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1.0 INTRODUCTION

The Nedpac Engineering Pty Ltd Laboratory was commissioned to undertake testwork to determine various metallurgical indices on Mt Todd ore samples. The testwork was to cover the following indices:

- Bond Ball Mill Work Index
- Abrasion Index
- Impact Crushing Work Index
- Unconfined Compressive Strengths

2.0 SAMPLES

A total of twelve (12) samples were received on 21.11.88 at the Nedpac Engineering Pty Ltd Laboratory, for metallurgical indices determinations.

The samples were of fresh ore, transitional ore and weathered ore. Each sample was divided into sub-samples for Bond Ball Mill Index, Abrasion Index, Impact Crushing Work Index and Unconfined Compressive Strength determinations.

3.0 SUMMARY

The results of the testwork are summarised in Table 3.1. The values for Impact Crushing Work Index and Unconfined Compressive Strengths are averages for all the determinations undertaken.

Table 3.1: Summary of Results

	Weathered Ore	Transitional Ore	Fresh Ore
Bond Ball Mill Work Index kWh/t	8.4	20.2	20.4
Abrasion Index	0.018	0.0046	0.066
Impact Crushing Work Index kWh/t	4.1	2.6	7.8
Unconfined Compressive Strengths MPa	24.0	29.1	57.1

4.0 TESTWORK

4.1 Sample Preparation

The samples as received consisted of various sub-samples of drill core approx. 83mm in diameter, of each of the ore types. The sub-samples for Bond Ball Mill Work Index determinations were crushed using a jaw crusher and a cone crusher and then screened at 3.35mm. The oversize was recrushed until the entire sample was minus 3.35mm in size.

The sub-samples for abrasion index were broken by hand until at least 2500 grams of the sample was minus 19.0mm to plus 13.2mm in size.

No sample preparation other than rebagging and relabelling was required on the sub-samples for Impact Crushing Work Index and Unconfined Compressive Strength tests.

4.2 Bond Ball Mill Work Index

The Bond Ball Mill Work Indices for the three samples were determined using the standard Bond procedure, at a limiting screen size of 125 microns.

4.3 Abrasion Index

The Abrasion Indices of the samples were determined according to the standard procedure, developed by Bond, using standardised equipment.

4.4 Impact Crushing Work Index

The Impact Crushing Work Indices were determined according to the standard Bond procedure, using the standard twin pendulum hammers equipment.

4.5 Unconfined Compressive Strengths

The Unconfined Compressive Strengths of the three ore types were determined in accordance with ASTM D2938-71A testing procedure.

5.0 RESULTS

5.1 Bond Ball Mill Work Indices

The Bond Ball Mill Work Indices for the three ores were determined as:

WI = 8.4kWh/t - weathered ore
WI = 20.2kWh/t - transitional ore
WI = 20.4kWh/t - fresh ore

These values were determined at a limiting screen aperture of 125 microns.

The detailed test data sheets are given in Appendix 2.

5.2 Abrasion Indices

The abrasion indices were determined to be as follows:

AI = 0.018 - weathered ore
AI = 0.0046 - transitional ore
AI = 0.066 - fresh ore

The detailed test data sheets are given in Appendix 3.

5.3 Impact Crushing Work Indices

The Impact Crushing Work Indices for the three ores were determined from at least 20 rock specimens for each ore. The results are summarised in Table 5.1 and full details are given in Appendix 3.

Table 5.1: Impact Crushing Work Indices Summary

	Weathered Ore	Transitional Ore	Fresh Ore
No of Tests	20	24	22
Range of Values kWh/t	1.7-12.6	1.7-6.5	4.3-16.1
Average kWh/t	4.1	2.6	7.8

5.4 Unconfined Compressive Strengths

The Unconfined Compressive Strengths were determined on 5 rock specimens of each ore type. The results are summarised in Table 5.2 and detailed test data sheets are given in Appendix 4.

Table 5.4: Unconfined Compressive Strengths Summary

	Weathered Ore	Transitional Ore	Fresh Ore
Range of Values, MPa	10.3-35.2	6.7-60.4	29.0-93.7
Average, MPa	24.0	29.1	57.1

6.0 DISCUSSION

6.1 Bond Ball Mill Work Indices

The Bond Ball Mill Work Indices for the Mt Todd ores ranged from 8.4kWh/t for the weathered ore to 20.4kWh/t for the fresh ore. In comparison, an average ball mill work index for Australasian gold ores is 15.9kWh/t with a range of 10.7kWh/t to 23.2kWh/t. This indicates that the weathered Mt Todd ore has a low power requirement for ball milling. In fact the weathered ore Bond Ball Mill Work Index is almost half the value of the average for Australasian gold ores and is only 80% of the lowest value included in this average.

The transitional ore and fresh ore Bond Ball Mill Work Indices are very similar at 20.2kWh/t and 20.4kWh/t respectively and are towards the higher end of the range of Australasian gold ores. Both the transitional and fresh ores' power requirements in ball milling are approximately 2.5 times higher than that for the weathered ore.

From this it can be seen that the throughput of a ball milling circuit, treating the Mt Todd ore, will depend upon the ratio of weathered ore to transitional or fresh ores.

6.2 Abrasion Indices

The Abrasion Indices for the Mr Todd ores were fairly low with .0046 for the transitional ore through to .066 for the fresh ore. In comparison, typical hard rock Australasian gold ore have an Abrasion Index of approximately 0.2 which is between 3 and 100 times higher than the Mt Todd Abrasion Indices.

6.3 Impact Crushing Work Indices

The Impact Crushing Work Indices relate to the power requirement of an ore during primary crushing. The values for the Mt Todd ore varied from a low of 2.6kWh/t for the transitional ore to a high of 7.6kWh/t for the fresh ore. It is of note that the value for the transitional ore was lower than that for the weathered ore ie. 2.6kWh/t compared to 4.1kWh/t.

6.4 Unconfined Compressive Strengths

The results for the Unconfined Compressive Strengths show considerable variations both within each ore type and between the ore types. As could be expected the highest average unconfined compressive strength was for the fresh ore and the lowest was for the weathered ore. The highest individual test was also for fresh ore but the lowest individual test was for a sample of transitional ore.

APPENDIX 1

SAMPLE DETAILS

SAMPLE DETAILS

Sample Label	Ore Type	Metallurgical Index	Weight (kg)
BD030FA	Fresh	Abrasion	3.20
BD030TA	Transition	Abrasion	3.06
BD030WA	Weathered	Abrasion	3.60
BD030FI	Fresh	Impact Crushing	34.05
BD030TI	Transition	Impact Crushing	39.70
BD030WI	Weathered	Impact Crushing	38.20
BD030FU	Fresh	UCS	21.40
BD030TU	Transition	UCS	25.60
BD030WU	Weathered	UCS	15.05
BD008F	Fresh	Bond Work	17.80
BD008T	Transition	Bond Work	16.20
BD008W	Weathered	Bond Work	17.15

APPENDIX 2

BOND BALL MILL INDICES

BOND WORK INDEX

SAMPLE : 8315 BD 008F

Wt. OF STANDARD VOLUME : 1215.7 g

IDEAL POTENTIAL PRODUCT : 347.3 g

% Undersize in Feed : 8.8 %

Period	Revs of Mill	Wt of New Feed	Total Wt U.S. Ex Mill	Wt of U.S. in Feed	Net Wt U.S. Ex Mill	Net U.S. Produced Per Rev	Revs Next Cycle
1	151	1215.7	206.5	106.6	99.9	0.661	498
2	360	206.5	295.7	18.1	277.6	0.771	417
3	417	295.7	344.8	25.9	318.9	0.765	415
4	415	344.8	381.3	30.2	351.1	0.846	371
5	371	381.3	338.0	33.4	304.6	0.821	387
6	387	338.0	352.2	29.6	322.6	0.834	380

MEAN OF LAST THREE VALUES : 0.833

MESH OF GRIND 125 microns
F_{80}: 2500 microns
P_{80}: 80 microns

BOND BALL MILL WORK INDEX : 20.4 KWhr T^{-1}

BOND WORK INDEX
FEED SIZE DISTRIBUTION

SAMPLE : 8315 BD 008F

SIZE FRACTION (um)	Wt (g)	%Wt	CUM. %Wt
+2800	24.6	9.6	100.0
-2800 + 2000	64.8	25.2	90.4
-2000 + 1400	50.3	19.6	65.2
-1400 + 1000	27.6	10.7	45.6
-1000 + 710	20.3	7.9	34.9
-710 + 500	15.0	5.8	26.9
-500 + 355	10.5	4.1	21.1
-355 + 250	8.7	3.4	17.0
-250 + 180	6.7	2.6	13.6
-180 + 125	5.7	2.2	11.0
-125	22.5	8.8	8.8
TOTAL	256.8	100.0	

BOND WORK INDEX
PRODUCT SIZE DISTRIBUTION

SAMPLE : 8315 BD 008F

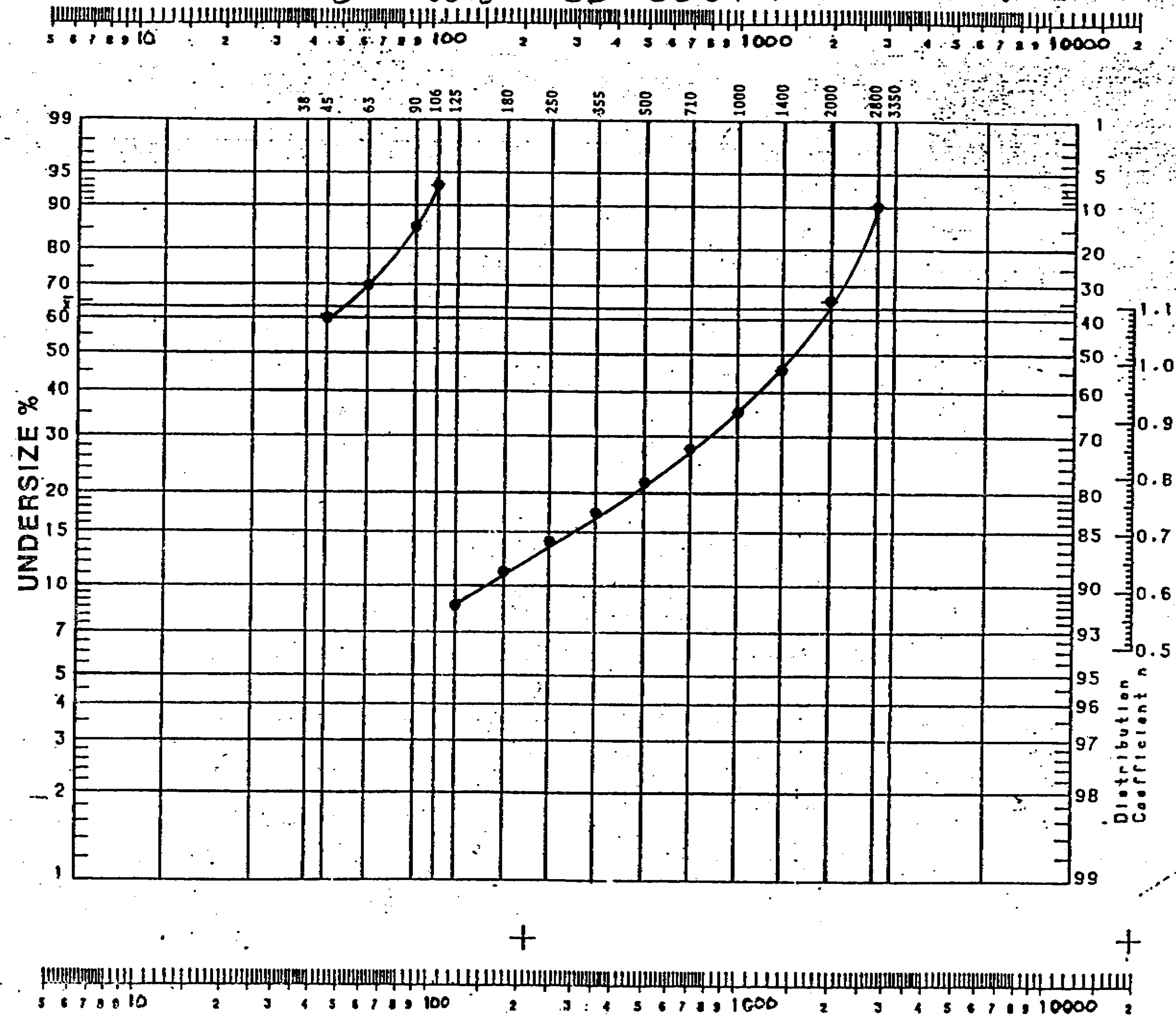
SIZE FRACTION (um)	Wt (g)	%Wt	CUM. %Wt
-125 + 106	2.74	6.85	100.00
-106 + 90	3.17	7.93	93.15
-90 + 63	6.29	15.73	85.23
-63 + 45	4.03	10.08	69.50
-45	23.77	59.43	59.43
TOTAL	40.00	100	

ROSIN RAMMLER PLOT OF SIZE ANALYSIS

JOB 8315 BD 008F.

F80 = 2500μ

P80 = 80μ



BOND WORK INDEX

SAMPLE : 8315 BD 008T

Wt. OF STANDARD VOLUME : 1168.1 g

IDEAL POTENTIAL PRODUCT : 333.7 g

% Undersize in Feed : 9.1 %

Period	Revs of Mill	Wt of New Feed	Total Wt U.S. Ex Mill	Wt of U.S. in Feed	Net Wt U.S. Ex Mill	Net U.S. Produced Per Rev	Revs Next Cycle
1	150	1168.1	218.8	106.3	112.5	0.750	418
2	350	218.8	323.6	19.9	303.7	0.868	351
3	351	323.6	348.6	29.4	319.2	0.909	332
4	332	348.6	329.5	31.7	297.8	0.897	339
5	339	329.5	333.7	30.0	303.7	0.896	339

MEAN OF LAST THREE VALUES : 0.901

MESH OF GRIND 125 microns

F_{80}: 2200 microns

P_{80}: 85 microns

BOND BALL MILL WORK INDEX : 20.2 KWhr T^{-1}

BOND WORK INDEX
FEED SIZE DISTRIBUTION

SAMPLE : 8315 BD 008T

SIZE FRACTION (um)	Wt (g)	%Wt	CUM. %Wt
+2800	7.0	3.5	100.0
-2800 + 2000	48.1	24.1	96.5
-2000 + 1400	43.3	21.7	72.3
-1400 + 1000	22.7	11.4	50.7
-1000 + 710	17.3	8.7	39.3
-710 + 500	13.0	6.5	30.6
-500 + 355	9.7	4.9	24.1
-355 + 250	8.3	4.1	19.2
-250 + 180	6.4	3.2	15.1
-180 + 125	5.5	2.7	11.9
-125	18.2	9.1	9.1
TOTAL	199.5	100.0	

BOND WORK INDEX
PRODUCT SIZE DISTRIBUTION

SAMPLE : 8315 BD 008T

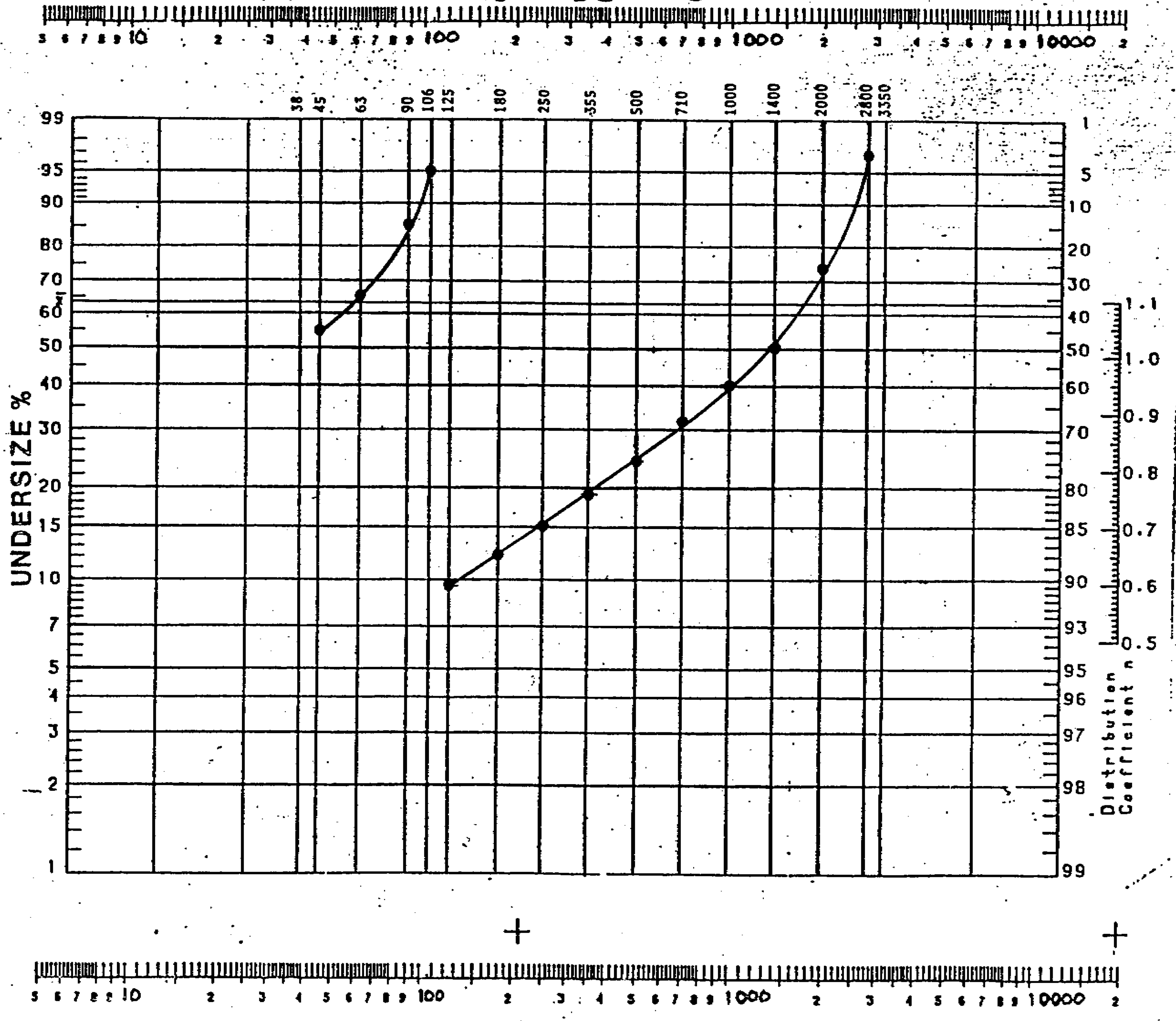
SIZE FRACTION (um)	Wt (g)	%Wt	CUM. %Wt
-125 + 106	2.08	4.99	100.00
-106 + 90	4.22	10.13	95.01
-90 + 63	8.19	19.66	84.87
-63 + 45	4.74	11.38	65.21
-45	22.42	53.83	53.83
TOTAL	41.65	100	

ROSIN RAMMLER PLOT OF SIZE ANALYSIS

FOR JOB 8315 BD 008T.

F_{80} = 2200μ

P_{80} = 85μ



BOND WORK INDEX

SAMPLE : 8315 BD 008W

Wt. OF STANDARD VOLUME : 1128.2 g

IDEAL POTENTIAL PRODUCT : 322.3 g

% Undersize in Feed : 21.3 %

Period	Revs of Mill	Wt of New Feed	Total Wt U.S. Ex Mill	Wt of U.S. in Feed	Net Wt U.S. Ex Mill	Net U.S. Produced Per Rev	Revs Next Cycle
1	100	1128.2	438.7	239.7	199.0	1.990	115
2	115	438.7	388.6	93.2	295.4	2.569	93
3	93	388.6	325.8	82.6	243.2	2.616	97
4	97	325.8	328.5	69.2	259.3	2.673	94
5	94	328.5	317.0	69.8	247.2	2.630	97
		MEAN OF LAST THREE VALUES :				2.640	

MESH OF GRIND 125 microns

F_{80}: 1600 microns

P_{80}: 80 microns

BOND BALL MILL WORK INDEX : 8.4 KWhr T^{-1}

BOND WORK INDEX
FEED SIZE DISTRIBUTION

SAMPLE : 8315 BD 008W

SIZE FRACTION (um)	Wt (g)	%Wt	CUM. %Wt
+2800	5.55	2.21	100.00
-2800 + 2000	28.47	11.34	97.79
-2000 + 1400	35.46	14.13	86.45
-1400 + 1000	24.08	9.59	72.32
-1000 + 710	21.42	8.53	62.73
-710 + 500	19.79	7.88	54.19
-500 + 355	17.12	6.82	46.31
-355 + 250	17.47	6.96	39.49
-250 + 180	14.75	5.88	32.53
-180 + 125	13.57	5.41	26.65
-125	53.33	21.25	21.25
TOTAL	251.01	100.00	

BOND WORK INDEX
PRODUCT SIZE DISTRIBUTION

SAMPLE : 8315 BD 008W

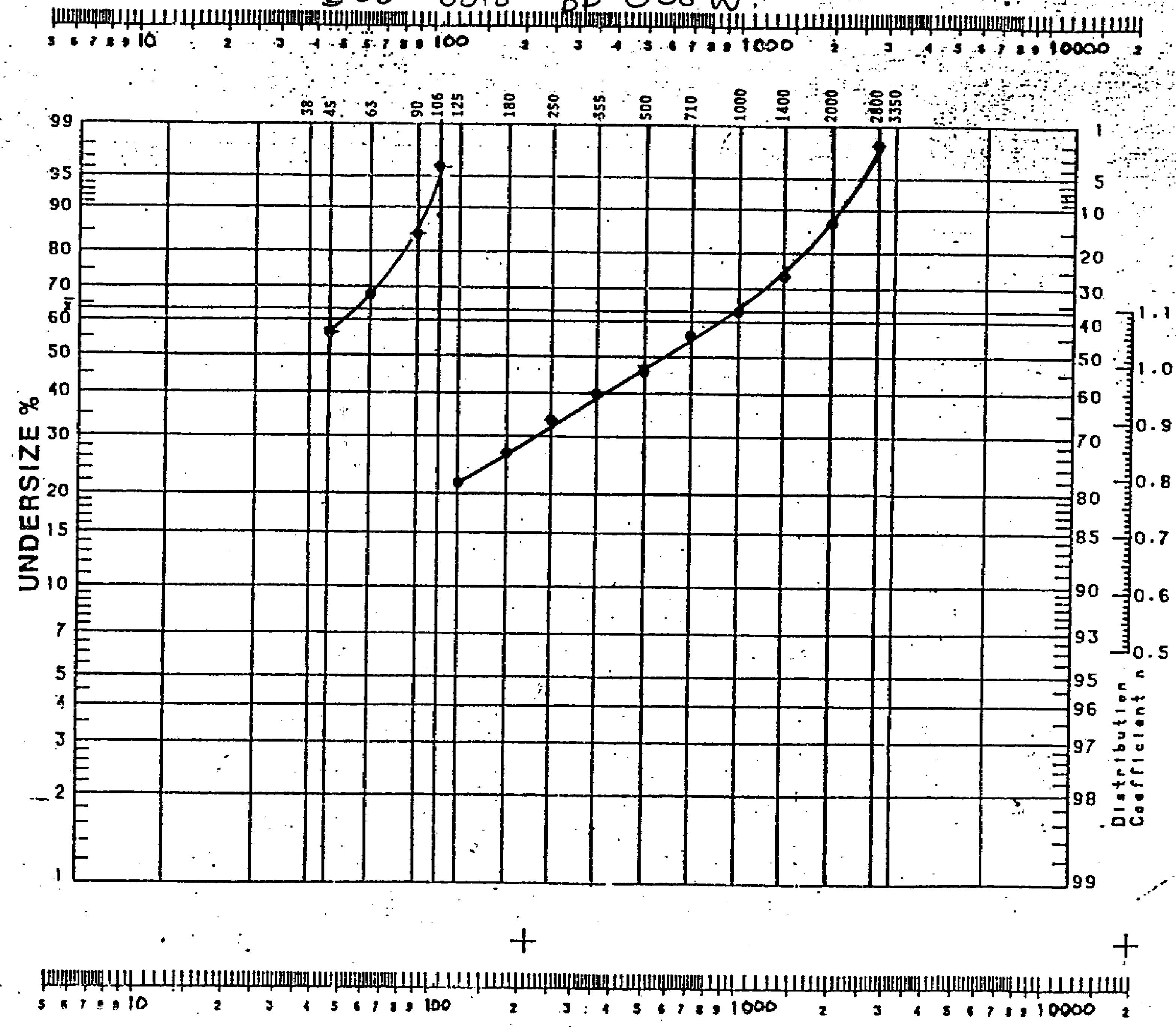
SIZE FRACTION (um)	Wt (g)	%Wt	CUM. %Wt
-125 + 106	1.6	3.6	100.0
-106 + 90	5.7	12.7	96.4
-90 + 63	7.2	16.2	83.7
-63 + 45	5.2	11.7	67.5
-45	24.9	55.8	55.8
TOTAL	44.7	100.00	

ROSIN RAMMLER PLOT OF SIZE ANALYSIS

JOB 8315 BD 008 W.

P80 = 80µ

F80 = 1600µ



APPENDIX 3

ABRASION INDICES AND IMPACT CRUSHING WORK INDICES

AMDEL REPORT NO. 06630(42)/89

NEDPAC PTY LTD

ABRASION INDEX AND IMPACT CRUSHING
WORK INDEX DETERMINATIONS

OD 3/0/0-06630(42)   December 1988

[redacted]

amdel
technology and enterprise

Amdel Limited
(Incorporated in S.A.)
International Operations Group
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Telephone: (08) 43 5733
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Eastwood, S.A. 5063,
Australia

Telex: AA82725
Facsimile: (08) 352 8243

7 December 1988

OD 3/0/0-06630(42)

Nedpac Pty Ltd
20 Bellows Street
WELSHPOOL WA 6016

Attention: Mr. P. Hayward

<u>REPORT NO. 06630(42)/89</u>

YOUR REFERENCE:	Purchase Order No. 0001 Project No. 8315
IDENTIFICATION:	Fresh Ore, Transition Ore, Weathered Ore
DATE RECEIVED:	29 November 1988
WORK REQUIRED:	Abrasion Index and Impact Crushing Work Index Determinations

Investigation and Report by: I.W. McPheat

General Manager, International Operations Group: P.M. Cameron

ABRASION INDEX AND IMPACT CRUSHING WORK INDEX DETERMINATIONS

Three ore samples were submitted for testing to determine Abrasion Index and Impact Crushing Work Indices. The samples had been prepared to the feed sizing requirements for the tests.

The procedures for the Abrasion Index and Impact Crushing Work Index tests are described in Appendices A and B respectively together with Published Indices for various materials. Information giving the relationship between Abrasion Index and metal wear is included in Appendix A.

The following results were obtained.

Sample	Abrasion Index	Impact Crushing Work Index, kWh/tonne
Fresh Ore	0.066	7.8
Transition Ore	0.0046	2.6
Weathered ore	0.018	4.1

The Impact Crushing Work Index data for the tests are given in Tables 1 to 3.

TABLE 1: IMPACT CRUSHING WORK INDEX
Fresh Ore

Specimen No.	Thickness mm	Impact Energy Joule	Work Index kWh/tonne
1	75	30.44	8.0
2	75	48.00	12.6
3	75	41.79	11.0
4	75	25.35	6.6
5	75	61.34	16.1
6	75	25.35	6.6
7	75	35.93	9.4
8	75	35.93	9.4
9	75	20.68	5.4
10	75	20.68	5.4
11	75	16.44	4.3
12	75	25.35	6.6
13	75	41.79	11.0
14	75	25.35	6.6
15	75	35.93	9.4
16	75	20.68	5.4
17	75	16.44	4.3
18	75	25.35	6.6
19	75	20.68	5.4
20	75	16.44	4.3
21	75	16.44	4.3
22	75	48.00	12.6

Average	7.8
Minimum	4.3
Maximum	16.1
Standard Deviation	3.2
95% Confidence Limits	
Lower	6.4
Upper	9.2

Specific Gravity 2.72

TABLE 2: IMPACT CRUSHING WORK INDEX
Transition Ore

Specimen No.	Thickness mm	Impact Energy Joule	Work Index kWh/tonne
1	75	9.34	2.4
2	75	9.34	2.4
3	75	9.34	2.4
4	75	16.44	4.2
5	75	9.34	2.4
6	75	9.34	2.4
7	75	9.34	2.4
8	75	9.34	2.4
9	75	20.68	5.3
10	75	9.34	2.4
11	75	6.51	1.7
12	75	9.34	2.4
13	75	6.51	1.7
14	75	9.34	2.4
15	75	12.65	3.3
16	75	6.51	1.7
17	75	6.51	1.7
18	75	6.51	1.7
19	75	6.51	1.7
20	75	9.34	2.4
21	75	6.51	1.7
22	75	9.34	2.4
23	75	25.35	6.5
24	75	6.51	1.7
Average			2.6
Minimum			1.7
Maximum			6.5
Standard Deviation			1.2
95% Confidence Limits			
Lower			2.1
Upper			3.1

Specific Gravity 2.77

TABLE 3: IMPACT CRUSHING WORK INDEX
Weathered Ore

Specimen No.	Thickness mm	Impact Energy Joule	Work Index kWh/tonne
1	75	12.65	3.3
2	75	9.34	2.4
3	75	9.34	2.4
4	75	12.65	3.3
5	75	9.34	2.4
6	75	12.65	3.3
7	75	30.44	8.0
8	75	20.68	5.4
9	75	25.35	6.6
10	75	9.34	2.4
11	75	12.65	3.3
12	75	12.65	3.3
13	75	12.65	3.3
14	75	20.68	5.4
15	75	48.00	12.6
16	75	12.65	3.3
17	75	12.65	3.3
18	75	6.51	1.7
19	75	12.65	3.3
20	75	9.34	2.4
Average			4.1
Minimum			1.7
Maximum			12.6
Standard Deviation			2.5
95% Confidence Limits			
Lower			2.9
Upper			5.3

Specific Gravity 2.72

APPENDIX A

BOND ABRASION INDEX

This test is used to determine the abrasiveness of a material in relation to metal wear in crushing and grinding.

The test material, in the size range minus 19.0 plus 12.7 mm, is tumbled in a drum and cascades over a hardened steel paddle which rotates concentrically with the drum.

The test material is replaced with a fresh charge after each 15 minutes and the test continues for a total period of one hour.

The weight (g) lost by the paddle for the full test period is the Abrasion Index.

Sample Requirement

Minimum of 3 kg of minus 19.0 plus 12.7 mm material.

Reference

BOND, F.C. (1963) "Metal Wear in Crushing and Grinding", 54th Ann. Mtg of AIME, Inst. Chem. Engrs. Houston, Texas, p.3.

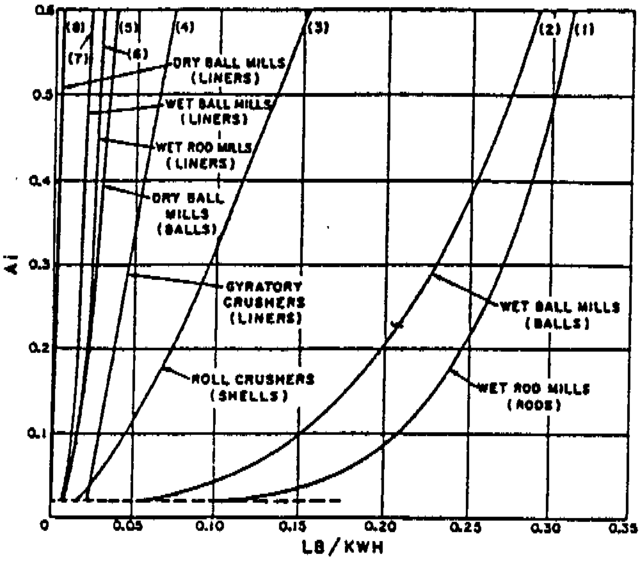

Fig. 1 – Abrasion Index plotted against metal wear in lb/kwn. *

TABLE 1 – ABRASION AVERAGES *

No.	Material	Ave.	Sg	Wi	P	Ai
(1)	Dolomite	5	2.7	–	–	.0160
(2)	Shale	5	2.62	9.9	11,700	.0209
(3)	L.S. for Cement	14	2.7	12.7	12,830	.0238
(4)	Limestone	9	2.7	11.7	–	.0320
(5)	Cement Clinker	8	3.15	13.5	13,070	.0713
(6)	Magnesite	3	3.0	–	–	.0783
(7)	Heavy Sulfides	10	3.56	11.4	12,000	.1284
(8)	Copper Ore	24	2.95	11.7	12,700	.1472
(9)	Hematite	7	4.17	8.5	13,450	.1647
(10)	Magnetite	2	3.7	13.0	–	.2217
(11)	Gravel	4	2.68	15.4	12,950	.2879
(12)	Trap Rock	20	2.80	17.8	14,400	.3640
(13)	Granite	11	2.72	16.6	14,630	.3880
(14)	Taconite	7	3.37	16.3	–	.6237
(15)	Quartzite	3	2.7	17.4	–	.7751
(16)	Alumina	7	3.9	17.5	15,800	.8911

* BOND F.C., (1963) "Metal Wear in Crushing and Grinding", 54th Ann. Mtg., American Inst. Chem. Engrs., Houston, Texas

APPENDIX B

Bond Impact Crushing Test for Work Index Determination

This test is used to determine the Work Index of an ore for use in calculating power requirements for primary crushing.

In this test ore pieces are broken using twin pendulum hammers which simultaneously impact on opposing faces of the rock piece.

The Impact Crushing Work Index is calculated from the energy required to fracture the rock, the thickness of the rock and the specific gravity.

Sample Requirement

The test is carried out on a minimum of 20 rock pieces selected as passing a 76 mm square aperture screen and being retained on a 51 mm square aperture screen. The specimens should not be slabby or acicular in shape.

Test Result

The Work Index (kWh/tonne) determined from this test is applicable to a primary crusher.

Reference

BOND, F.C. (1946) "Crushing Tests by Pressure and Impact", Trans. AIME Vol 169 pp 58–65.

Table 24. Average Impact Work Indices

Material	No. tests	Average	Range
Basalt	15	20.2	9.9–34.8
Bauxite	8	5.3	2.5–12.2
Calcite	4	8.2	5.8–12.2
Cement clinker	3	4.2	1.4– 8.8
Cement raw material	35	11.7	3.6–27.4
Clay	4	4.8	3.7– 6.1
Copper-nickel matte	3	6.3	5.7– 7.2
Copper-nickel ore	3	14.1	10.7–17.4
Copper ore	227	12.4	1.8–40.2
Copper silver ore	4	16.0	13.0–18.8
Coral	3	8.6	7.9– 9.5
Diorite	11	20.1	13.3–27.3
Dolomite	24	12.8	5.4–31.4
Ferrochrome alloy	13	9.5	1.9–24.5
Ferromanganese	6	4.8	3.2– 9.0
Ferrosilicon	6	7.1	3.3–11.4
Fullers earth	3	1.3	0.1– 3.3
Gabbro	7	18.6	16.7–21.2
Gneiss	7	15.9	8.0–23.7
Gold ore	15	17.5	3.7–34.2
Granite	63	15.7	6.7–38.0
Gravel	11	16.7	6.9–26.8
Gypsum rock	6	6.9	4.3–11.7
Ilmenite	3	12.7	10.7–16.4
Iron ore, unidentified	77	10.0	2.3–33.6
Hematite	64	9.6	2.0–29.4
Magnetite	44	10.1	2.4–19.2
Taconite	30	14.9	9.3–27.3
Lead ore	4	15.5	11.0–21.8
Lead-zinc ore	11	9.3	5.5–14.3
Limestone	178	11.1	3.3–27.6
Manganese ore	3	5.3	0.4– 8.9
Molybdenum ore	24	12.5	5.8–18.6
Nickel ore	8	10.1	2.1–19.0
Oil shale	7	15.8	11.5–20.2
Phosphate rock	7	3.3	0.5–11.7
Quartz	11	12.8	6.8–22.1
Quartzite	17	12.9	5.2–19.1
Sandstone	7	13.1	6.5–28.6
Schist	6	12.5	4.1–23.5
Shale	7	10.6	5.8–19.0
Silica rock	6	9.4	4.2–15.9
Slag	10	12.8	1.3–21.9
Stone	8	16.9	10.4–27.5
Tin ore	3	18.0	16.6–19.5
Trap rock	95	19.0	4.9–55.5
Zinc-lead ore	4	10.5	4.5–16.3
Total	1115		

From SME Mineral Processing Handbook (1985), Edited by N.L. Weiss, Society of Mining Engineers, N.Y.

APPENDIX 4

UNCONFINED COMPRESSIVE STRENGTHS

SRC Laboratories (W.A.) Pty. Ltd.
A Subsidiary of Sunmark Corporation Ltd.

Correspondence: P.O. 184, Doubleview W.A. 6018.
34 Walters Drive, Herdsman Business Park
Osborne Park W.A. 6017
Phone: (09) 244 1199. Telex: AA197099
Facsimile: 244 1457

SRC

Ref: S3632/JO:ma

29th November, 1988

NEDPAC PTY LTD
20 Bellows Street
WELSHPOOL WA 6106

ATTENTION: Mr P Hayward

Dear Sir

RE: SUBMITTED ROCK CORES

Attached are the following documents of report for work required by you on the above project :

15 PLATES — Unconfined Compressive Strength of Intact Roack Core Specimens Summary

If we can assist further, please advise.

Yours faithfully
JOHN OLIVER

Technical Manager
for SRC LABORATORIES (WA) PTY LTD

Enc

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 1 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 1

Core Number: 33054

PHYCICAL DESCRIPTION:

Name of Rock: Fresh Ore

SPECIMEN DATA:

Height:	147.8	mm
Diameter:	83.0	mm
Mass:	1338	g
Density:	-	t/m^3
Height/Diameter Ratio:	1.78 *	

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 93.7 MPa

REMARKS: Non Standard Core estimated surface area , half the diameter.
* NOTE: Non-standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD | **SHEET No.:** 2 **OF:** 15

PROJECT: SUBMITTED ROCK CORES | **JOB No.:** S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 2

Core Number: 3055

PHYSICAL DESCRIPTION:

Name of Rock: Fresh Ore

SPECIMEN DATA:

Height:	177.0	mm
Diameter:	83.0	mm
Mass:	261.7	g
Density:	2.730	t/m^3
Height/Diameter Ratio:	2.13	

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 29.0 MPa

REMARKS:

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 3 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 3

Core Number: 33056

PHYCICAL DESCRIPTION:

Name of Rock: Fresh Ore

SPECIMEN DATA:

Height:	174.0	mm
Diameter:	83.0	mm
Mass:	2599	g
Density:	2.760	t/m^3
Height/Diameter Ratio:	2.10	

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 29.2 MPa

REMARKS: Sheared down fracture plane

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 4 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 4

Core Number: 33057

PHYCICAL DESCRIPTION:

Name of Rock: Fresh Ore

SPECIMEN DATA:

Height:	179.0	mm
Diameter:	83.0	mm
Mass:	2674	g
Density:	2760	t/m^3
Height/Diameter Ratio:	2.16	

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 87.7 MPa

REMARKS: Shattered

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 5 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 5

Core Number: 33058

PHYSCICAL DESCRIPTION:

Name of Rock: Fresh Ore

SPECIMEN DATA:

Height:	178.5	mm
Diameter:	83.0	mm
Mass:	2666	g
Density:	2.760	t/m^3
Height/Diameter Ratio:	2.15	

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 45.8 MPa

REMARKS:

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 6 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 6

Core Number: 33059

PHYCICAL DESCRIPTION:

Name of Rock: Transition Ore

SPECIMEN DATA:

Height:	176.5	mm
Diameter:	83.0	mm
Mass:	2527	g
Density:	2.650	t/m^3
Height/Diameter Ratio:	2.13	

RATE OF STRAIN: 80. kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 27.7 MPa

REMARKS: Sheared down fracture planes

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 7 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 7

Core Number: 33060

PHYCICAL DESCRIPTION:

Name of Rock: Transition Ore

SPECIMEN DATA:

Height:	169.0	mm
Diameter:	83.0	mm
Mass:	2404	g
Density:	2.710	t/m^3
Height/Diameter Ratio:	2.00	

RATE OF STRAIN: 80. kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 18.7 MPa

REMARKS: Non standard core poor specimen
Failed down fracture planes

TESTED BY: AG **CHECKED BY:** AG **DATE:** 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 8 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 8

Core Number: 33061

PHYCICAL DESCRIPTION:

Name of Rock: Transition Ore

SPECIMEN DATA:

Height:	153.5	mm
Diameter:	83.0	mm
Mass:	1757	g
Density:	-	t/m^3
Height/Diameter Ratio:	1.85	*

RATE OF STRAIN: 20 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 6.7 MPa

REMARKS: Non standard core not cylindrical

* NOTE: Non standard L/D Ratio.

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 9 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 9

Core Number: 33062

PHYCICAL DESCRIPTION:

Name of Rock: Transition Ore

SPECIMEN DATA:

Height:	118.6	mm
Diameter:	83.0	mm
Mass:	1760	g
Density:	2.740	t/m^3
Height/Diameter Ratio:	1.43	*

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 60.4 MPa

REMARKS:

* NOTE: Non standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 10 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 10

Core Number: 33063

PHYCICAL DESCRIPTION:

Name of Rock: Transition Ore

SPECIMEN DATA:

Height:	127.0	mm
Diameter:	83.0	mm
Mass:	1899	g
Density:	2.760	t/m^3
Height/Diameter Ratio:	1.53	

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 31.9 MPa

REMARKS:

* NOTE: Non standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 11 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 11

Core Number: 33064

PHYSICAL DESCRIPTION:

Name of Rock: Weathered Ore

SPECIMEN DATA:

Height:	146.2	mm
Diameter:	82.8	mm
Mass:	1976	g
Density:	2.500	t/m^3
Height/Diameter Ratio:	1.76	*

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 28.4 MPa

REMARKS: Failed down fracture planes

* NOTE: Non standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

*Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 12 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 12

Core Number: 33065

PHYSICAL DESCRIPTION:

Name of Rock: Weathered Ore

SPECIMEN DATA:

Height:	154.5	mm
Diameter:	82.8	mm
Mass:	1801	g
Density:	-	t/m^3
Height/Diameter Ratio:	1.87	*

RATE OF STRAIN: 50 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 10.3 MPa

REMARKS: Non Standard core not cylindrical

* NOTE: Non standard L/D Ratio

TESTED BY: A G CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

SRC

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 13 **OF:** 15

JOB No.: s3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 13

Core Number: 33066

PHYSICAL DESCRIPTION:

Name of Rock: Weathered Ore

SPEÇIMEN DATA:

Height:	148.8	mm
Diameter:	83.0	mm
Mass:	2067	g
Density:	2.570	t/m^3
Height/Diameter Ratio:	1.8	*

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 35.2 MPa

REMARKS: Failed down fracture planes

* NOTE: Non standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 14 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 14

Core Number: 33067

PHYSICAL DESCRIPTION:

Name of Rock: Weathered Ore

SPECIMEN DATA:

Height:	152.4	mm
Diameter:	82.8	mm
Mass:	1990	g
Density:	2.410	t/m^3
Height/Diameter Ratio:	1.86	*

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 24.8 MPa

REMARKS:

* NOTE: Non standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

* Denotes use of Rock Colour Chart

CLIENT: NEDPAC PTY LTD

PROJECT: SUBMITTED ROCK CORES

SHEET No.: 15 **OF:** 15

JOB No.: S3632

DATE TESTED: 28-11-88

UNCONFINED COMPRESSIVE STRENGTH OF INTACT ROCK CORE SPECIMENS SUMMARY

Tested in accordance with ASTM D2938 - 71A.

SAMPLE IDENTIFICATION: 15

Core Number: 33068

PHYCICAL DESCRIPTION:

Name of Rock: Weathered Ore

SPECIMEN DATA:

Height:	100.0	mm
Diameter:	82.8	mm
Mass:	1303	g
Density:	2.410	t/m^3
Height/Diameter Ratio:	1.21	*

RATE OF STRAIN: 100 kN/minute

UNCONFINED COMPRESSIVE STRENGTH: 21.2 MPa

REMARKS: Failed dwon fracture planes

* NOTE: Non standard L/D Ratio

TESTED BY: AG CHECKED BY: AG DATE: 29-11-88

*Denotes use of Rock Colour Chart

BILLITON AUSTRALIA GOLD PTY. LTD.
MT. TODD JOINT VENTURE
Annual Report for Period Ending 28 January, 1989

APPENDIX XIII
Mt. Todd Gold Project & Prefeasibility Study
for a Carbon in Pulp Process
- Minproc

BILLITON AUSTRALIA

MOUNT TODD GOLD PROJECT
NORTHERN TERRITORY

PRE-FEASIBILITY STUDY FOR
A CARBON IN PULP PROCESS

JANUARY 1989

Minproc Engineers Pty. Ltd.

67 ST. PAUL'S TERRACE, SPRING HILL
QUEENSLAND 4000 AUSTRALIA
TELEX 44114 FAX: (07) 832 0101
TELEPHONE: (07) 839 0383

MT. TODD GOLD PROJECT

A PRE-FEASIBILITY STUDY

OF THE

CARBON IN PULP PROCESS

FOR

BILLITON AUSTRALIA GOLD PTY LTD

Prepared by:
Minproc Engineers Pty., Ltd.

REV	DATE	DESCRIPTION OF REVISION	BY	APPROVED	CLIENT

67 ST. PAUL'S TERRACE
P.O. BOX 544
SPRING HILL QLD. 4004
TELEPHONE: (07) 839 0383
FACSIMILE: (07) 832 0101
TELEX: 44114

MINPROC ENGINEERS PTY. LTD.

MTTODLET.11D

February 7, 1989

Mr. Martin Grier,
Billiton Australia Gold Pty. Ltd.,
Marland House,
570 Bourke Street,
MELBOURNE VIC 3000

Dear Sir,

The final prefeasibility study report on the Carbon in Pulp process route for the Mt. Todd Gold Project is enclosed.

This study defined the C.I.P. process as the more viable option for the purposes of a feasibility study.

We await your instructions with regard to the recommendations deriving from the prefeasibility study, and trust that this report meets your requirements.

Yours faithfully,
MINPROC ENGINEERS PTY. LTD.

M.J. Gunn
STUDY MANAGER

Encl.

CONTENTS

LIST OF DRAWINGS

LIST OF FIGURES

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EXECUTIVE SUMMARY

1. The C.I.P./C.I.L. process route is the recommended option for a Feasibility Study of the Mt. Todd Gold Project.

2. A significant amount of work is required prior to the Feasibility Study, as follows:

    1. Completion of the Mining Feasibility Study.
    2. Further leaching and adsorption testwork, plus refinement of grinding parameters.
    3. Cost comparison of alternate grinding circuit configurations.
    4. Further geotechnical investigations.
    5. Examination of preconcentration techniques, and their economic impact.

3. Capital costs were composed of ($ millions):

Comminution	$ 15.73	Infrastructure	$ 6.707
C.I.P.	$ 8.29	Mine Development	$ 2.223
Gold Recovery	$ 1.50	Owners Capital	$ 2.337
Services	$ 1.36	Working Capital	$ 1.063
Dams	$ 2.53		
Contingency	$ 4.15		
Management	$ 4.63		
TOTAL PROCESS	$ 38.19	TOTAL	$ 50.52

4. Operating costs were broken down as follows ($/tonne ore)

Mining	$ 3.99	(average over mine life)
Power	$ 1.29	
Manning	$ 1.29	
Consumables	$ 4.26	
Others	$ 0.23	
TOTAL	$ 12.68/tonne of ore	

5. The Nett Present Value of the project was calculated to be ($ millions):

    A. 2 million tonnes per annum

10 year mine life	$ 9.169
15 year mine life	$ 15.967

    B. 3 million tonnes per annum

7 year mine life	$ 3.831
10 year mine life	$ 20.409

6. The project is particularly sensitive to head grade. At this point in time, resources would be best employed in seeking a means by which head grade can be increased.

1.0 INTRODUCTION

The Mt. Todd Gold Project is currently a group of adjacent exploration leases located approximately 50 kilometers north of Katherine in the Northern Territory (Figure 1.). The leases are either held by the Joint Venture partners, Billiton Australia Gold Pty Ltd and Zapopan Consolidated Pty Ltd, or under option from Pacific Gold N.L.

The leases are accessed from either the Stuart Highway along 12 kilometres of gravel road, or via the Edith Falls road. The option to develop a new road into site from a point further east - after the Edith Falls road crosses the Edith River - has also been studied.

This report, prepared for Billiton Australia Gold Pty Ltd, forms the second part of a pre-feasibility study of the Mt. Todd Gold Project.

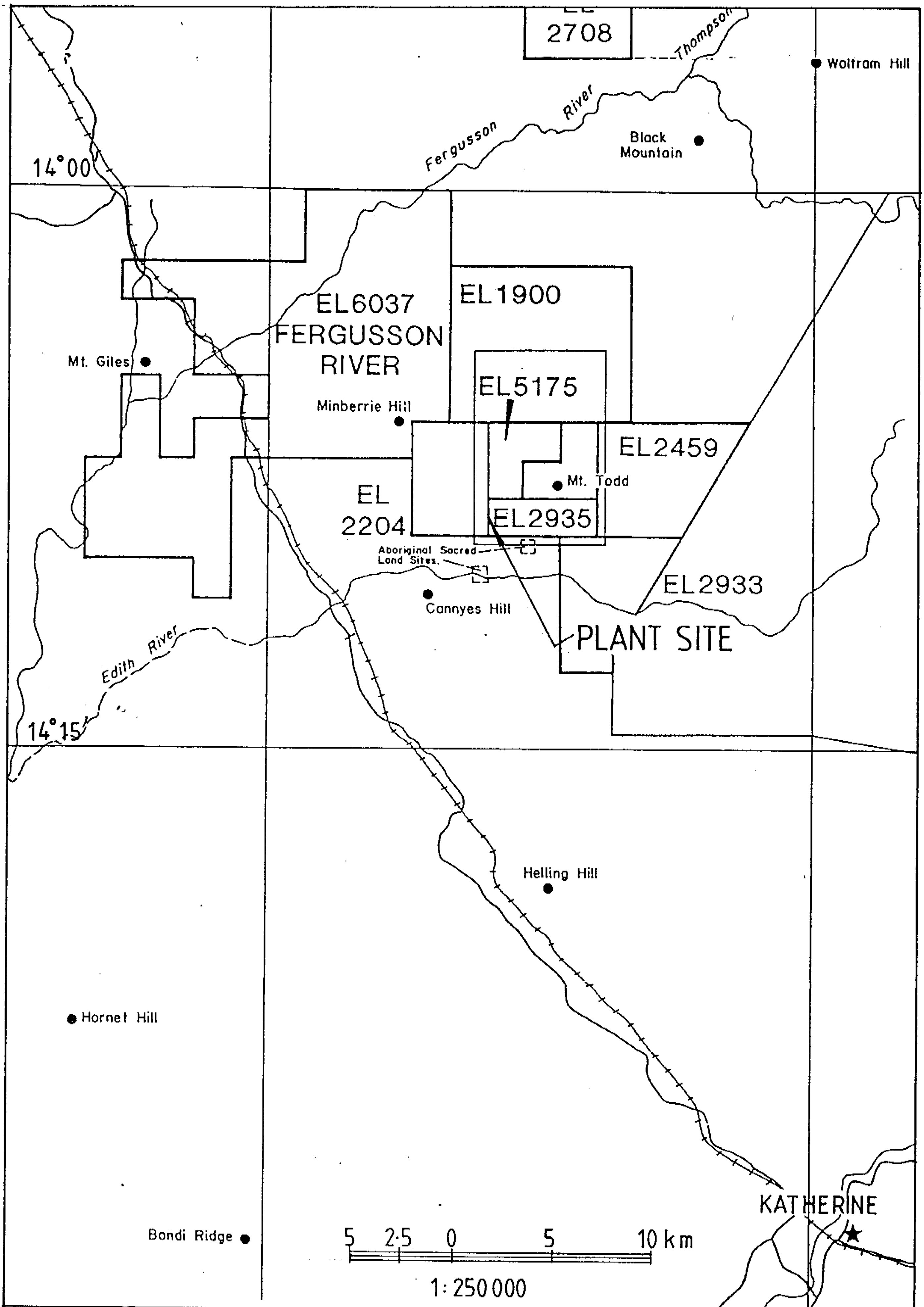
The first report focussed on the heap leaching process route and preliminary costing of infrastructure.

The heap leaching alternative required extremely fine crushing and recovery response was mediocre. The process was seriously considered because the orebody is low-grade, and sufficient open space for leach pads was available. Comminution, pre-treatment, ore transport and pad construction were expensive in terms of capital and operating costs relative to the majority of heap leach operations, where particle size is an order of magnitude larger and pads are located close to the process plant.

The C.I.P. process for Mt. Todd ore is straight forward, with the exception of high work index values for fine grinding. Recovery is relatively high, the ore appears to be free of contaminating species, both organic and inorganic, and slurry rheology is not a process problem.

The comminution stage of the process has been structured to minimise capital cost. In the time frame of the pre-feasibility studies, it has not been possible to design an all-purpose circuit which would service any permutation of heap and C.I.P. processing. H.P. rolls crushing and ore sorting were not incorporated, but are discussed in the report, as are the various circuit options (Section 9) requiring further investigation.

The mining pre-feasibility study has been reviewed and costs adjusted to fit with the overall infrastructure costs derived for the previous study. Contract mining budget quotations have been obtained and are contained in Appendix 2. Geology and mining studies were carried out in Melbourne and were included in the final report on heap leaching.



MT. TODD GOLD PROJECT

FIG. 1

2.0 SCOPE OF STUDY

2.1 GENERAL

This pre-feasibility study constitutes an examination of the technical and financial viability of an open pit mine and C.I.P. process for the Mt. Todd Gold Project. The scope of work is essentially as specified overall by Billiton Australia Gold Pty Ltd.

2.2 SCOPE OF WORK

Metallurgical Testwork

- Review of Metallurgical Testwork carried out to date and recommendations for any further testwork.

Process Description

- General Description;
- Preparation of preliminary flowsheets and design criteria;
- Indication of anticipated plant performance.

Plant Design

- Preliminary selection of all items of equipment and plant including capital cost and a review of process options.

- Operating costs;
- Power requirements;
- Water requirements;
- Manning requirements.

Water Management

- Water management requirements and tailings disposal.
- Evaluate water recovery/discharge systems.

Financial Evaluation

- Capital Costs;
- Operating Costs;
- Project Economics;
- Sensitivity Analysis. Annual production of ore is planned to be 2 million tonnes, however, sensitivity analysis of costs will be carried out for 3 million tonnes per annum production of ore.

Project Programme

- Engineering;
- Construction;
- Commissioning;
- Identification of long lead items.

Recommendations

- Comparative Analysis of heap leaching and C.I.P. studies;
- Recommendation on approach for a feasibility study;
- Recommendations for site investigations to obtain necessary geotechnical information.

3.0 INFRASTRUCTURE

3.1 SITE ACCESS AND ROADS

The Mt. Todd area is situated approximately 50 kilometers north of Katherine and 10 kilometers east of the Stuart Highway. The Edith Falls road is bitumen sealed and provides the most direct route from Katherine to the project via the existing gravel road connection described on Figure 3.1 as Option 1.

Development of Option 1 would require upgrading of the existing Edith River crossing with a reinforced concrete box culvert structure, which preliminary hydrological studies indicate would suffer inundation on an average of six days per year. A bridge would be required to achieve a higher level structure. This aspect is recommended for review in the final feasibility study. Under Option 1, the existing gravel road would require upgrading to rural road standard and is 4.5 kilometers in length from the Edith River road to the lease boundary.

Option 2 for site access is shown on Figure 3.1 and further east along the Edith Falls road from Option 1. It has the advantage of utilisation of an existing bridge crossing of the Edith River, but does require a culvert structure at the Stow Creek crossing. New road is needed for 2.5 kilometers and upgrading of existing road for 1.5 kilometers to the lease boundary.

Both Options 1 and 2 require provision of culverts at minor creek crossings. Cost estimates for these options are set out in Section 6.0.

A third means of road access is via a poor gravel road which proceeds in a northwesterly direction from the mine, and is 12 kilometers in length and would require some 12 culvert crossings and significant re-alignment to achieve safe road geometry. The road joins the Stuart Highway 4 kilometers north of the Edith Falls turnoff. Investigation of this alternative access was discontinued on instructions from the Client.

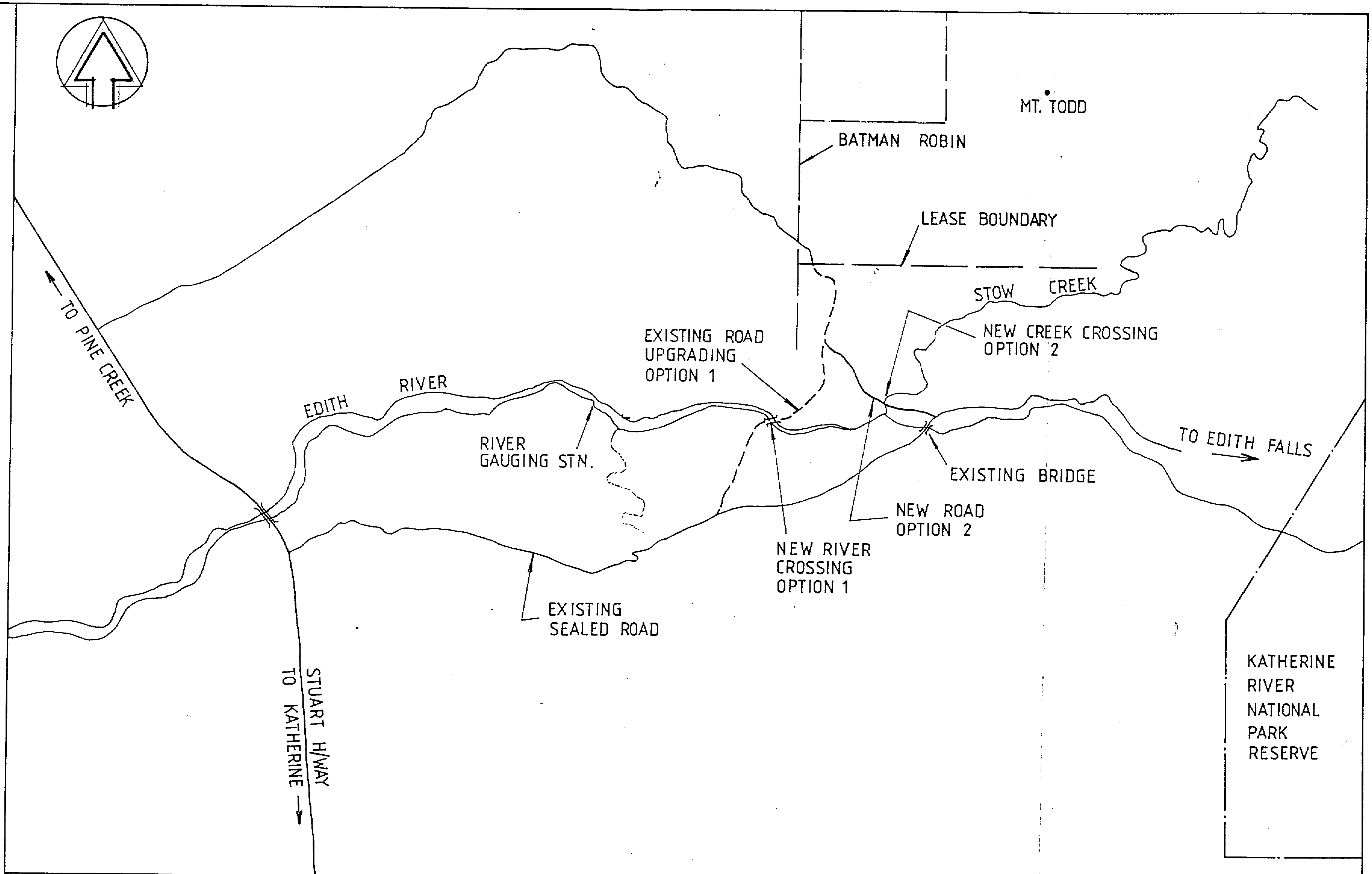
3.2 ACCOMMODATION

In view of the likely recruitment of single persons from interstate and shortage of accommodation in Katherine, the approach was adopted for the project to provide single accommodation for process plant and administration personnel either on site or in Katherine. The cost of these alternatives is reviewed in Section 6.

It has been assumed that the mining contractor would provide accommodation for mining personnel. Housing in Katherine is assumed to be provided for five senior staff:

. Mine Manager
. Mill Superintendent
. Mine Superintendent
. Administration Officer
. Geologist

Single accommodation requirements have been estimated at 40 rooms which may reduce if partial local recruitment is achieved or if some personnel make their own arrangements for accommodation.



MT. TODD GOLD PROJECT
SITE ACCESS

FIG.3·1

3.3 POWER SUPPLY

A review and costing of power generation options was carried out for the heap leaching pre-feasibility study.

After both the Client and Minproc had conducted preliminary discussions with the Power and Water Authority, it was understood that power would be available from the grid at a rate which would:

a) Compete with any other source;
b) Reflect the level of initial capital investment in transmission and substation made by the Client.

The C.I.P. study has proceeded on the assumption that capital investment in this area will be made, and that power will cost 10 cents per kW/hr.

The grid option is a more secure supply, requires negligible maintenance, and is sized and costed on the basis of total installed power of 12 Megawatts.

The capital cost increase relative to the heap leach option is primarily for extra transformers and upgrading of the Oil Cooled Breaker unit. Costs of capital contributions are:

Transmission line	-	$1.375 million
66 - 11 kV substation	-	$450,000
Transformers	-	$400,000
TOTAL		$2.225 million

Without this order of capital contribution it may be difficult to obtain power within a reasonable time frame. At best, a 6 month lead time is imposed on certain items of equipment (according to PAWA).

3.4 COMMUNICATIONS

A.1 Multichannel Radio

This links to the Telecom microwave system. For a system dimensioned for 10 exchange lines and 30 analogue extensions (and a 5 year term).

A.	Up front capital	$251,000
	10 line exchange installation	$14,250
	10 line exchange rental	$2,384
	TOTAL	$267,634
B.	Lease (5 years)	$138,075
	5 years rental	$223,590
	10 line exchange installation	$14,250
	10 line exchange rental	$2,384
	TOTAL	$378,299

A.2 Iterra is almost as expensive to install and running costs are far greater as maximum STD rates are charged on all calls. An up front payment on a permanent station plus five years rent, maintenance plus installation is $322,616 for 10 lines.

B. Terminal Equipment

Budget costs for a 9600 model PABX and a 9100 CMS (Communication management system) is $36,860.

C. Cabling

Facility cabling for 30 extensions plus lightning protection (level 2) - $7,200.

Total System Cost:

. Multichannel radio	$268,000
. Terminal Equipment	$37,000
. Cabling	$7,200
TOTAL	$312,000

Discussions with the independent group Consultel reveal that numerous options are available, and significant economies may be achieved. The options depend on the area, facilities in place and the attitude Telecom take in the region.

It is recommended that Consultel be commissioned to prepare a proposal on this basis.

3.5 SITE BUILDING

The following buildings have been included in the scope of site facilities for process plant and administration personnel. It is assumed that the mining contractor will provide office, amenities and maintenance facilities.

. Main Office	15m x 12m
. Male Ablutions	12m x 3m
. Female Ablutions	6m x 3m
. Crib Room	9m x 3m
. First Aid	3m x 3m
. Laboratory	12m x 3m
plus sample preparation area	12m x 9m
. Crib Room - Plant area including toilet	9m x 3m
. Workshop/Warehouse	24m x 12m
. Storage Compound	24m x 24m
. Plant Office	6m x 3m

Provision has been made for office equipment, basic workshop equipment and warehouse shelving. A forklift has been included for materials handling.

3.6 WATER SUPPLY

3.6.1 The water requirement for the 2 MTPA case is 12.6 million m^3 P.A. The current concept of water supply can deliver water at the appropriate rate to the process plant for the dry season, and it is anticipated that a wet season top up will cover the remainder.

3.6.2 Bores

Five production bores have been proven by Rockwater, yielding on average 400,000 cubic metres per annum. Pumps and pipelines to deliver this water to the raw water tank have been selected and costed.

Water quality from this source is excellent, but for optimum desorption performance, and domestic use, a water softener has been included.

Table 3.2 shows the parameters for each bore. Tables 3.3 - 3.7 show the major ion balance and a calculated Langelier Index. All indices infer that scale formation is negligible and the water is high quality in inorganic terms.

3.6.3 Raw Water Dam

Provision for a large raw water dam has been made to meet the water requirements of the operation for 2,000,000 tonnes per annum. The dam wall is located approximately 500 metres to the west of the projected pit (i.e. beyond the pit location considered in this study). This was due to concerns relating to potential leakage via faulting into the pit. A spillway directs overflow into an existing watercourse.

The dam catchment area of 500 hectares has been assessed as adequate to fill the dam each wet season. Based on rainfall records for Katherine 1873 - 1987, and assuming an average runoff coefficient of 0.70 for the generally steep rocky terrain, the catchment yield has been assessed as 1.85 million cubic metres at 95% confidence (i.e. expected in 19 out of 20 years). A dam capacity of 800,000 cubic metres has been taken as appropriate to provide an annual usage of 600,000 cubic metres, after evaporation losses from the dam surface. It should be noted that reliable yield from the dam in the first year of operation can only be obtained after the initial wet season fill, and this may have some bearing on project timing. A risk factor may be associated with late wet season conditions depending on the scheduled start-up time.

Preliminary geotechnical investigations by Golder Associates indicate that the dam floor and foundation consists of strong tightly jointed rock at shallow depths, so that minimal seepage losses and good foundation conditions for the dam wall are anticipated. Clayey materials which would be used for dam core construction were identified at the proposed Stow Creek crossing, and closer sources may be located by further investigation.

There is potential for expansion. As discussed in Section 5.3, the annual requirement for the 2 million tonnes per annum case will be met by the existing borefield and the dam. If the 50% increase in raw water requirement associated with the 3 million tonnes per annum case is anticipated, the dam should be redesigned and constructed for the higher capacity rather than attempt an upgrade later. The location is shown on the site plan. Submersible pumps are used to transfer water to the plant site.

3.6.4 Tailing Water Reclaim

For design purposes, only 15% reclaim has been used. Evaporation losses will be significant over the expanse of the tailings facility as it is currently conceived. This figure may be improved upon in practice, and the flow will be significant in the wet season. Catchment is minimised, and it is anticipated that the flow will be of manageable proportions - either used in process, or stored in the dam over a short period.

TABLE 3.2

Summary of Production Bore Data

Bore Number	Location	Depth Drilled (m)	Recommended Pumping Rate (cu m/d)	Pump Setting (m)	Internal Diameter of Pump Housing (mm)	Salinity (mg/l TDS)
BW 1P	9906N 9806E	62	250	45	155	102
BW 2P	10482N 9689E	72	300 or 350	55	155	151
BW 6P	8984N 10362E	70	300 or 350	50	155	302
BW 8P	9615N 254 10266E	60	300 or 250	50	155	
BW 10P	9162N 9982E	70	250	65	155	84
		SUM	1400			

TABLES 3.3 - 3.7 FOLLOW

Langelier Index Calculation

DISC: 8802
H2OQUALT.WR1

Program is effective for temperatures
between 5 and 93 degrees C.

PROGRAM CALCULATES LANGELIER INDEX
JOB NUMBER MT TODD
SAMPLE NUMBER 8WP1
DATE 30/10/88

ION	IONIC WT	VALENCE	EQUIVLNT WT	p.p.m.	e.p.m.	EQUIVLNT CaCO3
CATION						
Aluminium	27.0	3	9.00		0.00	
Ammonium	18.0	1	18.00		0.00	
Calcium	40.1	2	20.05	8	0.40	20
Copper	63.5	2	31.75	0.002	0.00	
Hydrogen	1.0	1	1.00		0.00	
Ferrous Ion	55.8	2	27.90	4	0.14	
Ferric Ion	55.8	3	18.60		0.00	
Magnesium	23.4	2	11.70	9	0.77	38
Manganese	54.9	2	27.45	0.11	0.00	
Potassium	39.1	1	39.10	3	0.08	
Sodium	23.0	1	23.00	28	1.22	61
Silver	107.9	1	107.90		0.00	
Gold	197.0	1	197.00		0.00	
Zinc	65.4	2	32.70	0.07	0.00	
TOTAL CATION				52.1	2.61 (epm +)	
ANION						
Bicarbonate	61.0	1	61.00	125	2.05	102
Carbonate	60.0	2	30.00		0.00	0
Chloride	35.5	1	35.50	85	2.39	
Flouride	19.0	1	19.00		0.00	
Iodide	126.9	1	126.90		0.00	
Hydroxide	17.0	1	17.00		0.00	
Nitrate	62.0	1	62.00	0.1	0.00	
Phosphate (tri)	95.0	3	31.67		0.00	
Phosphate (di)	96.0	2	48.00		0.00	
Phosphate (mon)	97.0	1	97.00		0.00	
Sulphate	96.1	2	48.05	5	0.10	
Bisulphate	97.1	1	97.10		0.00	
Sulphite	80.1	2	40.05		0.00	
Bisulphite	81.1	1	81.10		0.00	
Sulphide	32.1	2	16.05		0.00	
Cyanide	26.0	1	26.00		0.00	
TOTAL ANION				215.10	4.55 (epm -)	

DATA INPUT (values marked * must be entered,
other values are optional)

pH *		6.65	Measured (actual)
Temp *		22	Celcius
Suspended solids	HERE		p.p.m.
Dissolved O2	HERE		p.p.m.
Dissolved CO2	HERE		p.p.m.
Measured TDS	HERE		p.p.m.
Conductivity	HERE		u mho/cm
Resistivity	HERE		ohm/m
Langelier "C"	HERE		Off nomogram.

CALCULATIONS

Total Hardness	58.41	mg/l as CaCO3
Total Alkalinity	102.46	mg/l as CaCO3
Cation/Anion balance	-27.09	%
Sum of ions (TDS)	267	p.p.m.
Calced pCa	3.70	
Calced pAlk	2.69	
"C" value (calculated)	2.11	at 22 Celcius
Sum of pH's	8.50	
Actual pH	6.65	
Difference (Langelier Index)	-1.25	

DISC: 5302
HIOQUALT.WR1

Program is effective for temperatures
between 5 and 35 degrees C.

PROGRAM CALCULATES LANGELIER INDEX
JOB NUMBER MT TODD
SAMPLE NUMBER EWP2
DATE 12/11/88

ION	IONIC WT	VALENCE	EQUIVLNT WT	p.p.m.	e.p.m.	EQUIVLNT CaCO3
CATION						
Aluminium	27.0	3	9.00		0.00	
Ammonium	18.0	1	18.00		0.00	
Calcium	40.1	2	20.05	10	0.50	25
Copper	63.5	2	31.75	0.002	0.00	
Hydrogen	1.0	1	1.00		0.00	
Ferrous Ion	55.8	2	27.90	2.6	0.09	
Ferric Ion	55.8	3	18.60		0.00	
Magnesium	23.4	2	11.70	20	1.71	85
Manganese	54.9	2	27.45	0.25	0.01	
Potassium	39.1	1	39.10	8	0.20	
Sodium	23.0	1	23.00	35	1.52	76
Silver	107.9	1	107.90		0.00	
Gold	197.0	1	197.00		0.00	
Zinc	65.4	2	32.70	0.26	0.01	
TOTAL CATION				75.9	4.04 (epm +)	
ANION						
Bicarbonate	61.0	1	61.00	165	2.70	135
Carbonate	60.0	2	30.00		0.00	0
Chloride	35.5	1	35.50	60	1.69	
Flouride	19.0	1	19.00		0.00	
Iodide	126.9	1	126.90		0.00	
Hydroxide	17.0	1	17.00		0.00	
Nitrate	62.0	1	62.00	8.5	0.14	
Phisphate (tri)	95.0	3	31.67		0.00	
Phosphate (di)	96.0	2	48.00		0.00	
Phosphate (mon)	97.0	1	97.00		0.00	
Sulphate	96.1	2	48.05	25	0.52	
Bisulphate	97.1	1	97.10		0.00	
Sulphite	80.1	2	40.05		0.00	
Bisulphite	81.1	1	81.10		0.00	
Sulphide	32.1	2	16.05		0.00	
Cyanide	26.0	1	26.00		0.00	
TOTAL ANION				258.50	5.06 (epm -)	

DATA INPUT (values marked * must be entered,
other values are optional)

pH *	7.05	Measured (actual)
Temp *	23	Celcius
Suspended solids	HERE	p.p.m.
Dissolved O2	HERE	p.p.m.
Dissolved CO2	HERE	p.p.m.
Measured TDS	HERE	p.p.m.
Conductivity	HERE	u mho/cm
Resistivity	HERE	ohm/m
Langelier "C"	HERE	Off nomogram.

CALCULATIONS

Total Hardness	110.41	mg/l as CaCO3
Total Alkalinity	135.25	mg/l as CaCO3
Cation/Anion balance	-11.22	%
Sum of Ions (TDS)	335	p.p.m.
Calced pCa	3.60	
Calced pAlk	2.57	
"C" value (calculated)	2.12 at	23 Celcius
Sum of pH's	8.29	
Actual pH	7.05	
Difference (Langelier Index)	-1.24	

DISC: 8302
HIGQUALT.WR1

Program is effective for temperatures
between 5 and 95 degrees C.

PROGRAM CALCULATES LANGELIER INDEX
JOB NUMBER MT TODD
SAMPLE NUMBER BWP6
DATE 2/11/88

ION	IONIC WT	VALENCE	EQUIVLNT WT	p.p.m.	e.p.m.	EQUIVLNT CaCO3
CATION						
Aluminium	27.0	3	9.00		0.00	
Ammonium	18.0	1	18.00		0.00	
Calcium	40.1	2	20.05	22	1.10	55
Copper	63.5	2	31.75	0.002	0.00	
Hydrogen	1.0	1	1.00		0.00	
Ferrous Ion	55.8	2	27.90	1.2	0.04	
Ferric Ion	55.8	3	18.60		0.00	
Magnesium	23.4	2	11.70	42	3.59	179
Manganese	54.9	2	27.45	0.2	0.01	
Potassium	39.1	1	39.10	4	0.10	
Sodium	23.0	1	23.00	83	3.61	180
Silver	107.9	1	107.90		0.00	
Gold	197.0	1	197.00		0.00	
Zinc	65.4	2	32.70	0.21	0.01	
TOTAL CATION				152.4	8.45 (epm +)	
ANION						
Bicarbonate	61.0	1	61.00	425	6.97	348
Carbonate	60.0	2	30.00		0.00	0
Chloride	35.5	1	35.50	115	3.24	
Flouride	19.0	1	19.00		0.00	
Iodide	126.9	1	126.90		0.00	
Hydroxide	17.0	1	17.00		0.00	
Nitrate	62.0	1	62.00	0.1	0.00	
Phosphate (tri)	95.0	3	31.67		0.00	
Phosphate (di)	96.0	2	48.00		0.00	
Phosphate (mon)	97.0	1	97.00		0.00	
Sulphate	96.1	2	48.05	5	0.10	
Bisulphate	97.1	1	97.10		0.00	
Sulphite	80.1	2	40.05		0.00	
Bisulphite	81.1	1	81.10		0.00	
Sulphide	32.1	2	16.05		0.00	
Cyanide	26.0	1	26.00		0.00	
TOTAL ANION				545.10	10.31 (epm -)	

DATA INPUT (values marked * must be entered,
other values are optional)

pH * 6.3 Measured (actual)
Temp * 25 Celcius
Suspended solids HERE p.p.m.
Dissolved O2 HERE p.p.m.
Dissolved CO2 HERE p.p.m.
Measured TDS HERE p.p.m.
Conductivity HERE u mho/cm
Resistivity HERE ohm/m
Langelier "C" HERE Off nomogram.

CALCULATIONS

Total Hardness 234.35 mg/l as CaCO3
Total Alkalinity 348.36 mg/l as CaCO3
Cation/Anion balance -9.94 %
Sum of ions (TDS) 698 p.p.m.
Calced pCa 3.26
Calced pAlk 3.16
"C" value (calculated) 1.15 at 25 Celcius
Sum of pH's 7.57
Actual pH 6.3

Difference -3.77
(Langelier Index)

DISC: 5502 Program is effective for temperatures
HIDQUALT.WR1 between 5 and 95 degrees C.

PROGRAM CALCULATES LANGELIER INDEX
JOB NUMBER MT TODD
SAMPLE NUMBER 2WP3
DATE 16/11/88

ION	IONIC WT	VALENCE	EQUIVLNT WT	p.p.m.	e.p.m.	EQUIVLNT CaCO3
CATION						
Aluminium	27.0	3	9.00		0.00	
Ammonium	18.0	1	18.00		0.00	
Calcium	40.1	2	20.05	24	1.20	60
Copper	63.5	2	31.75	0.002	0.00	
Hydrogen	1.0	1	1.00		0.00	
Ferrous Ion	55.8	2	27.90	0.7	0.03	
Ferric Ion	55.8	3	18.60		0.00	
Magnesium	23.4	2	11.70	30	2.56	128
Manganese	54.9	2	27.45	0.39	0.01	
Potassium	39.1	1	39.10	5	0.13	
Sodium	23.0	1	23.00	60	2.61	130
Silver	107.9	1	107.90		0.00	
Gold	197.0	1	197.00		0.00	
Zinc	65.4	2	32.70	0.14	0.00	
TOTAL CATION				120.1	6.54 (epm +)	
ANION						
Bicarbonate	61.0	1	61.00	335	5.49	275
Carbonate	60.0	2	30.00		0.00	0
Chloride	35.5	1	35.50	115	3.24	
Flouride	19.0	1	19.00		0.00	
Iodide	126.9	1	126.90		0.00	
Hydroxide	17.0	1	17.00		0.00	
Nitrate	62.0	1	62.00	0.1	0.00	
Phosphate (tri)	95.0	3	31.67		0.00	
Phosphate (di)	96.0	2	48.00		0.00	
Phosphate (mon)	97.0	1	97.00		0.00	
Sulphate	96.1	2	48.05	10	0.21	
Bisulphate	97.1	1	97.10		0.00	
Sulphite	80.1	2	40.05		0.00	
Bisulphite	81.1	1	81.10		0.00	
Sulphide	32.1	2	16.05		0.00	
Cyanide	26.0	1	26.00		0.00	
TOTAL ANION				460.10	8.94 (epm -)	

DATA INPUT (values marked * must be entered, other values are optional)

pH *	6.95	Measured (actual)
Temp *	23	Celcius
Suspended solids	HERE	p.p.m.
Dissolved O2	HERE	p.p.m.
Dissolved CO2	HERE	p.p.m.
Measured TDS	HERE	p.p.m.
Conductivity	HERE	u mho/cm
Resistivity	HERE	ohm/m
Langelier "C"	HERE	Off nomogram.

CALCULATIONS

Total Hardness	188.06	mg/l as CaCO3
Total Alkalinity	274.59	mg/l as CaCO3
Cation/Anion balance	-15.53	%
Sum of Ions (TDS)	520	p.p.m.
Calced pCa	3.22	
Calced pAlk	2.26	
"C" value (calculated)	2.14 at	23 Celcius
Sum of pH's	7.63	
Actual pH	6.95	
Difference (Langelier Index)	-0.68	

DISC: 3302
H2OQUALT.WR1

Program is effective for temperatures
between 5 and 95 degrees C.

PROGRAM CALCULATES LANGELIER INDEX
JOB NUMBER MT TODD
SAMPLE NUMBER BWF10
DATE 9/11/88

ION	IONIC WT	VALENCE	EQUIVLNT WT	p.p.m.	e.p.m.	EQUIVLNT CaCO3
CATION						
Aluminium	27.0	3	9.00		0.00	
Ammonium	18.0	1	18.00		0.00	
Calcium	40.1	2	20.05	5	0.25	12
Copper	63.5	2	31.75	0.002	0.00	
Hydrogen	1.0	1	1.00		0.00	
Ferrous Ion	55.8	2	27.90	0.2	0.01	
Ferric Ion	55.8	3	18.60		0.00	
Magnesium	23.4	2	11.70	12	1.03	51
Manganese	54.9	2	27.45	0.069	0.00	
Potassium	39.1	1	39.10	4	0.10	
Sodium	23.0	1	23.00	18	0.78	39
Silver	107.9	1	107.90		0.00	
Gold	197.0	1	197.00		0.00	
Zinc	65.4	2	32.70	0.225	0.01	
TOTAL CATION				39.4	2.17 (epm +)	
ANION						
Bicarbonate	61.0	1	61.00	110	1.80	90
Carbonate	60.0	2	30.00		0.00	0
Chloride	35.5	1	35.50	55	1.55	
Flouride	19.0	1	19.00		0.00	
Iodide	126.9	1	126.90		0.00	
Hydroxide	17.0	1	17.00		0.00	
Nitrate	62.0	1	62.00	0.1	0.00	
Phosphate (tri)	95.0	3	31.67		0.00	
Phosphate (di)	96.0	2	48.00		0.00	
Phosphate (mon)	97.0	1	97.00		0.00	
Sulphate	96.1	2	48.05	5	0.10	
Bisulphate	97.1	1	97.10		0.00	
Sulphite	80.1	2	40.05		0.00	
Bisulphite	81.1	1	81.10		0.00	
Sulphide	32.1	2	16.05		0.00	
Cyanide	26.0	1	26.00		0.00	
TOTAL ANION				170.10	3.46 (epm -)	

DATA INPUT (values marked * must be entered, other values are optional)

pH *		6.15	Measured (actual)
Temp *		23	Celcius
Suspended solids	HERE		p.p.m.
Dissolved O2	HERE		p.p.m.
Dissolved CO2	HERE		p.p.m.
Measured TDS	HERE		p.p.m.
Conductivity	HERE		u mho/cm
Resistivity	HERE		ohm/m
Langelier "C"	HERE		Off nomogram.

CALCULATIONS

Total Hardness	63.75	mg/l as CaCO3
Total Alkalinity	90.16	mg/l as CaCO3
Cation/Anion balance	-22.32	%
Sum of Ions (TDS)	209	p.p.m.
Calced pCa	3.90	
Calced pAlk	2.74	
"C" value (calculated)	2.09	at 23 Celcius
Sum of pH's	8.74	
Actual pH	6.15	
Difference (Langelier Index)	-2.59	

4.0 PROCESS PLANT DESIGN

4.1 METALLURGY - REVIEW OF TESTWORK

(1) Mineralogy

The Batman orebody is composed of greywacke host rock finely veined by sulphide and carbonate bearing quartz. The veining is up to centimetre order width at 10 centimetre order spacings.

Greywacke is assumed to be composed of hornfels, which usually derives from the metamorphosis of clay based rocks, and is predominantly Aluminum and Magnesium silicates.

The quartz carries small amounts (<1%) of Iron sulphides (pyrite/pyrrhotite/marcasite) and Lead, Zinc and Copper sulphides (galena/sphalerite/chalcopyrite) in the primary ore zone. The transition ore zone through to the weathered (surface) ore zone displays a typical sulphide breakdown to goethite/limonite/haematite.

Mineralogical examination has noted that the chlorite - Magnesium silicates - have weathered to kaolinite from the primary to the weathered ore zone. This comprises 20% of the ore, with 50% being quartz and 30% muscovite (Pottassium, Aluminum silicates).

The orebody is composed of approximately 64% primary ore, 22% transition ore and 14% weathered ore. The specific gravities are 2.78, 2.67 and 2.48 respectively.

An ICP scan indicates 65 to 67% silica content, with Aluminum, Iron and Potassium as oxides making up the balance. The scan also indicates Mercury is not a potential problem and confirms that Copper is the dominant base metal. Silver assays are anomalous, so for design purposes a grade of 2 g/t was assumed for a 1.3 g/t Gold head grade. High Rubidium content was also noted, although its significance is not known (Table 1).

The chalcopyrite content should not present any problems with regard to cyanide consumption. From work done by Healy and Tabachnik (1958 - "Chemistry of Cyanidation") it is shown that chrysocolla and chalcopyrite are the least soluble of the Copper minerals, with about 10% soluble in cyanide solutions.

Gold is assumed to be associated with the sulphide mineralisation in the quartz veining. Work is continuing to determine associations and distribution. Some Gold sighted has been extremely fine at 2-4 microns size, ex solution blebs in chalcopyrite and also contained in Iron oxide. Distinct grains up to 150 microns in quartz are also reported. Recent information indicates that Gold is intergrown with, or in solid solution in, Chalcopyrite and Bismuthinite.

TABLE 4.1

ICP Scans of Composite Ore Samples, Full Drill Core

Ore Type Composite No	Weathered B1WB1-1D	Transitional B1TB1-1D	Primary B1PB1-1D
AL_2O_3%	16.4%	15.9%	13.6%
CaO	0.02	0.06	0.12
Fe_2O_3	8.45	7.90	7.90
K_2O	3.62	4.48	4.08
MgO	0.63	0.57	2.28
MnO	0.04	<0.01	0.04
Na_2O	0.20	0.18	0.17
P_2O_5	0.04	0.04	0.06
SiO_2	65.2	66.2	67.3
TiO_2	0.63	0.52	0.72
As	0.015	0.026	0.009
Ba	0.071	0.062	0.053
Cd	<0.002	<0.002	<0.002
Co	0.003	<0.002	0.003
Cr	0.008	0.010	0.015
Cu	0.101	0.141	0.253
La	<0.002	<0.002	<0.002
Mo	<0.002	<0.002	<0.002
Nb	<0.020	<0.020	<0.020
Ni	0.002	0.002	0.004
Pb	0.016	<0.005	0.005
Sn	<0.005	<0.005	<0.005
Sr	0.002	0.004	0.003
Ta	<0.020	<0.020	<0.020
V	0.010	0.007	0.011
W	<0.020	<0.020	<0.020
Y	0.003	0.004	0.002
Zn	0.030	0.028	0.036
Zr	0.016	0.019	0.021
S	0.007	0.017	0.490
Hg(ppm)	<0.01 ppm	<0.01 ppm	<0.01 ppm
Ag	70	3	3
Cs	<10	15	10
Ga	18	12	8
Ge	<4	<4	<4
In	<10	<10	<10
Rb	145	220	170
Tl	10	10	<10
Ce	50	80	70
Se	<2	3	<2
Sb	6	<4	<4
Te	<10	<10	<10
Th	14	30	20
U	<4	<4	<4

(2) Metallurgical Data

Testwork was carried out at AMDEL over the Christmas period of 1988. Only parts of the raw data had been sighted during the process design exercise.

A. Comminution

The comminution circuit was designed on the basis of Bond ball mill work index data supplied by Billiton Australia Gold Pty. Ltd. as follows:

Weathered Ore	14.3,	9.2		kW/hrs/tonne
Transition Ore	12.2,	22.2		kW/hrs/tonne
Primary Ore	23.3,	28.4,	22.4	kW/hrs/tonne

and from impact crusher indices, unconfined compressive strength and abrasion index data determined by NEDPAC Engineering (detailed in Design Criteria).

The average of the primary ore work indices was used to size mills (WI = 25). The Semi Autogenous Mill feed size was dictated by the capability of the 1.2 x 1 metre jaw crusher (d80 = 200 mm) at 400 tph. SAG mill discharge was specified at 2 mm to provide a near optimum feed size to the ball mill circuit.

Ball mill product size for leaching was specified by Billiton Australia Gold Pty. Ltd. as 90% passing 75 microns. To translate this to a d80 size, data from a 106 micron agitation leach residue sizing was used to determine the slope of the size distribution, resulting in a d80 of 53 microns for leach feed. This decision fits logically with the residue analyses obtained by size from leach tests (Table 4.2) in the context of optimum leach performance. Testwork later indicated that for a 48 hours leach duration a product d80 of 75 microns achieved Gold extractions very close to those obtained at a grind d80 of 53 microns as shown in Figures 4.2 and 4.3.

TABLE 4.2 A

Mt. Todd Cyanidation Residue (Gold by Size) Lot 1-4 Comp

Product	Weight %	Assay Gold s/t	Assay Ag ppm	Distribution % Gold	Distribution % Ag
+106μm	17.99	0.34	82.01	45.69	21.11
+75μm	13.92	0.23	68.09	23.91	10.89
+53μm	10.61	0.17	57.48	31.47	6.22
+38μm	9.65	0.21	47.83	15.14	5.66
-38μm	47.83	0.00		1.79	56.12
Head(calc.)	100.00	0.13		100.00	100.00

TABLE 4.2 B

Mt. Todd Cyanidation Residue (Gold by Size) Lot 5-8 Comp

Product	Weight %	Assay Gold s/t	Assay Ag ppm	Distribution % Gold	Distribution % Ag
+106μm	19.54	0.25	80.46	45.19	33.38
+75μm	14.87	0.17	65.59	23.39	16.93
+53μm	10.84	0.12	54.75	12.03	18.52
+38μm	9.85	0.19	44.90	17.31	5.61
-38μm	44.90	0.00		2.08	25.57
Head(calc.)	100.00	0.11		100.00	100.00

TABLE 4.2 C

Mt. Todd Cyanidation Residue (Gold by Size) Lot 9-11 Comp

Product	Weight %	Assay Gold s/t	Assay Ag ppm	Distribution % Gold	Distribution % Ag
+106μm	21.17	0.12	78.83	16.88	18.28
+75μm	15.39	0.27	63.44	27.60	13.29
+53μm	10.56	0.28	52.85	19.64	22.79
+38μm	9.69	0.29	43.19	18.67	8.36
-38μm	43.19	0.06		17.21	37.28
Head(calc.)	100.00	0.15		100.00	100.00

TABLE 4.2 D

Mt. Todd Cyanidation Residue (Gold by Size) Sp. Deep Pry Comp

Product	Weight %	Assay Gold s/t	Assay Ag ppm	Distribution % Gold	Distribution % Ag
+106μm	19.24	0.28	80.76	21.50	19.24
+75μm	14.11	0.55	66.65	30.97	14.11
+53μm	10.64	0.50	56.01	21.23	10.64
+38μm	8.89	0.37	47.12	13.13	8.89
-38μm	47.12	0.07		13.16	47.12
Head(calc.)	100.00	0.25		100.00	100.00

B. Leaching

An optimum leach feed d80 of 53 microns was confirmed by receipt of the leaching tests conducted at grind sizes of 106, 75 and 53 micron d80 (Table 4.3). As mentioned previously, a d80 of 75 microns was selected at longer leach times.

Of some concern is the variability of work index for the various ore types, combined with the sensitivity of recovery response to grind size. Assumptions made in the laboratory as to conditions required to achieve a particular (optimum) liberation size may not be valid over a range of samples. This was a possible explanation for the inconsistency in performance evident in the leaching testwork carried out at varying pH, % solids and cyanide addition (75 micron d80), however residue sizing checks showed size distributions very close to the 75 microns d90 target.

TABLE 4.3

Lot No.	Size microns	Gold Ext. 24 hr	Silver Ext. 24 hr	Max Gold Ext.
1-4	106	79	42	82
	75	83	43	91
	53	85	39	92
5-8	106	64	47	86
	75	66	52	90
	53	74	52	93
9-11	106	74	37	84
	75	76	22	87
	53	79	38	90
S.D.P.	106	75	38	83
	75	77	54	87
	53	79	40	87

Test Conditions

Cyanide	500 mg/l
pH	~11
% solids	40
duration	48 hours

Twenty eight leach tests were conducted (7 on each of 4 ore types) at varying cyanide addition, pH and pulp density.

The major factor influencing performance within an ore category was cyanide concentration. Policy on most operating plants is to sustain cyanide concentration at 500 - 700 mg/litre (0.05-0.07%) and thus several test results were rejected where concentrations were at 0.01% or less (Table 4.4). The exception was lot 1-4. The results in Table 4.3 are superior overall in terms of ultimate extraction after 48 hours, yet are poorer at the 24 hour mark than the results in Table 4.4. Rejected tests are marked (*) and an overall recovery was calculated to be 85% from the maximum (48 hour) results. A further effect appeared to be high pulp density as the 48% solids tests were all poor.

Aside from the concern about grind size, further possible problems could be:

- shortage of dissolved oxygen, due either to the presence of oxygen consuming species or grinding in a reducing environment.
- Gold either very fine, intergrown to a significant extent with sulphides, or surface coated.
- Gold in the form of Tellurides or electrum.

There are indications, particularly in transition ore tests, of excessive initial cyanide consumption.

TABLE 4.4

Cyanide Leach Tests - Varying Conditions

Lot No.	Cyanide %	Cyanide Consumption	pH	24 hr Gold Extraction	Max Gold Ext.	%Solids
1-4	0.050	0.35	9.5	87.6	90.7	40
	0.050	0.54	10.3	79.2	88.3	40
	0.015	0.08	11	92.8	92.8	40
	0.025	0.16	11	85.8	87.5	40
	0.100	0.46	11	75.7	88.0	39
	0.050	0.28	11	78.3	84.1*	48
	0.050	0.53	11	84.1	84.1	33
5-8	0.050	1.44	9.5	91.6	95.0	40
	0.050	1.14	10.3	84.6	86.8	40
	0.015	0.45	11	59.5	68.7*	40
(0.01)	0.025	0.6	11	45.1	75.4*	40
	0.100	1.27	11	69.9	74.3*	39
	0.050	0.84	11	74.7	79.5*	48
	0.050	1.15	11	91.5	91.5	32
9-11	0.050	0.87	9.5	81.5	81.5	40
	0.050	0.59	10.3	79.1	81.8	40
	0.015	0.15	11	64.8	72.5*	40
	0.025	0.30	11	67.9	77.8*	40
	0.100	0.62	11	82.3	82.3	39
	0.050	0.31	11	68.2	70.5*	48
	0.050	0.41	11	76.8	79.9	33
S.D.P.	0.050	0.31	9.5	73.4	84.3	40
	0.050	0.52	10.3	77.9	84.5	40
	0.015	3.08	11	69.9	76.2*	40
	0.025	3.03	11	79.2	89.1	40
	0.100	3.37	11	80.9	80.9*	39
	0.050	4.73	11	83.5	83.5	48
	0.050	2.98	11	81.1	83.3	33

Test condition

Size p90 75 micron

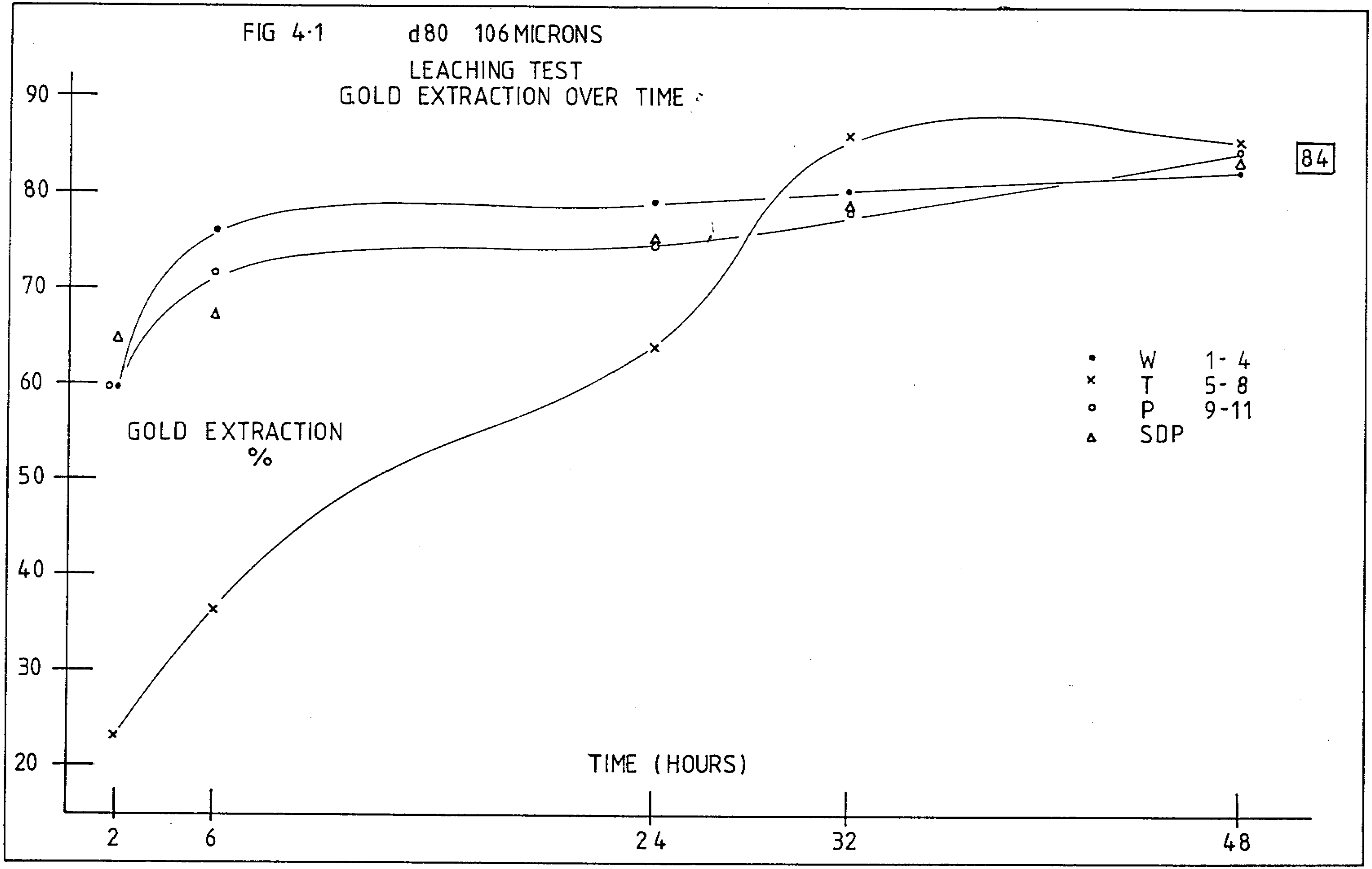


FIG 4·1 d80 106 MICRONS
LEACHING TEST
GOLD EXTRACTION OVER TIME
GOLD EXTRACTION %
TIME (HOURS)
90
80
70
60
50
40
30
20
2
6
24
32
48
W 1- 4
T 5- 8
P 9-11
SDP
84

FIG 4·1

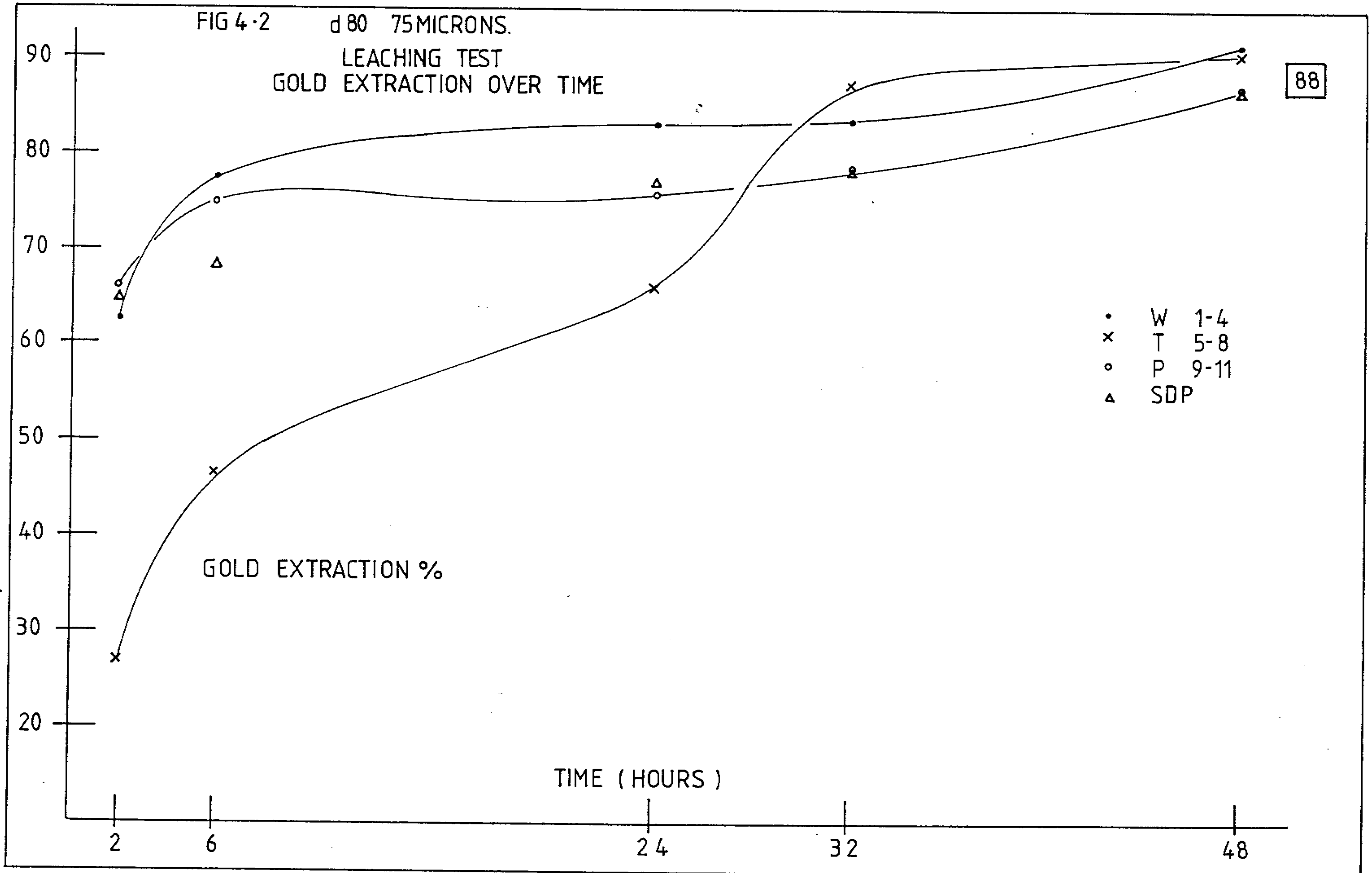


FIG 4·2
d 80 75MICRONS.
LEACHING TEST
GOLD EXTRACTION OVER TIME
88
90
80
70
60
50
40
30
20
W 1-4
T 5-8
P 9-11
SDP
GOLD EXTRACTION %
TIME (HOURS)
2
6
24
32
48

FIG 4·2

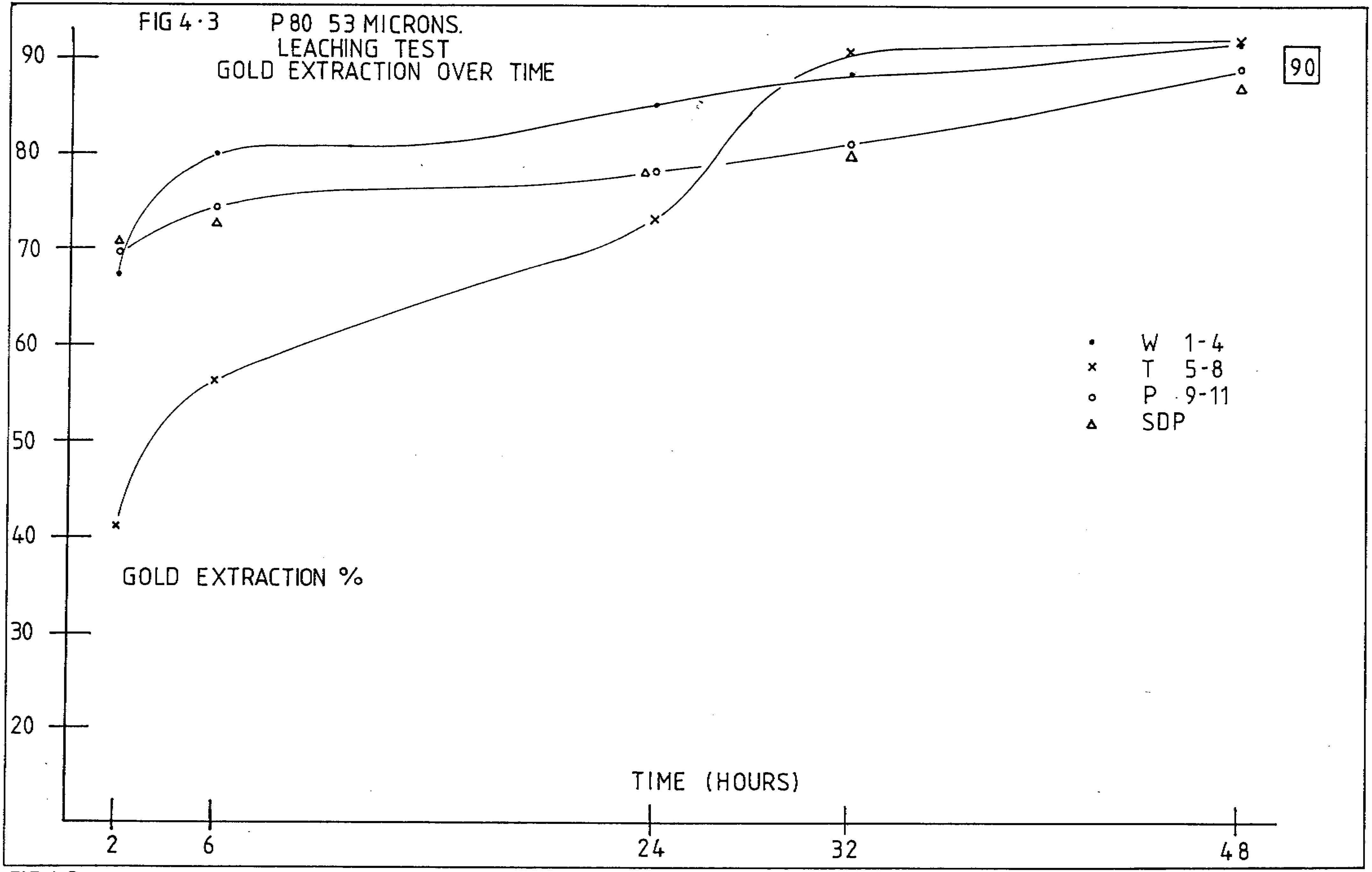


FIG 4·3
P 80 53 MICRONS.
LEACHING TEST
GOLD EXTRACTION OVER TIME
90
80
70
60
50
40
30
20
GOLD EXTRACTION %
W 1-4
T 5-8
P 9-11
SDP
90
TIME (HOURS)
2
6
24
32
48

FIG 4·3

In summary, the leaching results in Table 4.4 are extremely inconsistent, displaying several distinct extraction characteristics over time within an ore type. Several basic parameters remain undecided, and appropriate testwork should be commissioned with an alternate laboratory (AMMTEC, NORMET) to assess:

1. Grind size
2. Leach duration
3. Recovery

It would also assist to have AMDEL obtain a check on:

1. Dissolved oxygen levels
2. Residue sizings
3. Head assays

There is sufficient evidence of good extraction performance within each composite group to justify further work. Silver extraction of 40% agreed closely with Minproc's experience, however a reliable estimate of Silver head grade is required to finalise elution design. A Gold recovery of 85% on average is a realistic assessment for the pre-feasibility C.I.P. plant design on first pass, however for the purposes of the study a recovery of 90% was used for the following reasons:

1. The leaching tests conducted at varying reagent concentrations, pH and %solids (series A) indicate a sufficient number of good and bad results to make a simple average recovery meaningless. The aim of this program was to seek potential problems, rather than establish recovery.

2. Further examination of the tests conducted at varying grind size (series B) reveals that the tests are extraordinarily consistent by ore type, and display a logical progression in performance with grind size.

3. The average Gold in the residues from series B tests was 0.12 g/t, or 90.7% recovery to 1.29 g/t feed. The weighted average (14% W, 22% T, 64% P) was 0.1376% g/t which equates to 89.33% recovery to a 1.29 g/t feed.

4. Reprecipitation of Gold was a feature of the series A tests, as was a wide range of leaching rates. The series B test, 5-8 comp. sample displayed extreme cyanide consuming behaviour. These problems can be solved in practice with an overall increase in performance.

5. Increased residence time for leaching may be required; investigation of the oxygen injection or hydrogen peroxide addition processes should be considered to minimise residence time.

<u>Summary</u>

An average 90% recovery is achievable. A means should be sought to reduce the time required to a consistent 24 hours, or incorporate extra leaching capacity at the next stage of design. The option of grinding to a d80 of 75 microns for a recovery of 88% is realistic, and is used in the financial analysis.

C. Adsorption

A model developed by Minproc was used to design the adsorption circuit. The model is based on the Mintek adsorption model and performs a mass balance over each circuit size. Carbon inventory, stage efficiencies and Gold inventory are calculated to obtain a desired solution grade for a particular number of adsorption stages.

The basis is the carbon adsorption test. As data was not available during the design process, a circuit was designed on the basis of "k" and "n" parameters of 150 and 0.6 respectively in the equation:

$\triangle Au_c = k\, Au_s t^n$
$\triangle Au_c$ = change in Gold on carbon at time "t" (mg/l)
Au_s = equilibrium gold tenor (mg/l)
t = equilibrium time (hours)

This is rearranged to:

$$\log \frac{(\triangle Au_c)}{(Au_s)} = \log k + n \log t$$

which allows determination of "k" as the intercept at logt = 0, and "n" the slope.

AMDEL testwork, conducted at 10 g/l carbon (PICA G210 AS 6x12 ASTM) yielded average values of:

Lot No.	k	n
1-4	282	0.67
5-8	195	1.00
9-11	501	1.02
SDP	324	0.69

(See Figure 4.1)

There is some dispute as to the range of time over which the model is valid, and thus the number of data points taken into account can alter the results considerably. The 24 hour data points were ignored, as the model is generally only valid for the first 6 - 8 hours. The values of 150 and 0.6 were considered to be adequate for preliminary design purposes in the light of these test results. This is largely a reflection of the fact that fresh carbon gives high intercept values, whereas in practice carbon loses much of its adsorption capability quite rapidly.

FIG 4·4 ADSORPTION KINETICS

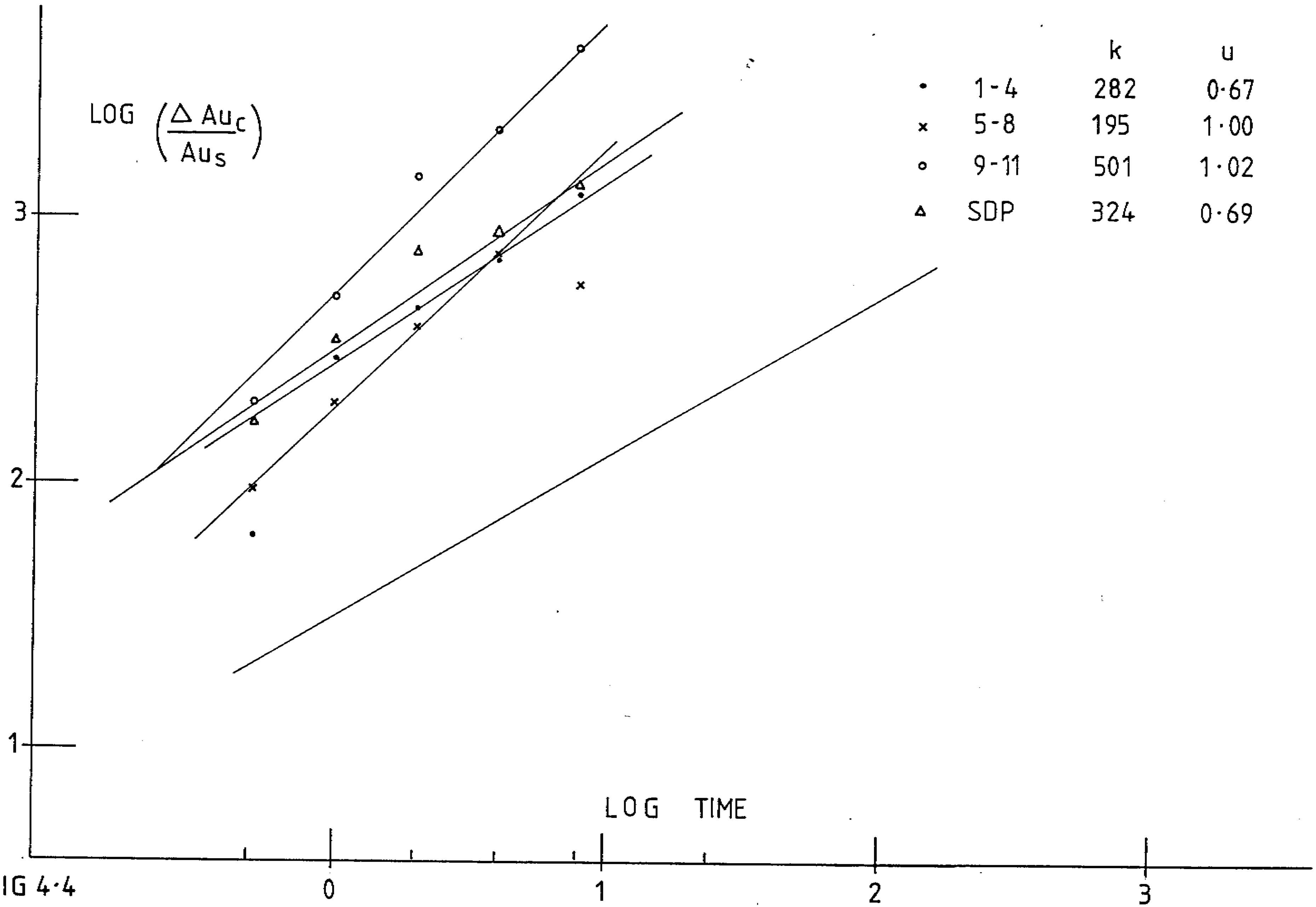


FIG 4·4

4.2 DESIGN PHILOSOPHY

A primary aim in the design process has been to contain capital costs. Some small further economies may be achieved when more extensive metallurgical test data is available, mainly in the C.I.P. area with larger scale tankage, in the thickener area by adopting a 12 metre unit, and by use of a smaller desorption plant.

The majority of the ore is very hard, and to achieve the required particle size a substantial investment in mills was necessary. This in turn creates a high operating cost with regard to steel and power. The variable speed SAG mill will permit maximum economy with regard to power usage.

The desorption process was sized to permit one strip per day, in turn permitting maximum flexibility if extra capacity is required. The small increase in capital cost over a plant size demanding 2 strips per day was justified on this basis.

To maximise the existing water resource, a tailing thickener is incorporated. Water recovery from tailings dams is not dependable, particularly where the area of the dam is large and evaporation rates are high. Dam design and site selection specifically attempts to minimise catchment, however a significant quantity of water is available during the wet season from tailings reclaim.

As the design exercise had to meet a tight schedule, the leaching/adsorption area was designed largely on the basis of experience. The test data available was reviewed late in the study period, and was relatively inconsistent in terms of indicative ore response. The data does reinforce the need for more extensive testwork on a larger scale and by a more rigorous procedure.

The foremost consideration in C.I.P. plant design is to incorporate the facilities to allow the operator to achieve maximum possible gold recovery. The grinding circuit must have the capacity to provide adequate liberation for any ore type. The leaching circuit is less critical due to emerging technologies such as CILO, Afrox or the Degussa processes for leaching, which can be added on at a small cost. It is, however, essential that the leaching characteristics of the range of ore types be firmly established.

4.3 DESIGN CRITERIA

Design Criteria are set out for the 2 million TPA and the 3 million TPA cases. They are based on guidelines supplied by Billiton Australia Gold Pty Ltd, testwork and data available at the time of writing this report, performance characteristics derived from process models and Minproc's experience of similar processes.

Initial design and costing was based on a Gold extraction of 90% and carbon loadings of 2200 g/t. After review of testwork and head grade the elution batch size changes from 3.4 to 3.2 tonnes of carbon per day. It is desireable to leave the column size (and cost) at 3.4 tonnes to cover feed grade, recovery and carbon loading variations.

The 3 million TPA case shows that batch size is getting too large for a single strip each day, and thus two strips have been nominated each at 2.8 tonnes carbon. Capital cost reduction would be small, the major compromise being loss of flexibility.

The tailings dam specification is 20 million tonne capacity of solids and thus 10 year life, although the design criteria states 7 years.

In summary:

1. Design and Costing carried out for 90% recovery from 1.5 g/t head grade. This effects cost of desorption. A 24 hour leach time was used, and 53 micron grind.

2. Design Criteria were finalised at 85% from 1.5 g/t, 24 hour leach at 53 micron grind.

3. Capital Cost of grinding and leaching areas was modified for final financial analysis to 88% recovery from 1.29 g/t head grade and 43 hours leaching plus 5 adsorption. The grind size was 75 microns d80. A revised design criteria is attach in Appendix 3. Also included is revised mass balances and mill selection data.

MT. TODD GOLD PROJECT - C.I.P./C.I.L. STUDY - DESIGN CRITERIA - 2MTPA AT 85% Au RECOVERY

CATEGORY			UNITS	DATA			
Schedules							
Mining	Tonnes	P.A.	Tonnes	2,000,000	Ore		
	Weeks	P.A.		52			
	Days	P.W.		6			
	Shifts	P.D.		2			
	Hours	P.S.		10			
	Tonnes	P.W.	Tonnes	38,462			c
	Tonnes	P.D.	Tonnes	6,410			c
Crushing	Tonnes	P.W.	Tonnes	38,462			c
	Days	P.W.		6			
	Shifts	P.D.		2			
	Hours	P.S.		10			
	% Availby			80			
Design	T.P.H.		Tonnes/Hr	401			c
Milling	Tonnes	P.W.	Tonnes	38,462			c
	Days	P.W.		7			
	Shifts	P.D.		2			
	Hours	P.S.		12	check		
	% Availby			92.50			
Nominal	T.P.H.		Tonnes/Hr	248			c
Design	T.P.H.		Tonnes/Hr	250			
Stripping	Days	P.W.		7			
	Strips	P.D.		1			
Design	Batch Size		Tonnes C	3.18	/strip		c
Bullion	Kg Au	P.A.	Kg	2,550			c
	Oz Au	P.A.	Oz	81,984			c
Ore Characteristic							
Grade	Au		g/t	1.50			
	Ag		g/t	2.00			
	S.G. (Dry)			2.48	W	Dis % 14	
				2.67	T	22	
				2.78	P	64	
				2.71			
	S.G. Av		%	******	check		
	Moisture						
	Bulk S.G.			1.45	W		
				1.48	T		
				1.48	P		
	U.C.S.		MPa	24.00	W		
				29.10	T		
				57.10	P		
	Impact		kWh/t	4.10	W		

CATEGORY			UNITS	DATA		
	Crusher Index			2.60	T	
				7.80	P	
	Abrasion Index			0.0180	W	
				0.0046	T	
				0.0660	P	
	Angle of Repose		Degrees	36.00	(Fine)	
	Free	SiO2	%	******		
Bond	B.M.	W.I.	kWh/t	11.70	W	
				17.20	T	
				24.70	P	
	R.M.	W.I.	kWh/t	******	Use B.M.	
<u>Crushing</u>						
Design	T.P.H.		Tonnes/Hr	401		c
R.O.M.	Size	d100	m.m.	1000		
Truck	Size		Tonnes	85		
R.O.M.	Bin Capacity		Tonnes	170		c
C.O.S.	Capacity	L	Hours	24		
			Tonnes	9615		c
Design	Product d80 Size		m.m.	200		
F.O.S.	Capacity	L	Hours	48		
			Tonnes	11880		c
Grinding Circuit						
Design	T.P.H.		Tonnes/Hr	250		c
	Feed d80 Size		m.m.	200		
	Product d80 Size		m.m.	0.053		
Recirculating Load					UF/OF	
	SAG Mill		%	200		
	Ball Mill		%	150		
Classification						
SAG Mill	Product					
	Density		%Sol. WT.	45		
	Pulp S.G.		t/m^3	1.40		
			%Sol. Vol.	23		
	T.P.H. Pulp		Tonnes/Hr	556		
	Vol Flow		m^3/Hr	398		
Ball Mill	Product					
	Density		%Sol. WT.	42		
	Pulp S.G.		t/m^3	1.36		
			%Sol. Vol.	21		c

CATEGORY		UNITS	DATA	
Ball Mill	T.P.H Pulp	Tonnes/Hr	595	c
	Vol Flow	m^3/Hr	437	c
Leaching/Adsorption				
Design	T.P.H.	Tonnes/Hr	250	c
	Density	%Sol. WT.	42	
Solids	Flowrate	m^3/Hr	437	c
Solut.n	Flowrate	m^3/Hr	345	
Sol.n	Feed Grade	g/t	0.92	
Trash Screen				
	Aperture	Microns	1000	
C.I.P.	Hybrid			
Leach Time	Total	Hr	21.50	
	Absorb	Hr	5	From Model
	Total	Hr	26.50	c
No. of Stages	Total		9	
	Leach		4	
	Absorb		5	
Tank Vol.	Total	m^3	11590 (Live)	c
	Leach	m^3	9403	c
	Adsorb	m^3	2187	c
Intertank	Screen		Airswept, Flooded W'Wire	
	Aperture	Microns	833	
Leach Recovery				
	Au	%	85 check	
	Ag	%	40 check	
Carbon	Type		Pica G210A5 (Amdel)	
	Size	ASTM	6 x 12	
Design Tail Grade	Au	g/t	0.23	
Design Sol.n Tail	Au	g/t	0.01	
Design Carbon				
	Au Load	g/t	2250	c
	Ag Load	g/t	*****	
	Stripped			
	Au Stripped	g/t	50	
	Ag Stripped	g/t	*****	
	Concentration	kg/m^3	14.18	c
	Mass Flow	kg/Hr	138.45	c
Total	Inventory	kg	31000 Adsorb	From Model
Stage Efficiency		%	61	From Model
Gold Inventory		kg	28	From Model
Res. Time per Stage		Hr	44.78	c
	Transfer		Cont.s Airlift	
	Removal		Recessed Imp. Pump	

CATEGORY			UNITS	DATA	
Recovery Screen					
	Aperture		Microns	800	
Thickening					
Design	T.P.H.		Tonnes	250	c
	F Density		%Sol. WT.	39	
	Pulp S.G.		t/m^3	1.33	c
			%Sol. Vol	19	c
	T.P.H. Pulp		Tonnes/Hr	641	c
	Vol. flow		m^3/Hr	483	c
Av Settling Rate					
	Compression		m/Hr		
	Bulk		m/Hr		
	Sol. Upflow		m/Hr		
	Max % Sol.		%Sol.WT.		
Design	%Sol.		%S0l. WT.		
	Flocc Use		g/t	50 Max35 Min	
Desorption/Electrowin/Refine					
Stripping System				AARL, Cold Acid Wash	
Carbon	Batch	Size	Tonne C	3.18	c
No. of Strips		P.W.		7	
	Strips	P.D.		1	
Metal Stripped		P.W.			
	Au		kg	49.04	c
	Ag		kg	30.77	
	Total		kg	80	c
Electrowin					
	No. Cells			1	
	No. Caths			9	
	Current		AMPS	700	
Barren	Eluate		PPM	10 Max	
Electrorefine					
	No. Cells			1	
	No. Caths			10	
	Current		AMPS	500	
Carbon Regen. Rate			kg/Hr	250	
Reagent Usage					
C.I.P.					
	NaCN		kg/t	0.80	
	Lime		kg/t	1.10	
	NaOH		kg/t	0.20	
	Pb(NO)3		kg/t	0.40 Max	
	Flocc		kg/t	0.05	

CATEGORY		UNITS	DATA		
Reagent Usage					
	Carbon	t/yr	31.00		
	G. Media	kg/t	1.50		
Desorb					
	NaCN	kg/t C	25.00	2% W/V	
	NaOH	kg/t C	20.00	2% W/V	
	HCI	kg/t C	85.00	3% W/V	
Water Balance					
Grind Product					
	T.P.H.	Tonnes/Hr	250		c
	Density	%Sol. WT.	42		
	Pulp S.G.	t/m^3	1.36		c
		%Sol. Vol.	21		c
	Pulp T.P.H.	Tonnes/Hr	595		c
	Vol. Flow	m^3/Hr	437		c
	Water T.P.H.	Tonnes/Hr	345		
Tail					
	Density	%Sol. WT.	39		
	Water	Tonnes/Hr	391		c
Thickener U/F					
	Density	%Sol. WT.	60		
	Water Loss	Tonnes/Hr	167		c
	Reclaim	Tonnes/Hr	25	15	%Rec F
Total Water Loss		Tonnes/Hr	142		
	Return (To Grinding)	Tonnes/Hr	249		c
Grind Make-up		Tonnes/Hr	96		c
Desorb Use		Tonnes/Hr	46		c
Other		Tonnes/Hr	2	check	
	Total	Tonnes/Hr	144		c
		TPD	3448		c
		TPW	24136		c
		TPA	1255072		c
Tailings Dam					
	Tonnes		14000000		c
Mine Life		Years	7		
	Density	%Sol. WT.	80	check	
	S.G. Tail	t/m^3	2.02		c
Dam Volume		m^3	8658818		c

MT. TODD GOLD PROJECT - C.I.P./C.I.L. STUDY - DESIGN CRITERIA
3 MTPA at 85% Recovery

CATEGORY			UNITS	DATA		
Schedules						
Mining	Tonnes	P.A.	Tonnes	3,000,000	ore	
	Weeks	P.A.		52		
	Days	P.W.		6		
	Shifts	P.D.		2		
	Hours	P.S.		10		
	Tonnes	P.W.	Tonnes	57,692		c
	Tonnes	P.D.	Tonnes	9,615		c
Crushing	Tonnes		Tonnes	57,692		c
	Days	P.W.		6		
	Shifts	P.D.		2		
	Hours	P.S.		10		
	% Availby			80		
Design	T.P.H.		Tonnes/Hr	601		c
Milling	Tonnes	P.W.	Tonnes	57,692		c
	Days	P.W.		7		
	Shifts	P.D.		2		
	Hours	P.S.		12	check	
	% Availby			92.50		
Nominal	T.P.H.		Tonnes/Hr	371		c
Design	T.P.H.		Tonnes/Hr	380		
Stripping	Days	P.W.		6		
	Strips	P.D.		2		
Design Batch Size			Tonnes c	2.79	per strip	c
Bullion	Kg Au	P.A.	Kg	3,825		c
	Oz Au	P.A.	Oz	122,977		c

CATEGORY			UNITS	DATA		
Ore Characteristic						
Grade	Au		g/t	1.50		
	Ag		g/t	2.00	check	
	S.G. (dry)			2.48	W Dist.%	14
				2.67	T	22
				2.78	P	64
	S.G.	Av		2.71		
	Moisture		%	******	check	
	Bulk	S.G.		1.45	W	
				1.48	T	
				1.48	P	
	U.C.S.		MPa	24.00	W	
				29.10	T	
				57.10	P	
	Impact		kWh/t	4.10	W	

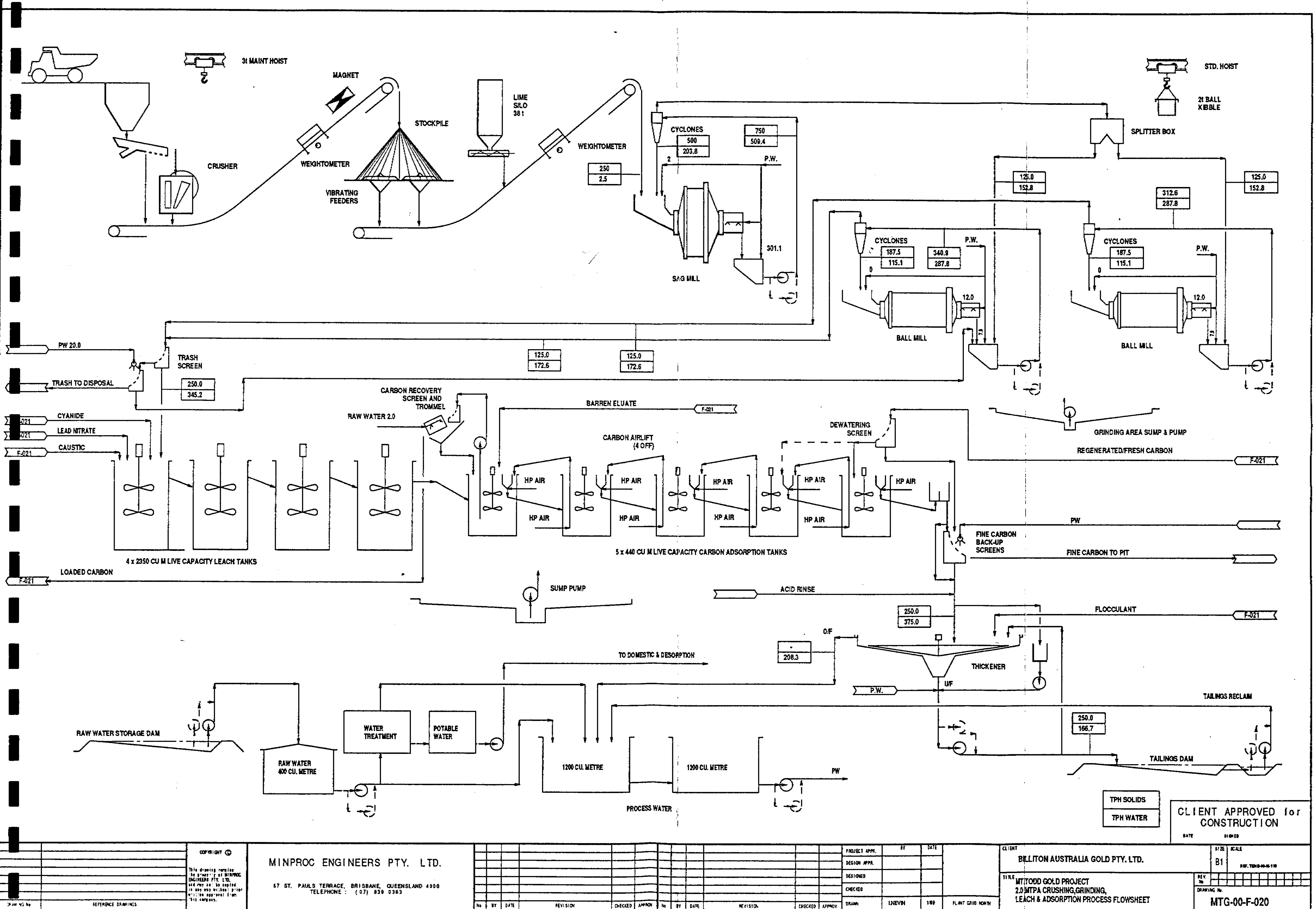
CATEGORY		UNITS	DATA	
	Crusher Index		2.60 T 7.80 P	
	Abrasion Index		0.0180 W 0.0046 T 0.0660 P	
	Angle of Repose	Degrees	37.00 (Fine)	
	Free Sio2	%	******	
Bond	B.M. W.I.	kWh/t	11.70 W 17.20 T 24.70 P	
	R.M. W.I.	kWh/t	****** Use B.M.	
Crushing Design	T.P.H.	Tonnes/Hr	601	c
R.O.M.	Size d100	mm	1,000	
Truck	Size	Tonnes	85	
R.O.M.	Bin Capacity	Tonnes	170	c
C.O.S.	Capacity L	Hours	24	
		Tonnes	14,423	c
Design	Project d80 Size	mm	200	
F.O.S.	Capacity L	Hours	48	
		Tonnes	17,820	c
Grinding Circuit Design	T.P.H.	Tonnes/Hr	380	c
	Feed d80 Size	mm	200	
	Product d80 Size	mm	0.053	
Recirculating Load			UF/OF	
	SAG Mill	%	200	
	Ball Mill	%	150	
Classification SAG Mill	Product Density	%Sol. WT.	45	
	Pulp S.G.	t/m^3	1.40	
		%Sol.Vol	23	
	T.P.H. Pulp	Tonnes/Hr	844	
	Vol. Flow	m^3/Hr	604	
Ball Mill	Product Density	%Sol.WT.	42	
	Pulp S.G.	t/m^3	1.36	c
		%Sol.Vol.		

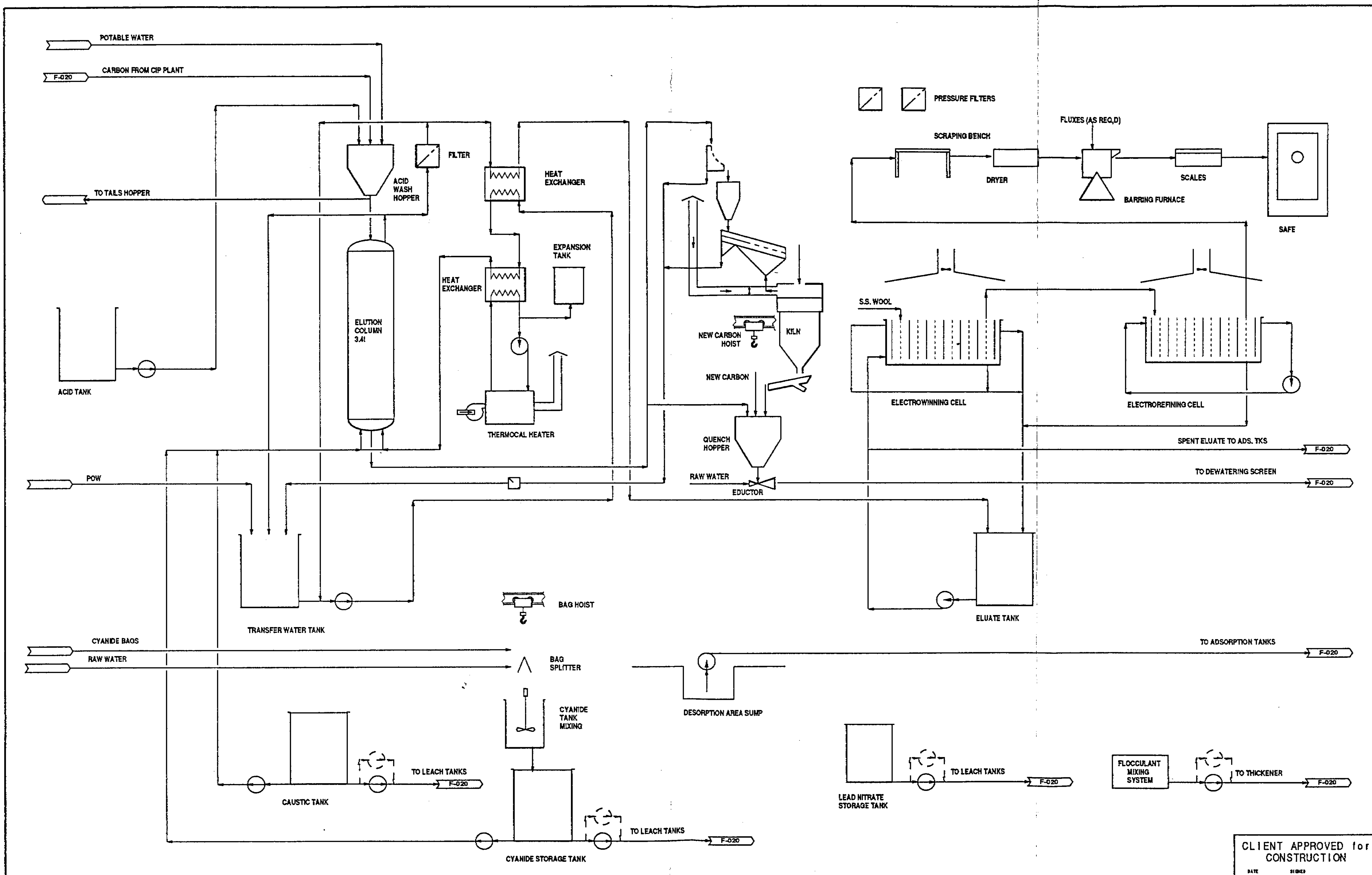
CATEGORY		UNITS	DATA	
Ball Mill	Product			
	T.P.H. Pulp	Tonnes/Hr	905	c
	Vol. Flow	m^3/Hr	665	c
Leaching/Adsorption				
Design	T.P.H.	Tonnes/Hr	380	c
	Density	%Sol. WT.	42	
Solids	Flowrate	m^3/Hr	665	c
Solut.n	Flowrate	m^3/Hr	525	
Sol.n	Feed Grade	g/t	0.92	
Trash Screen				
	Aperture	Microns	1,000	
C.I.P.	Hybrid			
Leach Time	Total	Hr	21.50	
	Adsorb	Hr	5	From Model
	Total	Hr	26.50	c
No. of Stages	Total		11	
	Leach		6	
	Adsorb		5	
Tank Vol.	Total	m^3	17,617 (Live)	c
	Leach	m^3	14,293	c
	Adsorb	m^3	3,324	c
Intertank Screen			Airswept, Flooded, W'Wire	
	Aperture	Microns	833	
Leach Recovery				
	Au	%	85 check	
	Ag	%	40 check	
Carbon	Type		PICA G210A5 (AMDEL)	
	Size	ASTM	6 x 12	
Design Tail Grade	Au	g/t	0.23	
Design Sol.n Tail	Au	g/t	0.01	
Design Carbon				
	Au Load	g/t	2,250	c
	Ag Load	g/t	******	
	Stripped			
	Au Stripped	g/t	50	
	Ag Stripped	g/t	******	
	Concentration	kg/m^3	13.84	c
	Massflow	kg/Hr	210.44	c
Total	Inventory	kg	46,000 Adsorb	From Model
Stage Efficiency		%	61	From Model
Gold Inventory		kg	42	From Model

CATEGORY			UNITS	DATA	
Res. Time per Stage			Hr	43.72	c
	Transfer			Cont.s Airlift	
	Removal			Recessed Imp. Pump	
Recovery Screen					
	Aperture		Microns	800	
Thickening					
Design	T.P.H.		Tonnes/Hr	380	c
	F. Density		%Sol. WT.	39	
	Pulp	S.G.	t/m^3	1.33	c
			%Sol.Vol	19	c
	T.P.H.	Pulp	Tonnes/Hr	974	c
	Vol.	Flow	m^3/Hr	734	c
Av. Settling Rate					
	Compression		m/Hr		
	Bulk		m/Hr		
	Sol.	Upflo	m/Hr		
	Max	%Sol.	%Sol. WT.		
Design	%Sol.		%Sol. WT.		
	Flocc	use	g/t	50 max	35 min
Desorption/Electrowin/Refine					
Stripping System				AARL, Cold Acid Wash	
Carbon Batch Size			Tonne C	2.79	c
No. of	Strips	P.W.		6	
	Strips	P.D.		2	
Metal Stripped		P.W.			
	Au		Kg	73.56	c
	Ag		Kg	46.15	
	Total		kg	120	c
Electrowin					
	No. Cells			2	
	No. Caths			9	
	Current		AMPS	700	
Barren	Eluate		PPM	10 max	
Electrorefine					
	No. Cells			1	
	No. Caths			10	
	Current		AMPS	500	
Carbon Regen.		Rage	Kg/Hr	350	
Reagent Usage					
C.I.P.					
	NaCN		kg/t	0.80	
	Lime		kg/t	1.10	
	NaOH		kg/t	0.20	
	Pb (NO)3		kg/t	0.40 max	

CATEGORY		UNITS			DATA
	Flocc	kg/t	0.05		
	Carbon	t/yr	46.00		
	G. Media	kg/t	1.50		
Desorb					
	NaCN	kg/t C	25.00	2% W/V	
	NaOH	kg/t C	20.00	2% W/V	
	HCl	kg/t C	85.00	3% W/V	
Water Balance					
Grind Product					
	T.P.H.	Tonnes/Hr	380		c
	Density	%Sol. WT.	42		
	Pulp S.G.	t/m^3	1.36		c
		%Sol. Vol	21		c
	Pulp T.P.H.	Tonnes/Hr	905		c
	Vol. Flow	m^3/Hr	665		c
	Water T.P.H.	Tonnes/Hr	525		
Tail					
	Density	%Sol. WT.	39		
	Water	Tonnes/Hr	594		c
Thickener U/F					
	Density	%Sol. WT.	60		
	Water Loss	Tonnes/Hr	253		c
	Reclaim	Tonnes/Hr	38	15 %Rec F	
Total Water Loss		Tonnes/Hr	215		
	Return (To Grinding)	Tonnes/Hr	379		c
Grind Make-up		Tonnes/Hr	146		c
Desorb Use		Tonnes/Hr	70		c
Other		Tonnes/Hr	2	check	
	Total	Tonnes/Hr	217		c
		TPD	5,216		c
		TPW	36,512		c
		TPA	1,898,624		c
Tailings Dam					
	Tonnes		14,100,000		c
Mine Life		Years	4.70		
	Density	%Sol. WT>	80	check	
	S.G. Tail	t/m^3	2.02		c
Dam Volume		m^3	8,720667		c

4.4 FLOWSHEETS


3t MAINT HOIST
MAGNET
CRUSHER
WEIGHTOMETER
STOCKPILE
VIBRATING FEEDERS
LIME SILO 38 t
WEIGHTOMETER
250
2.5
CYCLONES
500
203.8
750
509.4
P.W.
SAG MILL
301.1
STD. HOIST
2t BALL KIBBLE
SPLITTER BOX
125.0
152.8
312.6
287.8
125.0
152.8
CYCLONES
187.5
115.1
340.9
287.8
P.W.
12.0
BALL MILL
CYCLONES
187.5
115.1
P.W.
12.0
BALL MILL
PW 20.0
TRASH SCREEN
TRASH TO DISPOSAL
250.0
345.2
125.0
172.6
125.0
172.6
GRINDING AREA SUMP & PUMP
CARBON RECOVERY SCREEN AND TROMMEL
RAW WATER 2.0
BARREN ELUATE
F-021
CYANIDE
LEAD NITRATE
CAUSTIC
F-021
CARBON AIRLIFT (4 OFF)
DEWATERING SCREEN
REGENERATED/FRESH CARBON
F-021
HP AIR
4 x 2350 CU M LIVE CAPACITY LEACH TANKS
5 x 440 CU M LIVE CAPACITY CARBON ADSORPTION TANKS
PW
FINE CARBON BACK-UP SCREENS
FINE CARBON TO PIT
LOADED CARBON
F-021
SUMP PUMP
ACID RINSE
250.0
375.0
FLOCCULANT
F-021
O/F
208.3
THICKENER
U/F
P.W.
TO DOMESTIC & DESORPTION
TAILINGS RECLAIM
250.0
166.7
TAILINGS DAM
RAW WATER STORAGE DAM
WATER TREATMENT
POTABLE WATER
RAW WATER 400 CU. METRE
1200 CU. METRE
1200 CU. METRE
PW
PROCESS WATER
TPH SOLIDS
TPH WATER
CLIENT APPROVED for CONSTRUCTION
DATE
SIGNED
COPYRIGHT ©
MINPROC ENGINEERS PTY. LTD.
57 ST. PAULS TERRACE, BRISBANE, QUEENSLAND 4000
TELEPHONE : (07) 839 0363
REFERENCE DRAWINGS
No.
BY
DATE
REVISION
CHECKED
APPROV.
PROJECT APPR.
DESIGN APPR.
DESIGNED
CHECKED
DRAWN
I.NEVIN
PLANT GRID NORTH
CLIENT
BILLITON AUSTRALIA GOLD PTY. LTD.
TITLE
MT TODD GOLD PROJECT
2.0 MTPA CRUSHING, GRINDING,
LEACH & ADSORPTION PROCESS FLOWSHEET
SIZE
SCALE
B1
REV. No
DRAWING No.
MTG-00-F-020



CLIENT APPROVED for CONSTRUCTION

DATE | SIGNED

DRAWING No	REFERENCE DRAWINGS

MINPROC ENGINEERS PTY. LTD.

67 ST. PAULS TERRACE, BRISBANE, QUEENSLAND 4000
TELEPHONE : (07) 839 0383

No.	BY	DATE	REVISION	CHECKED	APPROV.	No.	BY	DATE	REVISION	CHECKED	APPROV.

	BY	DATE
PROJECT APPR.		
DESIGN APPR.		
DESIGNED		
CHECKED		
DRAWN	LNEVIN	1/89

PLANT GRID NORTH

CLIENT: BILLITON AUSTRALIA GOLD PTY.LTD.

SIZE: B1 | SCALE

TITLE: MT.TODD GOLD PROJECT
2MTPA DESORPTION GOLD ROOM
PROCESS FLOWSHEET

REV. No

DRAWING No. MTG-00-F-021

4.5 PLANT DESCRIPTION

4.5.1 Introduction

The Mt. Todd Gold Project C.I.P. treatment plant is designed to receive run of mine primary ore and process the ore to produce a Gold/Silver bullion product. Treatment of the ore will involve crushing, grinding, leaching and adsorption utilising carbon-in-pulp technology followed by recovery of gold using the Anglo American Research Laboratory (AARL) stripping method.

This section describes in brief the plant operation.

Section 4.3 sets down the design parameters for the process plant to achieve the production rate of 2.0 million tonnes per annum. These design criteria are based upon the results of a metallurgical test programme, in-house Minproc data where no testwork was available and, where appropriate from the performance of existing plants. A review of the metallurgical testwork programme is given in Section 4.1.

Principal design considerations are:

- Circuit arrangement is based upon single stage jaw crushing, closed circuit semi autogenous milling, parallel closed circuit ball milling, hybrid C.I.L. leach and adsorption and AARL stripping of loaded carbon.
- Gravity flow is used wherever possible.
- Common equipment sizes are used wherever possible to limit spare parts inventories and for ease of maintenance.
- All plant and equipment is designed to a standard commensurate with economic and reliable operation for an absolute minimum mine life of ten years.

4.5.2 Layout

The Mt Todd Gold Project C.I.P. plant has been located on a gently sloping area of land adjacent to the projected pit exit. Selection of this site will enable a minimum of earthworks. Workshops, administration offices, laboratory and store are in the same area. The 1:1000 contour maps did not cover the area closest to the pit exit. Relocation of the process plant to maximise use of natural landfall should be considered at a later stage of design.

4.5.3 Crushing and Fine Ore Storage

The single stage crushing plant will receive run of mine ore and crush it to a P_{80} of 200 mm product at an average rate of 400 tph. Feed to the plant enters by direct dumping of trucks or by front end loader using ore recovered from the coarse ore stockpile. The crusher is a 1.2 x 1.0 m jaw.

The crushed fine ore will be discharged to a 12,000 tonne open conical stockpile which is sufficient to keep the process running for 48 hours.

4.5.4 Fine Ore Recovery, Grinding, Cyclone Classification

Crushed ore from the fine ore stockpile is recovered by three vibratory feeders feeding onto the SAG mill feed conveyor. This conveyor will operate with a variable speed drive and weightometer linked to allow a constant feed to be maintained to the milling circuit. Feed rate is 250 TPH, reducing a feed d80 of 200 mm to a product d80 of 2 mm. The SAG mill is variable speed and grate discharge.

Lime, for pH control is added to the mill feed conveyor via a variable speed screw conveyor from a bulk storage bin. A furrow and plough is provided which turns the lime into the feed ore. The lime feeder is interlocked to prevent addition unless the belt is running and is loaded with ore. Grinding media (balls) are added direct to the mill feed chute via an automatic ball charger in both circuits.

The SAG mill is operated in closed circuit with two 760 mm hydrocyclones. (One standby unit)* Cyclone underflow is returned to SAG mill feed, and overflow is split equally by an adjustable weir box to the parallel ball mill circuit discharge hoppers. Each of the two parallel ball mill - cyclone grinding circuits is designed to reduce P_{80} 3 mm fine ore to eighty percent passing 53 micron at a feed rate of 125 tonnes per hour giving a total feed rate of 250 tph. The circuit consists of two 7.6 m x 5.3 m ϕ Ball Mills operating in parallel and closed with hydrocyclones (8 x 254 mm diameter).

Mill product discharges through a rotating scats trommel into the ball mill discharge pump hopper. The large steeply sloped hopper has twin outlet nozzles to the discharge pump and standby unit. The pumps discharge through a Tech-Taylor valve to a common mining hose feeding the cyclone feed pressure distributor. Each of the cyclones is valved to allow independent isolation. Cyclone underflow returns to the mill feed chute while the cyclone overflow discharges to a two stage trash screen.

The trash screen separates any trash such as fibre, blasting wire and plastic from the cyclone overflow. Both the cyclone feed pump and standby unit have variable speed drives controlled automatically to maintain a constant mill discharge hopper level. Mill water additions and pulp densities are maintained manually.

The ball mills and all associated equipment are located over a bunded slab containing a sump and sump pump. Floor slopes are graded to ensure gravity flow of spillage to the floor sump.

4.5.5 Leaching and Adsorption

The leach and adsorption circuit is designed to provide a total residence time of 21.5 hrs at the design feed rate of 250 tonnes per hour and a feed density of 42 percent solids. Experience has indicated that with the addition of lead nitrate the leach residence time, together with that provided by the adsorption circuit is adequate to ensure efficient leaching. The adsorption circuit is designed to recover Gold from solution onto carbon and to bring Gold in solution levels to acceptable values, before pumping to the tailing thickener and dam.

The leaching and adsorption circuit comprises nine agitated tanks arranged in two staggered rows. The four leach tanks are each 2350 m^3 live capacity, and the adsorption tanks 440 m^3 each. All tanks are fitted with identical open impeller agitation. These impellers are high efficiency axial flow type which have the impeller mounted well above the tank bottom. This arrangement safeguards the agitator should mechanical breakdown or loss of power allow settling of the pulp over the agitator blades.

No pre-aeration is provided. (Testwork may indicate a need for this).

The first four tanks are dedicated to leaching of Gold and Silver and are sized to provide 21.5 hours total residence time. Cyanide, caustic soda and Lead nitrate are introduced into the first tank via metering pumps from ground level storage tanks.

Tanks 5-9 are dedicated to the adsorption of Gold and Silver onto carbon and are fitted with air swept cylindrical wedge wire screens in their individual outlets to retain the carbon population required in each stage.

Carbon is air lifted counter current to the pulp stream using manually controlled continuous air lifts from tank 9 through the stages to tank 5. A submerged, recessed impeller pump operating in a batch mode, is employed to transport carbon via a static D.S.M. screen and rotating trommel screen to the acid wash hopper. Spray water is added to the screen installation to remove pulp and ensure only clean carbon enters the desorption facility. Screen underflow gravitates back to tank 5.

The tanks are interconnected by a pipework system fitted with removable isolation valves that allow a tank to be bypassed without interrupting the process flow.

An adjustable weir at the tank system outfall allows pulp level in the whole tank group to be adjusted to allow an increase in freeboard. Outflow from the leach/adsorption train gravitates by launder to the carbon recovery system where any escaping carbon is recovered before the slurry is discharged into the thickener feed launders. Carbon discharged from the screen is recovered in a portable container for storage until subsequent treatment. Screen underflow gravity feeds to the thickener. A screen bypass system is provided.

4.5.6 Thickener

A 15 m diameter Hi-rate thickener is used to increase tailings pulp density to a nominal 60% solids by weight to maximise water recovery.

Underflow from the thickener is pumped via an 6 x 4 Warman pump to the tailings dam. A second pump is provided for standby. Both sets are equipped with variable frequency drives and are controlled by a coriolis unit measuring pulp flow and density. A second stage pump is installed on the bypass line due to higher flowrates.

Flocculant is added proportional to thickener feed flow rate by automatic control, and thickener controls are overridden according to pulp/water interface level. Rake torque is monitored and automatic "rake-raise" protection at critical torque levels is provided.

A thickener bypass system is provided to allow discrete thickener maintenance and utilises a steep walled hopper for surge capacity in the feed to the tailings pumps The bypass system uses the same thickener underflow pumps automatically controlled on hopper level.

Thickener overflow (clear water) gravitates to the process water tanks.

A bunded slab graded to a sump and pump installation gives adequate clean up facilities under the thickener underflow and underflow pump area.

4.5.7 Tailings

Tailings is pumped to the tailings dam via a HDPE pipe line, and after settling, the decanted water is returned to the process water tank. A standby pump is provided.

4.5.8 Desorption and Gold Room

Clean loaded carbon is treated in the 3.40 tonne capacity Anglo American Research Laboratory stripping circuit. A total of 3.40 tonnes of carbon is stripped on each of 7 days per week.

The circuit employs a simple acid wash hopper for acid treating carbon to remove scale build up. Batch rinsing follows the acid soak and the carbon then gravitates into the AARL stripping column.

Following a caustic cyanide presoak, potable elution water, heated to 120° C is pumped through the column. This liquor returns via the heat recovery section of a heat exchanger to an eluate storage tank. The process continues until approximately six bed volumes have been pumped through the column.

Following the elution cycle, the pregnant eluate, at approximately 65 degrees centigrade, is cycled through an electrowinning cell where Gold and Silver are won on stainless steel wool cathodes. Spent eluate typically contains less than 10 ppm Gold.

Loaded cathodes from electrowinning are transferred to an electrorefining cell as anodes, and Gold and Silver are won onto stainless steel plates as a foil.

Loaded cathodes from the electrorefining cells are then removed and scraped. The foil is dried, fluxes added and then smelted in the barring furnace.

Regeneration of stripped carbon is carried out in a gas fired vertical kiln at 650 degrees centigrade. Reactivated or fresh carbon is educted back to the adsorption circuit from the quench hopper where it is drained over a DSM screen and metered to the adsorption tank.

4.5.9 Water Usage and Treatment

From the raw water dam water is pumped to 2 x 1200 m^3 process water tanks for make-up, via the 400 m^3 raw water surge tank.

Raw water pumps are sized for peak demand and standby units are provided at both installations.

Raw water is distributed to the desorption area for carbon transfer and screening as well as to safety showers, reagent mixing, and wash down hoses. A water softener is provided exclusively for elution water, to minimise scale-up of heat exchangers, acid washing, and for potable use.

SECTION 4.6

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

CRUSHING

Quantity	Description	Power (kW)
1	R.O.M. Bin	
1	Primary Crusher Grizzley Feeder 1.2 m x 5.0 m	22 kW
1	Primary Crusher 48 x 42 Double Toggle Jaw	130 kW
1	Primary Crusher Maintenance Hoist 3 t Electric Traveling	2 kW
1	Primary Crusher Dust Collector Dalamatic Type DLM	3 kW
1	Primary Crusher D/C Conveyor 1000 mm wide. 125 m Long	30 kW
1	Tramp Metal Magnet	-
1	Weightometer	

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

GRINDING

Quantity	Description	Power (kW)
3 (2+1S/B)	Crushed Ore Stockpile Reclaim Feeders Syntron FV-370	3.6
1	Reclaimed Ore Conveyor 750 mm wide x 80 m long	15
1	Weightometer	
1	Reclaim Tunnel	
1	Lime Silo c/w Feeder (38t capacity)	2
1	SAG Mill Ball Charger	
1	SAG Mill Feed Chute	
1	SAG Mill 5.3 m x 7.4 m	2200
1	SAG Mill D/C Hopper	
2 (1+S/B)	SAG Mill D/C Pumps 8.6 E-AH	110
1	Primary Cyclone Cluster 3 x 750 mm diameter	

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

GRINDING

Quantity	Description	Power (kW)
2	Ball Mills 5.3 m x 7.6 m x 4000 kW	8000 (2 x 4000)
2	Ball Mill D/C Hoppers	
4 (2+2S/B)	Ball Mill D/C Pumps 8/6EE-AH	264 (2 x 132)
2	Ball Mill Feed Chutes	
2	Secondary Cyclone Clusters 8 x 250 mm diameter	
1	Flow Splitter Box	
1	Grinding Area Sump Pump	11
1	Ball Charging Hoist 3t Travelling Electric	3
2	Ball Kibbles	
	TOTAL	10538.8

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

ADSORPTION

Quantity	Description		Power (kW)
2	Trash Screen DSM Type		
4	Leach Tanks 3250 m^3 vol		
4	Leach Tank Agitators 783Q125		360 (4 x 90)
5	Adsorption Tanks	440 m^3 vol	
5	Adsorption Tank Agitators 77Q30		110 (5 x 22)
5	Intertank Carbon Screen Assemblies		
4	Air Lift Assemblies		
1	Fine Carbon Backup Screen		
1	L.P. Air Blower		15

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

ADSORPTION

Quantity	Description		Power (kW)
1	Tails Thickener, 15 m Dia. H-R		7.5
1	Thickener Floc System		
3(1+1+1S/B)	Thickener U/F Pumps 6/4 EE-AH		300
1	Thickener by-pass Hopper		
2(1+S/B)	Tails Water Return Pumps (Submersible)		10
2	Process Water Tanks	1200 kl	
1	Raw Water Tanks	400 kl	
1	Water Treatment Plant Ion Exchange Type		2.2
2(1+S/B)	Raw Water Dam Pumps (Submersible)		25
2(1+S/B)	Process Water Pumps		90
1	Screen Handling Hoist	1 t	

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

ADSORPTION

Quantity	Description	Power (kW)
2(1+S/B)	Raw Water Pumps	90
1	Carbon Recovery Pump	7.5
1	Carbon Recovery Screen/Trommel	2.2
1	Adsorption Area Sump Pump	7.5
2	Return Carbon Dewatering Screens DSM Type	

MT. TODD HEAP LEACH GOLD PROJECT

EQUIPMENT LIST

DESORPTION PLANT

Qty.	Description	Power kW
1	Acid Wash Hopper	-
1	Elution Column (3.4 t)	-
2	Heat Exchangers	-
1	Thermocal Heater	.55
1	Kiln Feed Hopper Screen (D.S.M. Type)	-
1	Carbon Regen. Kiln c/w Predry Hopper & Discharge Feeder	.55
1	Carbon Quench Hopper	6
1	Carbon Eductor	-
1	Electrowinning Cell	-
1	Electrowinning Cell Hood/Fan	.55
1	Electrowinning Cell Rectifier	-
1	Electrorefining Cell	
1	Electrorefining Cell Hood/Fan	
1	Electrorefining Cell Rectifier	
1	Dryer	3
1	Barring Furnace	-
1	Electronic Balance	-
1	Safe	-
3	Eyewash/Shower	-
1	New Carbon Handling Hoist	2.5
1	Set Miscellaneous Goldroom Tools	-
1	Electrorefining Cell Circulation Pump	1.1
1	Inline Basket Filters (Eluate)	-

EQUIPMENT LIST (Cont)

DESORPTION PLANT

Qty.	Description	Power kW
1	Potable Water Tank	
1	Transfer Water Tank 15001	-
1	Transfer Water Pump	2.2
1	Caustic Storage Tank 120001	
2	Caustic Dosing Pumps	3.7
1	Eluate Tank	-
1	Eluate Pump	1.1
1	Cyanide Storage Tank	-
1	Cyanide Mixing Tank	-
1	Bag Splitter	-
1	Cyanide Mixing Tank Agitator	1.1
1	Elution Cyanide Metering Pump	.18
2(1+S/B)	Cyanide Dosing Pump	.37
1	Thermocal Expansion tank	
1	Desorb Area Sump Pump	7.5
1	Hot Oil Pump	37
2(1+S/B)	Potable Water Pumps	55
1	Lead Nitrate Storage Tank	
2(1+S/B)	Lead Nitrate Dosing Pumps	.18

EQUIPMENT LIST (Cont)

DESORPTION PLANT

Qty.	Description	Power kW
1	Cyanide Bag Hoist	3
1	Acid Storage Tank	
1	Acid Dosing Pump	.18
1	Elution Caustic Metering Pump	.18

MT TODD C.I.P. GOLD PROJECT

EQUIPMENT LIST

SERVICES

Quantity	Description	Power (kW)
1	H.P. Air Compressor	30
1	Instrument Air Dryer	2.2
5	Bore Pumps	25 (5 x 5)
1	L.P.G. Storage Facility	
	TOTAL	1206.71
	Plant Total	11745.51 kW

4.7 MANNING

The manning requirements have been assessed on the basis of experience and industry practice. A leading hand has been incorporated due to the significant number of off site elements in the operation - i.e. tailings dam, raw water dam, bores. The position would ideally be filled by a boilermaker to cope with minor maintenance, however the local union climate will determine the feasibility of this approach.

The bulk of the manning structure is as set out for the heap leaching study. An extra mining engineer, geologist and metallurgist have been included, plus the maintenance foreman position has been converted to an engineering role.

FIG 4.7

MT. TODD GOLD PROJECT

MANNING CHART

Total Strength - 46

- Mine Manager
  - Mine Superintendent
    - Mining Engineer
    - Mine Surveyor
    - Chainman
  - Typist/Secretary
  - Mine Geologist
    - Geologist
    - Grade Control Officerx2
    - Samplersx2
  - Administration Officer
    - Store Clerk
    - Computer Operator
    - Accounts Clerk
    - Typist/Secretary
  - Metallurgy Superintendent
    - General Foreman
      - Crushing 3 x 1
      - Grinding & CIP 3 x 2
      - Leading Hand 3 x 1
      - Day Crew x 2
    - Maintenance Engineer
      - Fitters x2
      - Boilermakers x2
      - T/A x2
      - Elect/Instrumt x2
      - (Tradesmen on Contract)
    - Chemist
    - Plant Metallurgist
      - Metallurgist
      - Gold Room Foreman

4.8 REAGENT MIXING

A schedule is illustrated in Table 4.8, showing for all metallurgical consumables their estimated consumptions and handling methods.

This data has been determined by testwork and from experience of similar processes. The costs are tabulated in Section 7, although lead nitrate was not included in the total cost.

Most reagent hold/mixing/dosing equipment costs are incorporated in the desorption area. The exceptions are flocculant (in thickener cost) and grinding media (in grinding area).

TABLE 4.8

Reagent Schedule - 2.0 MTPA

A summary of the reagents and consumable used for a 2.0 MTPA ore treatment rate is presented.

REAGENT	PRIMARY USE	PHYSICAL FORM	DELIVERY	EST. MONTHLY CONSUMPTION (t)	STORAGE FACILITY CAPACITY TONNES	STORAGE FACILITY STORAGE TYPE
Sodium Cyanide (NaCN)	Gold & Silver leaching	Solid briquettes (99% NaCN)	Boxed 1 tonne bulker bags on pallets	133	300	Boxes in fenced compound
Lead Nitrate ($PbNO_3$)	Gold & Silver leaching	30% solution (w/w) (1.35 t/m^3)	22 tonne tanker truckloads	50	28	Fibreglass tank vented to atmosphere
Hydrochloric Acid (HCl)	Carbon Washing	35% solution (w/w) (1.55 t/m^3)	22 tonne tanker truckloads	9	11	Fibreglass tank vented to atmosphere
Caustic Soda (NaOH)	Carbon stripping & pH adjustment	50% solution (w/w) (1.55 t/m^3)	22 tonne tanker truckloads	52	30	Mild Steel tank vented to atmosphere
Quick Lime (CaO)	Ph adjustment	Powder	22 tonne truck pneumatic transfer	250	38	Mild steel silo with bag filter vent
Flocculant	Thickener settling	Solution	200l drum	9	20	Drums stored in fenced compound
Borax ($Na_2B_4O_7$)	Smelting Flux	Powder	40 kg Bags	0.16	2	In store building
Niter ($NaNO_3$)	Smelting Flux	Powder	22.5 kg Bags	0.22	2	In store building
Silica Flour (S_1O_2)	Smelting Flux	Powder	22.5 kg Bags	0.22	2	In store building
Grinding Media	Grinding	Steel balls	Approx. 1 tonne drum in 22 tonne loads	250	600	Stacked drums
Carbon	Adsorption	Granular coconut Shell	100 kg drums in containers	3	20	Remain in containers until required

5.0 REVIEW OF PROCESS OPTIONS

5.1 COMMINUTION

The grinding circuit is designed on the basis of 92.5% availability. An overlap is incorporated between the two grinding lines with respect to size - a P_{80} for the SAG mill of 2000 microns (product) and a feed F_{80} for the ball mill of 3000 microns.

Primary grinding classification for the purposes of this study is accomplished by hydrocyclone. A screen option should be examined in detail at a later stage (and selection of grate gap dimensions). The primary (SAG) mill is variable speed to accommodate the variability in ore types (12 - 25 WI) and permit control over power usage.

The SAG mill product size was selected to maximise ball mill efficiency and was scaled up on this basis. Initially the SAG mill was sized to serve in open circuit. This introduces a 20% inefficiency in power, so a cyclone classifier was incorporated as mentioned above. The circuit balance was selected to ensure a cyclone underflow density suitable for SAG mill feed.

Ball mills were selected on the basis of a WI of 25, and are sized to achieve a 53 micron product d80. This size may be a little conservative. Classifiers and pumps were sized on the basis of a 150% recirculating load (250% of feed rate in cyclone feed), and a d80 equivalent to the product p_{80} specification. The ball mills are fed from a two way splitting device which is adjustable to balance flow between the two circuits. This is more practical than attempting to use a common sump and/or cyclone cluster at this stage.

A mass balance and cyclone selection parameters are shown for each circuit in Tables 5.1 and 5.2.

This approach is significantly cheaper and simpler than a crushing/rod milling circuit (and more efficient). If a combined heap leaching/CIP process is considered in the future, there is potential for dewatering SAG mill product, and either stockpiling or agglomerating. This could be achieved by DELKOR linear screening or a small aperture vibrating screen, and carried out on a campaign basis when ore of 0.6-1.0 g/t was available.

The three million TPA case would require two 5.30 x 5.2 SAG mills feeding three 5.30 x 7.6 ball mills. Table 5.3 shows the revised SAG mill mass balance.

No evidence is available to suggest that a gravity circuit in the ball mill recirculating load would be feasible. It is recommended that recess liners be used in the mills, and that material collected there be assayed for Gold.

The Mt. Todd ore has a low crushing impact work index relative to the ball mill work index. This creates uncertainty with regard to SAG mill behaviour and the potential for critical size buildup. The effect can be summarised by envisaging good breakage to a size range from 20 - 60 mm, with only attrition grinding beyond this point. This could lead to large recirculating loads and a much finer product, which in turn would make poor use of the ball mill capacity.

A rod mill/ball mill circuit should be considered for the feasibility study as an alternative approach.

TABLE 5.1

ENTER	SG SOLIDS	2.71		%RL =	200.00		
	COF %SOL	45.00		TPH	250.00		
	% MOIST	1.00		CUF WATER SPLIT	0.40	Min 0.2	

SAG MILL MT TODD GOLD PROJECT

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	%Sol Vol
NEW FEED	250.00	2.53	99.00	2.66	252.53	94.78	97.34
WATER ADD		2.00	0.00	1.00	2.00	2.00	0.00
MILL FEED	750.00	208.28	78.27	1.98	958.28	485.03	57.06
MILL DISC	750.00	208.28	78.27	1.98	958.28	485.03	57.06
WATER ADD		301.10	0.00	1.00	301.10	301.10	0.00
SPILLAGE	0.06	0.00	100.00	2.71	0.06	0.02	100.00
CYC FEED	750.06	509.38	59.55	1.60	1259.44	786.16	35.21
CYC U/F	500.00	203.75	71.05	1.81	703.75	388.25	47.52
CYC O/F	250.06	305.63	45.00	1.40	555.69	397.90	23.19

C1	4.76			FEED kPa	60	
C2	1.04			ROPE LIM	61.80	%SOLv CUF
C3	0.98			VF D mm	937	
d50c DES	500	MICRONS		INLET Dmm	662	MAX
CYC DIAM	2342	mm MAX D			574	MIN
Q/CYC	3986.97	M3/HR				
NO. CYC.S	0.20					
APEX D	185	mm LOW	234 mm			
		HIGH	820 mm			

		mm	feed kPa 40	60	80	100	120	140	160	180	200
nominate lower sizes read no. of cyclones at feed kPa	1	1270	0.8	0.7	0.6	0.5	0.5	0.4	0.4	0.4	0.4
	2	1050	1.2	1.0	0.8	0.8	0.7	0.6	0.6	0.6	0.5
	3	900	1.6	1.3	1.2	1.0	0.9	0.9	0.8	0.8	0.7
	4	750	2.4	1.9	1.7	1.5	1.4	1.3	1.2	1.1	1.1
	5	660	3.0	2.5	2.1	1.9	1.8	1.6	1.5	1.4	1.4
	6	151	58.1	47.4	41.1	36.7	33.5	31.0	29.0	27.4	26.0

TABLE 5.2

ENTER	SG SOLIDS	2.71	%RL =	150.00	
	COF %SOL	42.00	TPH	125.00	
	% MOIST	55.00	CUF WATER SPLIT	0.40	Min 0.2

REVERSE BALL MILL CIRCUIT MT TODD GOLD PROJECT

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	%Sol Vol
NEW FEED	125.00	152.78	45.00	1.40	277.78	198.90	23.19
WATER ADD		0.00	0.00	1.00	0.00	0.00	0.00
MILL FEED	187.50	115.14	61.96	1.64	302.64	184.32	37.54
MILL DISC	187.50	115.14	61.96	1.64	302.64	184.32	37.54
WATER ADD		19.92	0.00	1.00	19.92	19.92	0.00
SPILLAGE	0.06	0.00	100.00	2.71	0.06	0.02	100.00
CYC FEED	312.56	287.84	52.06	1.49	600.40	403.17	28.61
CYC U/F	187.50	115.13	61.96	1.64	302.63	184.32	37.54
CYC O/F	125.06	172.70	42.00	1.36	297.76	218.85	21.09

C1	3.03		FEED kPa	180	
C2	0.76		ROPE LIM	60.67	%SOLv CUF
C3	0.98		VF D mm	98	
d50c DES	53	MICRONS	INLET Dmm	69	MAX
CYC DIAM	245	mm MAX D		60	MIN
Q/CYC	75.47	M3/HR			
NO. CYC.S	5.34				
APEX D	65	mm LOW	24 mm		
		HIGH	86 mm		

			feed kPa								
nominate lower sizes		mm	40	60	80	100	120	140	160	180	200
read no. of	1	760	1.2	1.0	0.8	0.7	0.7	0.6	0.6	0.6	0.5
cyclones at	2	660	1.6	1.3	1.1	1.0	0.9	0.8	0.8	0.7	0.7
feed kPa	3	508	2.6	2.1	1.9	1.7	1.5	1.4	1.3	1.2	1.2
	4	305	7.3	6.0	5.2	4.6	4.2	3.9	3.7	3.4	3.3
	5	254	10.5	8.6	7.4	6.7	6.1	5.6	5.3	5.0	4.7
	6	151	29.8	24.3	21.1	18.8	17.2	15.9	14.9	14.0	13.3

TABLE 5.3

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	%Sol Vol
NEW FEED	187.50	1.99	99.00	2.66	189.39	71.08	97.34
WATER ADD		2.00	0.00	1.00	2.00	2.00	0.00
MILL FEED	562.50	156.72	78.21	1.97	719.22	364.29	56.98
MILL DISC	562.50	156.72	78.21	1.97	719.22	364.29	56.98
WATER ADD		225.35	0.00	1.00	225.35	225.35	0.00
SPILLAGE	0.06	0.00	100.00	2.71	0.06	0.02	100.00
CYC FEED	562.56	382.07	59.55	1.60	944.63	589.65	35.20
CYC U/F	375.00	152.83	71.05	1.81	527.83	291.20	47.52
CYC O/F	187.56	229.24	45.00	1.40	416.80	298.45	23.19

ENTER

SG SOLIDS	2.71	%RL =	200.00	
COF %SOL	45.00	TPH	187.50	
% MOIST	1.00	CUF WATER SPLIT	0.40	Min 0.2

SAG MILL MT TODD GOLD PROJECT

C1	4.76		FEED kPa	60
C2	1.04		ROPE LIM	61.80 %SOLv CUF
C3	0.98		VF D mm	937
d50c DES	500 MICRONS		INLET Dmm	662 MAX
CYC DIAM	2342 mm MAX D			574 MIN
Q/CYC	3988.15 M3/HR			
NO. CYC.S	0.15			
APEX D	185 mm	LOW 234 mm		
		HIGH 820 mm		

feed kPa

nominate lower sizes		mm	40	60	80	100	120	140	160	180	200
read no. of	1	1270	0.6	0.5	0.4	0.4	0.4	0.3	0.3	0.3	0.3
cyclones at	2	1050	0.9	0.7	0.6	0.6	0.5	0.5	0.5	0.4	0.4
feed kPa	3	900	1.2	1.0	0.9	0.8	0.7	0.7	0.6	0.6	0.5
	4	750	1.8	1.4	1.2	1.1	1.0	0.9	0.9	0.8	0.8
	5	660	2.3	1.9	1.6	1.4	1.3	1.2	1.1	1.1	1.0
	6	151	43.6	35.6	30.8	27.6	25.2	23.3	21.8	20.5	19.5

5.2 C.I.P. PLANT

A. Leaching and Adsorption

Test data for the study from AMDEL was delayed, thus the design exercise proceeded on the following data:

- 24 hour leach duration
- 90% Gold recovery from solids
- 2,000,000 TPA at 1.5 g/t Gold
- 90% passing 75 micron grind.

Kinetic characteristics were approximated from previous experience, and design proceeded on the following assumptions:

- pulp density/viscosity effects are negligible
- oxygen supply was adequate by entrainment
- carbon performance is "typical"
- there are no carbon fouling problems
- a 15 m high rate thickener would be adequate
- reagent consumptions are "normal".

Without leaching and carbon adsorption profiles, a Carbon in Leach or hybrid C.I.L. circuit could not be designed with any confidence. A Carbon In Pulp circuit was used with reasonably conservative residence times. A 42% solids by weight slurry was adopted, as slurries of higher density are difficult to classify to the required size, and commonly reduce carbon kinetic activity.

Referring to the original AMDEL agitation leach testwork, a fairly rapid leach was attained. This implied that the design in this report may be refined to a C.I.L. circuit at the next stage of design, with potential for reduction of capital cost. Typically, an economic analysis requires comparison of the reduction in capital cost with the extra costs of carbon inventory (and Gold lockup).

Leach kinetics are often significantly enhanced with Lead nitrate addition , and often overall recovery increases as well. This is a double effect, in that Lead will scavenge Sulphur products from sulphide oxidation (which would both passivate and reprecipitate Gold, and consume cyanide by forming thio cyanates), and carbon activity is enhanced by the nitrate ion. This offers further potential for capital cost reduction.

Further testwork indicated that the 24 hour leach residence time is inadequate. A 48 hour leach should be checked in the next stage of testwork, plus the strategy of liquid oxygen injection or hydrogen peroxide addition.

Leaching was based on a 21.5 hour residence time requirement. This was a function of tank construction limitations and experience. Adsorption capacity was scaled up by use of a procedure which assumes a steady state process (with regard to head grade and slurry characteristics) and ideal plug flow. A minimum of 5 stages is utilised to avoid short circuiting, each with a residence time of 1 hour. Effects of carbon degradation have not been incorporated at this stage. Five stages were selected as this yields a stage efficiency of ~61% for a recovery of 90% Au from 1.5 g/t ore. A sixth stage was considered, however the extra capital cost is greater than the reduction in carbon inventory.

The screen, carbon transfer and flow control systems are based on well tried Minproc designs which have been built up over many plant design and performance cycles. Variants which would be examined more closely at a later stage of design would be screen selection for trash and carbon safety duties (DSM versus high frequency vibrating screens) and carbon transfer methods (air lift versus water eduction).

The impact of processing 3 million TPA would be felt in the cost of tankage, although some enconomy of scale may apply.

Leaching profiles were so varied that the initial decision to use C.I.P. may well stand. A C.I.L. process generally requires rapid initial leaching rates. Some of the tests exhibited a reprecipitation characteristic, however, which could weigh in favour of a hybrid C.I.L. circuit.

B. Desorption

The Anglo American Research Laboratory (AARL) stripping process has been selected to maintain maximum flexibility, and minimum labour requirement.

The AARL process is much faster and more automated then the Zadra process, and a new strip can start as electrowinning proceeds. Cooling of eluate is unnecessary and heating costs are lower.

In the Mt. Todd situation cost of pure water is negligible, and if 50% cyanide recovery is assumed then operating costs of Zadra and AARL systems are very similar. Capital cost for the two systems is also very similar.

Ultimately, the advantage of AARL for the Mt. Todd project is that shift crews can be kept to a minimum, and any significant influx of Silver will have less effect. If capacity expansion is required, even temporarily, an extra back shift can be worked.

An acid wash hopper system has been designed to fit over the elution vessel. This is cheaper than a second column and the associated carbon transfer equipment.

The electrowinning circuit operation is less critical for AARL since it operates in a closed loop with the eluate tank, and pass efficiencies need only be 55-75%. Current efficiency is higher and cell and cathode sizes are lower relative to a Zadra system. The flow sheet is illustrated on flowsheet 2.

5.3 WATER MANAGEMENT

Figure 5.1 sets out a conceptual water balance for the process. This represents a steady state situation, with the only water usage not accounted for being the internal recycle of trash screen and carbon safety screen sprays.

In the wet season a significant amount of water will be available from the tailings dam water recovery system, which may reduce the raw water requirement from 147 m^3/hr to ~16 m^3/hr (for desorption, reagent mixing and domestic use). In the dry season it is anticipated that only 15% recovery of tailings dam water will be achieved.

A fall from 42% solids in leach feed to 39% solids in leach tail has been allowed. This accounts for hose up, carbon recovery screen sprays, addition of reagents and desorption recycles into the C.I.P. process.

Raw water is pumped from the raw water dam to a 400 m^3 raw water tank on a demand basis. Pumps distribute water to the water softener for domestic and desorption use which feeds a small potable water tank. This tank pumps water to the process transfer water tank in the desorption area and into the domestic ring main. The Raw Water tank pumps also top up the process water tanks on a demand basis. These comprise 2 linked 1200 m^3 tanks, which will keep the operation on line for 24 hours should either the thickener be by-passed or the external raw water supply fail. The raw water tank is sized to run the reagent/desorb/domestic system for 24 hours should the external raw water supply fail.

Raw water dam pumps are sized to deliver 200 m^3/hour. Tailing reclaim pumps are the same capacity, but will only pump when a high level is achieved in the reclaim pond, and shut down at low level. Tailing thickener overflow water will gravitate to the Process Water tanks. The thickener selection was based on experience of other operations. Testwork data arrived after the bulk of the report had been written. Using a worst case of 0.7 m^2/tpd, a 12 m High Rate thickener is required whereas a 15 m unit cost was used.

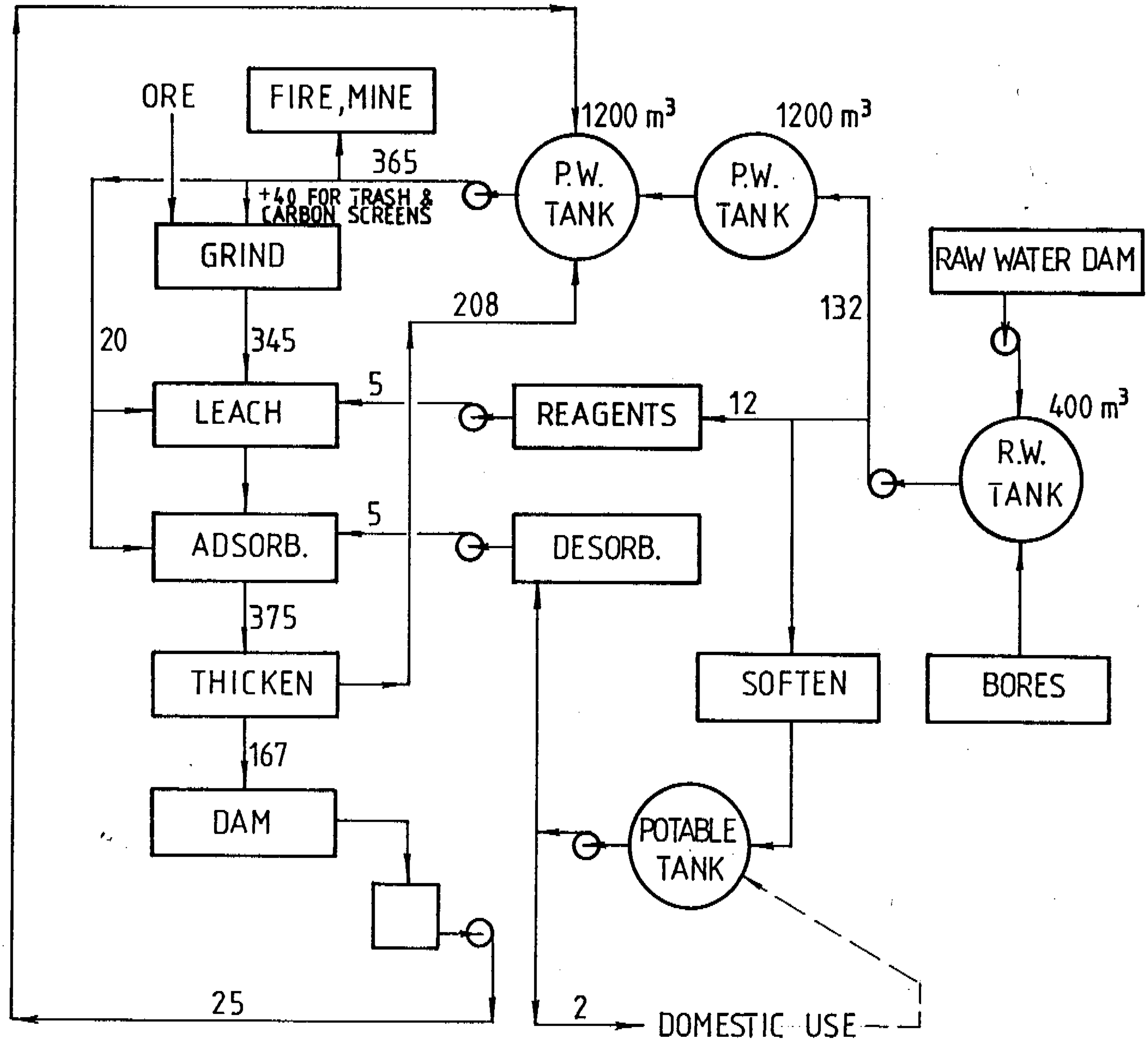
The raw water dam is anticipated to yield 600,000 m^3 per year of the 800,000 m^3 contained. The bore source should yield 400,000 m^3, leaving a deficit of 260,000 m^3. This capacity should easily cover the dry season requirement, and the wet season should quickly replenish both sources, particularly as tailings dam catchment will have to be used for process water top up due to the slight contamination involved.

At 3 million TPA, 50% more water is required. Either source is amenable to capacity expansion.

5.4 TAILINGS DISPOSAL

The site selected for the tailings dam system (shown on the site plan) is based on the following criteria:

1. The near surface geology is assumed to be similar to that at the raw water dam site, where Golder Associates test pit logs 1 - 5 indicate strong tight rock at shallow depths. Thus base permeability is expected to be very low in this area, minimising the likelihood of groundwater contamination.



MT. TODD GOLD PROJECT

C.I.P. PROCESS WATER BALANCE

FIG. 5·1

2. The topography provides adequate storage for 20 million tonnes of tailings with minimum earthworks construction for dam walls. Volumetric capacity is based on a final settled density of tailings of 1.50 (tonnes of solids per cubic metre) which should be readily achieved using controlled sub-aerial beaching, and by minimizing deposition and storage of tailings under water.

3. Construction is staged to provide storage initially for one year's production of tailings, with successive lifts to the various dams providing increased capacity for the ongoing production.

4. Storage must be provided for rainfall arising from wet season conditions; whenever possible in a location not over tailings. Wet season storage is based on the worst record season (December 1897 - March 1898), allowing run-off coefficients of 1-0 for rainfall on dam surfaces and 0.7 for rainfall off the dam. Diversion drainage is provided to minimize off dam catchments, but because of the steep nature of the topography it is not practicable to completely eliminate these. Evaporation losses from wet surfaces of tailings and retention ponds have been allowed for in sizing retention areas, and it has been assumed that contaminated water in retention ponds would be used preferentially for process water where possible to minimize the storage requirement.

5. Water recovery from tailings would be by fixed decant towers with recovery pipes laid under dam walls to decant panels.

6. Dams intended as water storages would be constructed with impervious core composed of compacted select fill including adequate fines placed under controlled moisture conditions. Dams intended to retain tailings only may be constructed from select mine waste rock to form a semi porous wall. This construction will provide for initial drainage of the tailings placed adjacent to the wall, but migration of tailings fines will seal the wall as deposition progresses.

5.5 FURTHER WORK

It is difficult to comment until the final AMDEL report is available. However, the following recommendations are based on current knowledge, and the anticipated requirements for the next study stage.

1. A repeat of the grind optimization should be carried out for leaching, with finer sizes included.

2. Lead Nitrate additions should be tested.

3. A "practical" mineralogist such as Wally Fander should look at the ore.

4. Tests should document/carry out:

   - dissolved oxygen, D.O. uptake.
   - residue sizings on all tests.
   - residue assays and head assays.
   - a series of reproducibility tests to 24 hours (to obtain residue grade).
   - regrind and releaching of ore residue.

5. In light of the large power draws involved, High Pressure rolls crushing testwork should be pursued, with the intent of reducing ball mill power use.

6. Bond ball mill work index tests should be carried out close to the chosen product size. Rod mill work index testwork should also be carried out.

7. A residue sample should be sent to both Larox for thickener testwork and Lightnin for checks on mixer selection.

A provisional program of further testwork is included in Appendix 4.

6.0 CAPITAL COST ESTIMATE

6.1 INTRODUCTION

Capital cost estimates have been prepared for the project and are summarised on Figure 6.1. All costs are in Australian Dollars as at November 1988.

Figure 6.1 shows the overall capital expenditure, with timing of expenditure shown on Figure 8.1 and associated financial analyses schedules.

All capital estimates prepared for this study are based on current mine and process plant development costs.

All units used are metric.

The estimates are regarded as having an accuracy of ± 25%.

6.2 PLANT AND SERVICES (Refer Figure 6.1 - sheet 1 of 3)

Capital costs for process plant and services have been estimated by undertaking sufficient preliminary design to establish the scope of work and utilising budget prices obtained from suppliers and actual costs from comparable projects. The estimate is summarised on Figure 6.1; it should be noted that the estimates include a contingency provision. A figure of $6.1 million is spread over 10 years to cover ongoing tailings dam expansion.

The scope of facilities covered by this estimate is set out in Section 2.0 and below. The interface points in the assumed scope of work and other aspects in clarification of the scope may be further stated as :

- Initial earthworks at the R.O.M. stockpile, but excluding the fill for the ramp and stockpile area,
- Connection of power from the plant area switchboard,
- Earthworks and roadworks are limited to the treatment plant area, pads and ponds, rain water dam and site roads within the lease area. It is assumed that mine waste is delivered to the required areas for construction purposes e.g. dams. No provision is made for mine haul roads which are anticipated to be constructed as part of mining operations.
- It is assumed that the mining operation is undertaken by a contractor who will provide appropriate support facilities e.g. workshop, store, pumping, lighting, fencing and magazine.
- Estimates are based on new equipment,
- Included in the estimate for the plant is the cost of engineering, project and construction management services associated with design and construction of the plant and initial site works.
- No provision is made in the plant estimate for "all risks" insurance during construction as it is considered more economical for the owner to incorporate such cover in an overall project insurance, and provision is made under the heading of owners costs,

. Further points of clarification regarding the scope of the capital estimate are shown below.

6.2.1 Earthworks and Civils

Preliminary estimates of earthworks quantities have been prepared as indicated.

6.2.2 Concrete

The estimates contain concrete works for each facility shown.

6.2.3 Structural Steel

The estimates are for supply and erection of structural steel for each facility.

6.2.4 Platework & Tanks

The estimates are for supply and erection of platework and tanks.

6.2.5 Equipment

The estimates are for supply of equipment as listed on Table 6.1.

6.2.6 Mechanical Installation

The cost of installation of mechanical equipment has been estimated using information from comparable projects.

6.2.7 Pipework and Services

Plant piping and services reticulation have been estimated.

6.2.8 Buildings

Provision is made for supply and erection of the buildings as listed in Section 3.5.

6.2.9 Electrical

Electrical services are provided for the process plant with power supply included under infrastructure.

6.2.10 Furniture and Equipment

Provision is made for supply and installation of laboratory equipment. Furniture and equipment for other buildings is included as shown.

6.2.11 Freight

A separate provision is made for freight of equipment and materials purchased ex works.

6.2.12 Fees

Provision is made for construction fees incurred by a contractor.

6.2.13 Contingency

A contingency of approximately 10% of plant direct costs is included.

6.2.14 Design, Project Management, Site Temporary Works and Project Expenses

Provision is made for the costs incurred by a contractor undertaking design and construction of the process plant and associated site works necessary to bring the plant to the stage of commissioning.

6.3 INFRASTRUCTURE (Refer Figure 6.1 - sheet 2 of 3)

6.3.1 Water Supply

Water supply is obtained from a bore system adjacent to the mine and plant area. The cost of this work is included under process plant.

6.3.2 Power Supply

The alternatives for power supply are reviewed in Section 3.0. The estimate provides for power from the state grid.

6.3.3 Access Road

The alternatives for road access to the site are reviewed in Section 3.0 and the cost estimates for Options 1 and 2 are summarised below.

Option 1 - Upgrade Existing Road

. Road upgrading 4.5km @ $20,000/km	90,000
. Edith River Crossing	358,000
. Minor Creek Crossings (4 off)	40,000
	$488,000

Option 2 - Use Alternative Crossing

. New Road 2.5km @ $25,000	62,500
. Road upgrading 1.5km @ $20,000	30,000
. Stow Creek Crossing	186,000
. Minor Creek Crossings (4 off)	40,000
	$318,500

6.3.4 Accommodation

As discussed in Section 3.0, single accommodation could be provided either on site or in Katherine. The estimates for these two alternatives are summarised below.

(a) Accommodation on Site

For accommodation on site using transportable construction camp type buildings.

13 off 4 man accommodation units	865,000
50 man laundry	32,000
50 man kitchen/dinner	100,000
50 man recreation building	55,000
Site preparation, civils	100,000
EPCM	70,000
	$1,222,000

(b) Accommodation in Katherine

For accommodation in concrete block, airconditioned units, 2 storey, single rooms with toilet/bathroom shared by two occupants, central messing and laundry.

Accommodation units (50 off)	1,085,672
Carports	181,021
Kitchen Dinner	155,349
Fitting out	365,700
Services and	
Land Purchase (zoned land)	150,000
EPCM	140,000
	$2,077,742

It is likely that room numbers will be reduced to 40 with the possibility of local employment or personnel making other arrangements for accommodation. The cost of accommodation is reduced to $1,662,194 for the purposes of this study.

(c) Staff Housing

Staff Housing is included for five senior staff. Enquiries indicate that existing housing is readily available for purchase. However, for the purposes of this study, new housing has been assumed.

6.4 MINE DEVELOPMENT (Refer Figure 6.1 - sheet 2 of 3)

6.4.1 Permanent Mining Facilities

No provision is made for mining facilities as contract mining is assumed.

6.4.2 Preproduction Development

As indicated in the mining study, preproduction development has been estimated as shown on Figure 6.1.

6.4.3 Mining Equipment

No provision is made for mining equipment as contract mining is assumed.

6.5 OWNERS' CAPITAL (Refer Figure 6.1 - sheet 3 of 3)

6.5.1 Light Vehicles

Allocation of light vehicles has been assumed as set out below.

	OTHER	4WD	SEDAN	4WD UTE
Mine Manager			1	
Mine Supt		1		
Surveyor		1		
Geologist		1		
Met. Supt		1		
Admin. Officer			1	
Maintenance				1
Plant Foreman	Bus x2			
Store	Forklift			1
First Aid	Ambulance			

Summary

Toyota Land Cruiser 4WD	4 x 40,000	160,000
Falcon Sedan	2 x 27,000	54,000
Toyota Utility 4WD	2 x 40,000	80,000
Bus/Toyota 4WD Troop Carrier	2 x 40,000	80,000
Ambulance	1 x 50,000	50,000
Rough terrain forklift (2 tonnes)	1 x 60,000	60,000
		$484,000

6.5.2 Recruitment and Training

An allowance equal to three months labour cost is made to cover recruitment and training costs.

6.5.3 Process Plant Maintenance and Replacement

In calculation of operation costs, provision has been made for maintenance of plant and facilities including purchase of spare parts and equipment replacement. The annual expenditure on this item has been included on the financial analysis schedules in Section 8.0.

6.5.4 Insurance During Construction

Provision for insurance during construction has been included in the amount of $ 70,000.

6.5.5 Facilities in Katherine

A sundry provision of $ 50,000 has been made for facilities in Katherine.

6.6 WORKING CAPITAL

6.6.1 Process Plant

A provision of $ 800,000 has been made for working capital for the process plant and site facilities.

6.6.2 Katherine

A provision of $ 50,000 is made for working capital for facilities in Katherine.

6.7 EXCLUSIONS

Excluded from the capital costs are the following :

- Land acquisition for mine lease area
- Owners Head Office Costs
- Mining Equipment
- Lease Equipment
- Environmental permits and associated costs.
- Legal Fees.
- Water rights and associated costs.
- Taxes and statutory fees except as stated herein.

6.8 CHANGES TO CAPITAL

Extra leaching capacity was costed and included as follows (43 hours):

1.	Platework	$ 1.177 million
2.	Steelwork	$ 0.176 million
3.	Concrete	$ 0.144 million
4.	Agitators	$ 0.440 million

These costs apply to 8 x 2350 m^3 tanks, whereas economics would be achieved using larger units.

The estimate of capital cost reduction by grinding to 75 microns (rather than 53 microns) is $ 1.2 million.

MT. TODD GOLD PROJECT
PRE-FEASIBILITY STUDY
CIP PROCESS PLANT

CAPITAL COST SUMMARY

FIGURE 6.1

(SHEET 1 OF 3)

1. PROCESS PLANT

(MTODES82) 05-Feb-89

	GENERAL	CRUSHING SCREENING STOCKPILE	GRINDING AND CLASSIF.N	LEACHING AND ADSORPTION	DESORPTION AND GOLD ROOM	PLANT SERVICES	WORKSHOP WAREHOUSE LABORATORY	OFFICE & AMENITIES	RAW WATER DAM	TAILINGS DAM	TOTAL
EARTHWORKS & CIVILS	0	481600	1146756	566955	42570	103200	0	0	631025	1728061	4,700,166
& CONCRETE	0	0	0	0	0	0	0	0	0	0	0
STRUCTURAL STEEL	0	367435	203175	478590	192962	0	0	0	4515	0	1,246,677
PLATEWORK & TANKS	0	199756	117530	3086471	112049	0	0	0	0	0	3,515,807
EQUIPMENT	0	688990	11127979	1824155	413763	60582	0	0	16243	17684	14,149,395
MECHANICAL INSTALLATION	0	54062	330084	76615	26975	231544	0	0	1075	1075	721,430
PIPEWORK & SERVICES	0	0	0	1417063	345825	197783	0	0	93364	38270	2,092,305
BUILDINGS	0	0	0	0	0	0	174150	215000	0	0	389,150
ELECTRICAL	0	231816	780302	844214	363850	132034	0	0	0	0	2,352,216
FURNITURE & EQUIPMENT	0	0	0	0	0	0	166000	80000	0	0	246,000
FREIGHT	300000	0	0	0	0	0	0	0	0	0	300,000
FEES	10000	0	0	0	0	0	0	0	0	0	10,000
SUB-TOTAL	310000	2023658	13705825	8294063	1497994	725142	340150	295000	746222	1785090	29,723,144
CONTINGENCY											4,153,200
DESIGN & PROJECT MANAGEMENT											4,069,800
SITE TEMPORARY WORKS & EXPENSES											250,000
TOTAL PROCESS PLANT COST	310000	2023658	13705825	8294063	1497994	725142	340150	295000	746222	1785090	38,196,144

MT. TODD GOLD PROJECT
PRE-FEASIBILITY STUDY
CIP PROCESS PLANT

CAPITAL COST SUMMARY

FIGURE 6.1

2. INFRASTRUCTURE

(SHEET 2 OF 3)

MT.TODD	ESTIMATE	CONTINGENCY	%	
WATER SUPPLY		0	0%	0
POWER SUPPLY(GRID POWER)	2250000	562500	25%	2812500
ROAD ACCESS	318500	79625	25%	398125
COMMUNICATIONS	310000	77500	25%	387500
SITE ACCOMMODATION		0	0%	0
		0	0%	0

SUB-TOTAL 3598125

KATHERINE	ESTIMATE	CONTINGENCY	%	
STAFF HOUSING	825000	206250	25%	1031250
SINGLE QUARTERS	1662194	415549	25%	2077743
		0	0%	0
		0	0%	0
		0	0%	0

SUB-TOTAL 3108993

TOTAL - INFRASTRUCTURE 6,707,118

3. MINE DEVELOPMENT

MT. TODD	ESTIMATE	CONTINGENCY	%	
PERMANENT MINING FACILITIES		0	0%	0
PREPRODUCTION DEVELOPMENT	1778000	444500	25%	2222500
MINING EQUIPMENT		0	0%	0
		0	0%	0
		0	0%	0

TOTAL MINE DEVELOPMENT 2222500

MT. TODD GOLD PROJECT
PRE-FEASIBILITY STUDY
CIP PROCESS PLANT

CAPITAL COST SUMMARY

FIGURE 6.1

4.OWNER'S CAPITAL

(SHEET 3 OF 3)

MT. TODD	ESTIMATE	CONTINGENCY	%	
MOBILE EQUIPMENT	484000	121000	25%	605000
RECRUITMENT & TRAINING	400000	100000	25%	500000
PROCESS PLANT MAINTENANCE		0	0%	0
& REPLACEMENT (ANNUAL COST)	865750	216438	25%	1082188
INSURANCE DURING CONSTRUCTION	70000	17500	25%	87500

SUB-TOTAL 2274688

MT. TODD GOLD PROJECT
PRE-FEASIBILITY STUDY
CIP PROCESS PLANT

CAPITAL COST SUMMARY — FIGURE 6.1

KATHERINE

	ESTIMATE	CONTINGENCY	%	
SUNDRY FACILITIES	50000	12500	25%	62500
		0	0%	0
		0	0%	0
		0	0%	0
		0	0%	0
SUB-TOTAL				62500

TOTAL - OWNER'S CAPITAL 2,337,188

5. WORKING CAPITAL

MT. TODD

	ESTIMATE	CONTINGENCY	%	
PROCESS PLANT	800000	200000	25%	1000000
		0	0%	0
		0	0%	0
		0	0%	0
		0	0%	0
SUB-TOTAL				1000000

KATHERINE

	ESTIMATE	CONTINGENCY	%	
FACILITIES IN KATHERINE	50000	12500	25%	62500
		0	0%	0
		0	0%	0
		0	0%	0
		0	0%	0
SUB-TOTAL				62500

TOTAL - WORKING CAPITAL 1,062,500

TOTAL CAPITAL EXPENDITURE $50,525,449

7.0 OPERATING COSTS

Operating costs for mining have been documented in the mining pre-feasibility study (Appendix 3). This cost is comprised of the basic mining activity (drilling/blasting/handling/transporting) for both ore and waste, plus the cost of grade control and mine site services. Because this study is to be compared with the heap leaching result, the equivalent cost will be used in the financial analysis. It would be preferable to process the costs by year, reflecting the changes in ore to waste ratios. The mining operating costs have been stripped of any administration and labour costs which are covered by the overall manning or infrastructure costs.

Manning/labour costs are based on a total strength of 46 persons, and the manning is based on the same structure defined in the heap leach operation. After further enquiries at Pine Creek, the process operator wages were increased to between $41 - 43,000 P.A. The manning chart is illustrated in Figure 4.7 and Table 7.1 shows the costs.

Process costs are divided into reagents and steel in Table 7.2, and power costs in Table 7.3 are based on 10 cents/kWhr.

All operating costs have been updated to reflect recent changes in manning, grind size and leaching capacity.

The process operating costs (as revised) are:

Manning	-	$ 1.29/tonne of ore
Consumables	-	$ 4.26/tonne of ore
Power	-	$ 2.91/tonne of ore
Total	-	$ 8.46/tonne of ore

TABLE 7.1

Billiton Aust. Gold P.L.
Mt. Todd Gold Project
Manning

Title/Position Staff	No.	Salary	Burden Cost Factor	Cost/Tonne
Mine Manager	1	80,000	1.3	
Mine Engineer	1	75,000		
Mine Surveyor	1	55,000		
Mine Geologist	1	75,000		
Met. Supt.	1	75,000		
Plant Met.	1	45,000		
Gen. Foreman	1	50,000		
Gold Room Foreman	1	38,000		
Mining Engineer	1	42,000		
Geologist	1	42,000		
Metallurgist	1	42,000		
Chemist	1	38,000		
Maint. Engineer	1	45,000		
Admin. Officer	1	50,000		
Staff Total	14	772,000	1,003,600	50.18 c/tonne
Award				
Process				
Crushing	1 x 3	41,000		
Grinding	1 x 3	41,000		
C.I.P.	1 x 3	41,000		
Day Works	2	32,000		
Lead/hand/maint.	1 x 3	43,000		
Pit				
Grade Controllers	2	45,000		
Samplers	2	40,000		
Chainman	1	40,000		
Administration				
Store Clerk	1	30,000		
Accounts Clerk	1	30,000		
Computer Op.	1	25,000		
Typist/Reception	2	20,000		
Maintenance - Contract				
Fitter/Turner	2	44,000		
Boilermaker	2	44,000		
T/A	2	35,000		
Elect/Instruments	2	48,000		
Award Total	32	1,219,000	1,584,700	79.24 c/tonne
Site Total	46	1,991,000	2,588,300	$ 1.29 /tonne

TABLE 7.2

Operating Costs - Consumables

Consumable	Addition	Unit Cost $/kg	Annual Cost $'000	Annual Consumption tonnes	$/tonne ore
Sodium Cyanide	0.8 kg/t	2.80	4,480	1,600	2.24
Lime	1.1 kg/t	0.18	396	2,200	0.20
Sodium Hydroxide	0.2 kg/t	0.8	320	400	0.16
Lead Nitrate	0.4 kg/t max.	1.83	1,464	800	(0.73)
Flocculant	50 g/t max.	4.20	420	100	0.21
Carbon	15 g/t	4.5	135	30	0.07
Grinding Media	1.5 kg/t	0.8	2,400	3,000	1.20
Reagents & Fuel (Gold Room)					0.12
Water Treatment					0.01
Workshop/Lab.					0.03
Overheads					0.02
TOTAL					4.26

TABLE 7.3

Power Costs for a 2 million TPA
C.I.P. Process Plant
Basis 10 centers per KWht

Plant	Installated kW	Av.Drawn kW	Annual kWhrs	$/tonne	kWhrs/tonne
Comminution and Process	8,738.8 1,455.02	5,658.37 942.13	49,567,347 8,253,019	2.48 0.41	24.78 4.13
Process	111.69	39.09	342,442	0.02	0.17
Total	10,305.51	6,639.59	58,162,808	2.91	29.08
Basis	Comminution & Process	92.5%	availability,		7 days/week
	Process	50 %	availability,		7 days/week
	Diversity	70 %			

8.0 FINANCIAL ANALYSIS

8.1 GENERAL

The financial analysis of the project has been influenced by a number of factors:

- Gold price
- Exchange rates ($US/$A)
- Plant treatment rate
- Ore reserves
- Head grade
- Variation in operation cost
- Variation in capital cost

Table 8.1 presents the following information for use in the financial evaluation:

- Annual pit production
- Recovery rate
- Gold production
- Exchange rate
- Revenue
- Operating costs
- Capital costs

The net project cash flow before tax is presented together with net present value calculated with a discount rate of 14%.

8.2 BASIS OF FINANCIAL EVALUATION

The financial parameters are adopted in the base financial case:

Gold Price	-	an unescalated price of US$400 per oz. with an alternative case using US$550
Exchange Rate	-	exchange rate for USA have been used as shown below
Inflation	-	no account has been taken of inflation
Tax	-	the proposed tax on gold mining has been ignored
Royalty	-	royalty charges have been excluded
Discount Rate	-	a discount rate of 14% has been used to calculate net present value (NPV)

Internal Rate Return	-	internal rate of return (IRR) is the discount rate for which the NPV is zero
Finance	-	no financing charges have been included.

SCHEDULE OF GOLD PRICES AND EXCHANGE RATES

			CASE 1		CASE 2	
GOLD PRICE $US/oz			400		550	
EXCHANGE RATE ($US/$A)						
1989	Year	1	0.74		0.72	
1990		2	0.70		0.66	
1991		3	0.72		0.66	
1992		4	0.75		0.67	
1993		5	0.77		0.68	
1994		6	0.78		0.69	
1995		7	0.79		0.70	
1996		8	0.80		0.71	
1997		9	0.80		0.72	
1998		10	0.80		0.73	
1999		11	0.80		0.74	
2000+		12	0.80		0.75	

8.3 FINANCIAL ANALYSIS

The analysis undertaken is shown on the following tables:

Table No.	Case No.	Gold Price $US/oz	Treatment Rate (tpa)	Ore Reserve (t)
8.1A	1A	400	2,000,000	18,500,000
8.1B	1B	400	2,000,000	27,750,000
8.1C	1C	400	3,000,000	18,500,000
8.1D	1D	400	3,000,000	27,750,000
8.2A	2A	550	2,000,000	18,500,000
8.2B	2B	550	2,000,000	27,750,000
8.2C	2C	550	3,000,000	18,500,000
8.2D	2D	550	3,000,000	27,750,000

For each of the above cases, sensitivity to variation in grade, operating cost and capital cost by ±10% has been shown.

A cost scale up factor of 1.275 has been used to determine capital costs for a production rate of 3 mtpa.

FINANCIAL ANALYSIS SUMMARY (SHEET 1 OF 2)
MT. TODD C.I.L.
(MTODFBS1)
06-Feb-89

FILE NO.	TABLE NO.	CASE NO.	GOLD PRICE $US/$A	TREATMENT RATE tpa	ORE RESERVE t	FINAL YEAR
MTODFB01	8.1A	1A	400	2,000,000	20,018,912	YEAR 11
MTODFB02	8.1B	1B	400	2,000,000	30,028,368	YEAR 16
MTODFB03	8.1C	1C	400	3,000,000	20,018,912	YEAR 8
MTODFB04	8.1D	1D	400	3,000,000	30,028,368	YEAR 11
MTODFB05	8.2A	2A	550	2,000,000	20,018,912	YEAR 11
MTODFB06	8.2B	2B	550	2,000,000	30,028,368	YEAR 16
MTODFB07	8.2C	2C	550	3,000,000	20,018,912	YEAR 8
MTODFB08	8.2D	2D	550	3,000,000	30,028,368	YEAR 11

CASE 1A
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	-11233	22878	14945	23907	87283	68303
FACTOR	1	9169	9169	9169	61894	61894	61894
	1.1	29571	-4540	3393	99881	36505	55485

CASE 1B
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	-7459	31837	21928	43498	132108	103425
FACTOR	1	15967	15967	15967	96218	96218	96218
	1.1	39394	98	10007	148937	60327	89010

CASE 1C
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	-20060	19948	11222	6228	68977	51693
FACTOR	1	3831	3831	3831	43758	43758	43758
	1.1	27721	-12286	-3560	81287	18538	35822

CASE 1D
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	-9847	40851	28182	40062	135579	106109
FACTOR	1	20409	20409	20409	97155	97155	97155
	1.1	50665	-33	12636	154249	58731	88201

FINANCIAL ANALYSIS SUMMARY (SHEET 2 OF 2)
MT. TODD C.I.L.
(MTODFBS1)
06-Feb-89

FILE NO.	TABLE NO.	CASE NO.	GOLD PRICE SUS/SA	TREATMENT RATE tpa	ORE RESERVE t	FINAL YEAR
MTODFB01	8.1A	1A	400	2,000,000	20,018,912	YEAR 11
MTODFB02	8.1B	1B	400	2,000,000	30,028,368	YEAR 16
MTODFB03	8.1C	1C	400	3,000,000	20,018,912	YEAR 8
MTODFB04	8.1D	1D	400	3,000,000	30,028,368	YEAR 11
MTODFB05	8.2A	2A	550	2,000,000	20,018,912	YEAR 11
MTODFB06	8.2B	2B	550	2,000,000	30,028,368	YEAR 16
MTODFB07	8.2C	2C	550	3,000,000	20,018,912	YEAR 8
MTODFB08	8.2D	2D	550	3,000,000	30,028,368	YEAR 11

CASE 2A
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	83405	128032	120098	202197	285383	266403
FACTOR	1	114323	114323	114323	259994	259994	259994
	1.1	145240	100613	108547	317791	234605	253585

CASE 2B
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	99603	150795	140886	281770	396855	368172
FACTOR	1	134925	134925	134925	360965	360965	360965
	1.1	170248	119056	128965	440159	325074	353757

CASE 2C
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	91071	143426	134700	183880	266368	249084
FACTOR	1	127309	127309	127309	241149	241149	241149
	1.1	163548	111192	119918	298418	215930	233214

CASE 2D
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.9	130931	197270	184602	308332	433658	404188
FACTOR	1	176828	176828	176828	395234	395234	395234
	1.1	222726	156387	169055	482135	356810	386280

TABLE 8.1A CONTINUED

(MTODF801)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
CAPITAL COST																	
Salvage	$A '000																0
Process Plant	$A '000	38196		996	996	996	996	996			996						44172
Infrastructure	$A '000	6707															6707
Mine Development	$A '000	2223															2223
Owner's Capital	$A '000	1255			561				561								2376
Working Capital	$A '000	1063										-1063					0
Plant Maint. Capital	$A '000	280	860	860	860	860	860	860	860	860	860	589	0	0	0	0	8608
Cost Scale-Up Factor	1.00																
SUB-TOTAL CAPITAL COST	$A '000	49724	860	1856	2417	1856	1856	1856	1421	860	1856	-474	0	0	0	0	64087
TOTAL EXPENDITURE	$A '000	57618	24150	26756	27317	27856	27856	27856	27421	26860	27856	16431	0	0	0	0	317977
NET PROJECT CASH FLOW	$A '000	-47269	12712	9082	9200	10652	10739	11396	10775	11619	11471	11518	0	0	0	0	61894
CUMULATIVE CASH FLOW	$A '000	-47269	-34558	-25476	-16275	-5623	5115	16511	27286	38905	50376	61894	61894	61894	61894	61894	

NET PRESENT VALUE @ 14% = $9,169,004

CASE 1A

SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	NET PRESENT VALUE OPERATING COST	NET PRESENT VALUE CAPITAL COST	NET PROJECT CASH FLOW GRADE	NET PROJECT CASH FLOW OPERATING COST	NET PROJECT CASH FLOW CAPITAL COST
SENSITIVITY	0.90	-11233	22878	14945	23907	87283	68303
FACTOR	1.00	9169	9169	9169	61894	61894	61894
	1.10	29571	-4540	3393	99881	36505	55485

PROJECT: MT.TODD - C.I.L (MTCDFB01) FINANCIAL ANALYSIS TABLE 8.1A
06-Feb-89

CASE I A
NO TAX
GOLD PRICE - $US/ 400.00 troy oz.

RESERVE TONNAGE 20,018,912 t
PLANT CAPACITY 2,000,000 tpa
COST SCALE-UP FACTOR 1.00
PLANT MAINT. CAPITAL 0.43 $/t

WASTE TONNAGE 18,337,948 t
COSTS AT NOV. 1988
ROYALTIES EXCLUDED
INFLATION NOT INCLUDED

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
PRODUCTION																	
HEAP LEACH PRODUCTION																	
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	941	2000	2000	2000	2000	2000	2000	2000	2000	877					18338
C.I.L PRODUCTION																	
Ore Type 3	'000 t	650	2000	2000	2000	2000	2000	2000	2000	2000	2000	1369	0	0	0	0	20019
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
GOLD REVENUE																	
RECOVERY OF GOLD																	
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900										900					
GOLD PRODUCED	oz.	20046	64508	64508	68469	74128	75260	77523	76391	76957	78655	54958	0	0	0	0	731443
EXCHANGE RATE	$US/$A	0.74	0.70	0.72	0.75	0.77	0.78	0.79	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	
GOLD PRICE	$A/oz.	540.54	571.43	555.56	533.33	519.48	512.82	506.33	500.00	500.00	500.00	500.00	500.00	500.00	500.00	500.00	
TOTAL PRODUCTION REVENUE	$A '000	10349	36862	35838	36517	38508	38595	39252	38196	38479	39327	27949	0	0	0	0	379871
EXPENDITURE																	
OPERATING COSTS																	
Operating Costs - Unit Rates																	
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																	
Mining Cost	$A '000	2246	5910	7520	7520	8620	8620	8620	8620	8620	8620	5010	0	0	0	0	79926
Treatment Cost (incl. Admin.)	$A '000	5499	16920	16920	16920	16920	16920	16920	16920	16920	16920	11581	0	0	0	0	169360
Infrastructure	$A '000	85	260	260	260	260	260	260	260	260	260	178	0	0	0	0	2602
Rehabilitation	$A '000	65	200	200	200	200	200	200	200	200	200	137	0	0	0	0	2002
SUB-TOTAL OPERATING COST	$A '000	7395	23290	24900	24900	26000	26000	26000	26000	26000	26000	16906	0	0	0	0	253890

PROJECT: MT.TODD - C.I.L (MTODF502) FINANCIAL ANALYSIS TABLE 8.15
06-Feb-89

CASE 1 5	RESERVE TONNAGE	30,028,368 t	WASTE TONNAGE	28,839,213 t
NO TAX	PLANT CAPACITY	2,000,000 tpa	COSTS AT NOV. 1988	
GOLD PRICE - $US/ 400.00 troy oz.	COST SCALE-UP FACTOR	1.00	ROYALTIES EXCLUDED	
	PLANT MAINT. CAPITAL	0.43 $/t	INFLATION NOT INCLUDED	

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	YEAR 16	TOTAL
PRODUCTION																		
HEAP LEACH PRODUCTION																		
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	941	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	1378	28839
C.I.L PRODUCTION																		
Ore Type 3	'000 t	650	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	1378	30028
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	1.29	
GOLD REVENUE																		
RECOVERY OF GOLD																		
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900															900	
GOLD PRODUCED	oz.	20046	64508	64508	68469	74128	75260	77523	76391	76957	78655	80352	72996	72996	72996	72996	50308	1099090
EXCHANGE RATE	$US/$A	0.74	0.70	0.72	0.75	0.77	0.78	0.79	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	
GOLD PRICE	$A/oz.	540.54	571.43	555.56	533.33	519.48	512.82	506.33	500.00	500.00	500.00	500.00	500.00	500.00	500.00	500.00	500.00	
TOTAL PRODUCTION REVENUE	$A '000	10349	36862	35838	36517	38508	38595	39252	38196	38479	39327	40176	36498	36498	36498	36498	25604	563695
EXPENDITURE																		
OPERATING COSTS																		
Operating Costs - Unit Rates																		
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																		
Mining Cost	$A '000	2246	5910	7520	7520	8620	8620	8620	8620	8620	8620	8620	8620	8620	8620	8620	5941	123957
Treatment Cost (incl. Admin.)	$A '000	5499	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	11661	254040
Infrastructure	$A '000	85	260	260	260	260	260	260	260	260	260	260	260	260	260	260	179	3904
Rehabilitation	$A '000	65	200	200	200	200	200	200	200	200	200	200	200	200	200	200	138	3003
SUB-TOTAL OPERATING COST	$A '000	7895	23290	24900	24900	26000	26000	26000	26000	26000	26000	26000	26000	26000	26000	26000	17919	384904

TABLE 8.1B CONTINUED

(MTODFB02)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	YEAR 16	TOTAL
CAPITAL COST																		
Salvage	$A '000																	0
Process Plant	$A '000	38196		996	996	996	996	996			996	996	996	996	996			48156
Infrastructure	$A '000	6707																6707
Mine Development	$A '000	2223																2223
Owner's Capital	$A '000	1255			561				561				561					2937
Working Capital	$A '000	1063															-1063	0
Plant Maint. Capital	$A '000	280	860	860	860	860	860	860	860	860	860	860	860	860	860	860	593	12912
Cost Scale-Up Factor	1.00																	
SUB-TOTAL CAPITAL COST	$A '000	49724	860	1856	2417	1856	1856	1856	1421	860	1856	1856	2417	1856	1856	860	-470	72935
TOTAL EXPENDITURE	$A '000	57618	24150	26756	27317	27856	27856	27856	27421	26860	27856	27856	28417	27856	27856	26860	17448	457839
NET PROJECT CASH FLOW	$A '000	-47269	12712	9082	9200	10652	10739	11396	10775	11619	11471	12320	8081	8642	8642	9638	8155	105856
CUMULATIVE CASH FLOW	$A '000	-47269	-34558	-25476	-16275	-5623	5115	16511	27286	38905	50376	62697	70778	79420	88062	97700	96218	

NET PRESENT VALUE @ 14% = $17,346,714

CASE 1B
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	NET PRESENT VALUE OPERATING COST	NET PRESENT VALUE CAPITAL COST	NET PROJECT CASH FLOW GRADE	NET PROJECT CASH FLOW OPERATING COST	NET PROJECT CASH FLOW CAPITAL COST
SENSITIVITY	0.90	-7459	31837	21928	43498	132108	103425
FACTOR	1.00	15967	15967	15967	96218	96218	96218
	1.10	39394	98	10007	148937	60327	89010

PROJECT: MT.TODD - C.I.L (MTODFS03) FINANCIAL ANALYSIS TABLE 8.1C
06-Feb-89

CASE 1 C
NO TAX
GOLD PRICE - $US/ 400.00 troy oz.

RESERVE TONNAGE 20,018.912 t
PLANT CAPACITY 3,000,000 tpa
COST SCALE-UP FACTOR 1.275
PLANT MAINT. CAPITAL 0.43 $/t

WASTE TONNAGE 18,337.948 t
COSTS AT NOV. 1988
ROYALTIES EXCLUDED
INFLATION NOT INCLUDED

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
PRODUCTION																	
HEAP LEACH PRODUCTION																	
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	1641	3200	3274	3200	3200	3000	303								18338
C.I.L PRODUCTION																	
Ore Type 3	'000 t	650	3000	3000	3000	3000	3000	3000	1369	0	0	0	0	0	0	0	20019
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
GOLD REVENUE																	
RECOVERY OF GOLD																	
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900							900								
GOLD PRODUCED	oz.	20046	96762	96762	102704	111192	112889	116285	52287	0	0	0	0	0	0	0	708927
EXCHANGE RATE	$US/$A	0.74	0.70	0.72	0.75	0.77	0.78	0.79	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	
GOLD PRICE	$A/oz.	540.54	571.43	555.56	533.33	519.48	512.82	506.33	500.00	500.00	500.00	500.00	500.00	500.00	500.00	500.00	
TOTAL PRODUCTION REVENUE	$A '000	10349	55293	53757	54775	57762	57892	58878	26593	0	0	0	0	0	0	0	375300
EXPENDITURE																	
OPERATING COSTS																	
Operating Costs - Unit Rates																	
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																	
Mining Cost	$A '000	2246	9214	11584	11696	13292	13292	12930	3971	0	0	0	0	0	0	0	78226
Treatment Cost (incl. Admin.)	$A '000	5499	25380	25380	25380	25380	25380	25380	11581	0	0	0	0	0	0	0	169360
Infrastructure	$A '000	85	390	390	390	390	390	390	178	0	0	0	0	0	0	0	2602
Rehabilitation	$A '000	65	300	300	300	300	300	300	137	0	0	0	0	0	0	0	2002
SUB-TOTAL OPERATING COST	$A '000	7895	35284	37654	37766	39362	39362	39000	15867	0	0	0	0	0	0	0	252190

TABLE 8.1C CONTINUED

(MTODF303)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
CAPITAL COST																	
Salvage	$A '000																0
Process Plant	$A '000	38196	996	996	996	996	996	996									44172
Infrastructure	$A '000	6707															6707
Mine Development	$A '000	2223															2223
Owner's Capital	$A '000	1255			561												1816
Working Capital	$A '000	1063							-1063								0
Plant Maint. Capital	$A '000	290	1290	1290	1290	1290	1290	1290	589	0	0	0	0	0	0	0	8608
Cost Scale-Up Factor	1.275																
SUB-TOTAL CAPITAL COST	$A '000	63397	2641	2641	3356	2641	2641	2641	-605	0	0	0	0	0	0	0	80995
TOTAL EXPENDITURE	$A '000	71292	37925	40295	41122	42003	42003	41641	15262	0	0	0	0	0	0	0	333186
NET PROJECT CASH FLOW	$A '000	-60943	17368	13462	13654	15759	15889	17238	11331	0	0	0	0	0	0	0	42114
CUMULATIVE CASH FLOW	$A '000	-60943	-43575	-30113	-16460	-700	15189	32426	43758	43758	43758	43758	43758	43758	43758	43758	

NET PRESENT VALUE @ 14% = $3,830,821

CASE 1C
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.90	-20060	19948	11222	6228	68977	51693
FACTOR	1.00	3831	3831	3831	43758	43758	43758
	1.10	27721	-12286	-3560	81287	18538	35822

PROJECT: MT.TODD - C.I.L (MTODFB04) FINANCIAL ANALYSIS TABLE 8.1D
06-Feb-89

CASE 1 D
NO TAX
GOLD PRICE - $US/ 400.00 troy oz.

RESERVE TONNAGE 30,028,368 t
PLANT CAPACITY 3,000,000 tpa
COST SCALE-UP FACTOR 1.275
PLANT MAINT. CAPITAL 0.43 $/t

WASTE TONNAGE 29,412,969 t
COSTS AT NOV. 1988
ROYALTIES EXCLUDED
INFLATION NOT INCLUDED

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
PRODUCTION																	
HEAP LEACH PRODUCTION																	
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	1641	3200	3274	3200	3200	3000	3000	3000	3000	2378					29413
C.I.L PRODUCTION																	
Ore Type 3	'000 t	650	3000	3000	3000	3000	3000	3000	3000	3000	3000	2378	0	0	0	0	30028
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
GOLD REVENUE																	
RECOVERY OF GOLD																	
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900										900					
GOLD PRODUCED	oz.	20046	96762	96762	102704	111192	112889	116285	114587	115436	117982	95554	0	0	0	0	1100199
EXCHANGE RATE	$US/$A	0.74	0.70	0.72	0.75	0.77	0.78	0.79	0.80	0.80	0.80	0.80	0.80	0.80	0.80	0.80	
GOLD PRICE	$A/oz.	540.54	571.43	555.56	533.33	519.48	512.82	506.33	500.00	500.00	500.00	500.00	500.00	500.00	500.00	500.00	
TOTAL PRODUCTION REVENUE	$A '000	10349	55293	53757	54775	57762	57892	58878	57294	57718	58991	48227	0	0	0	0	570936
EXPENDITURE																	
OPERATING COSTS																	
Operating Costs - Unit Rates																	
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																	
Mining Cost	$A '000	2246	9214	11584	11696	13292	13292	12930	12930	12930	12930	10251	0	0	0	0	123295
Treatment Cost (incl. Admin.)	$A '000	5499	25380	25380	25380	25380	25380	25380	25380	25380	25380	20121	0	0	0	0	254040
Infrastructure	$A '000	85	390	390	390	390	390	390	390	390	390	309	0	0	0	0	3904
Rehabilitation	$A '000	65	300	300	300	300	300	300	300	300	300	238	0	0	0	0	3003
SUB-TOTAL OPERATING COST	$A '000	7895	35284	37654	37766	39362	39362	39000	39000	39000	39000	30919	0	0	0	0	384242

TABLE 8.1D CONTINUED

(MTODP504)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
CAPITAL COST																	
Salvage	SA '000																0
Process Plant	SA '000	38196	996	996	996	996	1992	996	996	996	996						48156
Infrastructure	SA '000	6707															6707
Mine Development	SA '000	2223															2223
Owner's Capital	SA '000	1255			561				561								2376
Working Capital	SA '000	1063										-1063					0
Plant Maint. Capital	SA '000	280	1290	1290	1290	1290	1290	1290	1290	1290	1290	1023	0	0	0	0	12912
Cost Scale-Up Factor	1.275																
SUB-TOTAL CAPITAL COST	SA '000	62397	2641	2641	3356	2641	3637	2641	3356	2641	2641	-51	0	0	0	0	92278
TOTAL EXPENDITURE	SA '000	71292	37925	40295	41122	42003	42999	41641	42356	41641	41641	30867	0	0	0	0	476519
NET PROJECT CASH FLOW	SA '000	-60942	17368	13462	13654	15759	14893	17238	14938	16077	17350	17359	0	0	0	0	94416
CUMULATIVE CASH FLOW	SA '000	-60943	-43575	-30113	-16460	-700	14193	31430	46368	62445	79796	97155	97155	97155	97155	97155	

NET PRESENT VALUE @ 14% = $20,408,912

CASE 1D
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE			NET PROJECT CASH FLOW		
		GRADE	OPERATING COST	CAPITAL COST	GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.90	-9847	40851	28182	40062	135579	106109
FACTOR	1.00	20409	20409	20409	97155	97155	97155
	1.10	50665	-33	12636	154249	58731	88201

PROJECT: MT.TODD - C.I.L (MTODFB05) FINANCIAL ANALYSIS TABLE 8.2A
06-Feb-89

CASE 2 A	RESERVE TONNAGE 20,018,912 t	WASTE TONNAGE 18,337,948 t	
NO TAX	PLANT CAPACITY 2,000,000 tpa	COSTS AT NOV. 1988	
GOLD PRICE - $US/ 550.00 troy oz.	COST SCALE-UP FACTOR 1.00	ROYALTIES EXCLUDED	
	PLANT MAINT. CAPITAL 0.43 $/t	INFLATION NOT INCLUDED	

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
PRODUCTION																	
HEAP LEACH PRODUCTION																	
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	941	2000	2000	2000	2000	2000	2000	2000	2000	877					18338
C.I.L PRODUCTION																	
Ore Type 3	'000 t	650	2000	2000	2000	2000	2000	2000	2000	2000	2000	1369	0	0	0	0	20019
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
GOLD REVENUE																	
RECOVERY OF GOLD																	
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900										900					
GOLD PRODUCED	oz.	20046	64508	64508	68469	74128	75260	77523	76391	76957	78655	54998	0	0	0	0	731443
EXCHANGE RATE	$US/$A	0.72	0.66	0.66	0.67	0.68	0.69	0.70	0.71	0.72	0.73	0.74	0.75	0.75	0.75	0.75	
GOLD PRICE	$A/oz.	763.89	833.33	833.33	820.90	808.82	797.10	785.71	774.65	763.89	753.42	743.24	733.33	733.33	733.33	733.33	
TOTAL PRODUCTION REVENUE	$A '000	14625	53757	53757	56206	59956	59990	60911	59176	58787	59260	41546	0	0	0	0	577971
EXPENDITURE																	
OPERATING COSTS																	
Operating Costs - Unit Rates																	
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																	
Mining Cost	$A '000	2246	5910	7520	7520	8620	8620	8620	8620	8620	8620	5010	0	0	0	0	79926
Treatment Cost (incl. Admin.)	$A '000	5499	16920	16920	16920	16920	16920	16920	16920	16920	16920	11581	0	0	0	0	169360
Infrastructure	$A '000	85	260	260	260	260	260	260	260	260	260	178	0	0	0	0	2602
Rehabilitation	$A '000	65	200	200	200	200	200	200	200	200	200	137	0	0	0	0	2002
SUB-TOTAL OPERATING COST	$A '000	7895	23290	24900	24900	26000	26000	26000	26000	26000	26000	16906	0	0	0	0	253890

TABLE 8.2A CONTINUED

(MTDDF305)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
CAPITAL COST																	
Salvage	$A '000																0
Process Plant	$A '000	38196		996	996	996	996	996			996						44172
Infrastructure	$A '000	6707															6707
Mine Development	$A '000	2223															2223
Owner's Capital	$A '000	1255			561				561								2376
Working Capital	$A '000	1063										-1063					0
Plant Maint. Capital	$A '000	280	860	860	860	860	860	860	860	860	860	589	0	0	0	0	8608
Cost Scale-Up Factor	1.00																
SUB-TOTAL CAPITAL COST	$A '000	49724	860	1856	2417	1856	1856	1856	1421	860	1856	-474	0	0	0	0	64087
TOTAL EXPENDITURE	$A '000	57618	24150	26756	27317	27856	27856	27856	27421	26860	27856	16431	0	0	0	0	317977
NET PROJECT CASH FLOW	$A '000	-42993	29607	27001	28889	32100	32134	33055	31756	31927	31404	25114	0	0	0	0	259994
CUMULATIVE CASH FLOW	$A '000	-42993	-13386	13614	42504	74604	106738	139793	171548	203475	234880	259994	259994	259994	259994	259994	

NET PRESENT VALUE @ 14% = $114,322,581

CASE 2A

SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.90	83405	128032	120098	202197	285383	266403
FACTOR	1.00	114323	114323	114323	259994	259994	259994
	1.10	145240	100613	108547	317791	234605	253585

PROJECT: MT.TODD - C.I.L (MTODFB06) FINANCIAL ANALYSIS TABLE 8.28
06-Feb-89

CASE 2 B	RESERVE TONNAGE 30,028,368 t	WASTE TONNAGE 28,839,213 t	
NO TAX	PLANT CAPACITY 2,000,000 tpa	COSTS AT NOV. 1988	
GOLD PRICE - $US/ 550.00 troy oz.	COST SCALE-UP FACTOR 1.00	ROYALTIES EXCLUDED	
	PLANT MAINT. CAPITAL 0.43 $/t	INFLATION NOT INCLUDED	

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	YEAR 16	TOTAL
PRODUCTION																		
HEAP LEACH PRODUCTION																		
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	941	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	1378	28839
C.I.L PRODUCTION																		
Ore Type 3	'000 t	650	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	2000	1378	30028
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	1.29	
GOLD REVENUE																		
RECOVERY OF GOLD																		
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900															900	
GOLD PRODUCED	oz.	20046	64508	64508	68469	74128	75260	77523	76391	76957	78655	80352	72996	72996	72996	72996	50308	1099090
EXCHANGE RATE	$US/$A	0.72	0.66	0.66	0.67	0.68	0.69	0.70	0.71	0.72	0.73	0.74	0.75	0.75	0.75	0.75	0.80	
GOLD PRICE	$A/oz.	763.89	833.33	833.33	820.90	808.82	797.10	785.71	774.65	763.89	753.42	743.24	733.33	733.33	733.33	733.33	687.50	
TOTAL PRODUCTION REVENUE	$A '000	14625	53757	53757	56206	59956	59990	60911	59176	58787	59260	59721	53531	53531	53531	53531	35205	845474
EXPENDITURE																		
OPERATING COSTS																		
Operating Costs - Unit Rates																		
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																		
Mining Cost	$A '000	2246	5910	7520	7520	8620	8620	8620	8620	8620	8620	8620	8620	8620	8620	8620	5941	123957
Treatment Cost (incl. Admin.)	$A '000	5499	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	16920	11661	254040
Infrastructure	$A '000	85	260	260	260	260	260	260	260	260	260	260	260	260	260	260	179	3904
Rehabilitation	$A '000	65	200	200	200	200	200	200	200	200	200	200	200	200	200	200	138	3003
SUB-TOTAL OPERATING COST	$A '000	7895	23290	24900	24900	26000	26000	26000	26000	26000	26000	26000	26000	26000	26000	26000	17919	384904

TABLE 5.2B CONTINUED

(MTODF506)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	YEAR 16	TOTAL
CAPITAL COST																		
Salvage	SA '000																	0
Process Plant	SA '000	38196		996	996	996	996	996			996	996	996	996	996			48156
Infrastructure	SA '000	6707																6707
Mine Development	SA '000	2223																2223
Owner's Capital	SA '000	1255			561				561				561					2937
Working Capital	SA '000	1063															-1063	0
Plant Maint. Capital	SA '000	280	860	860	860	860	860	860	860	860	860	860	860	860	860	860	593	12912
Cost Scale-Up Factor	1.00																	
SUB-TOTAL CAPITAL COST	SA '000	49724	860	1856	2417	1856	1856	1856	1421	860	1856	1856	2417	1856	1856	860	-470	72935
TOTAL EXPENDITURE	SA '000	57618	24150	26756	27317	27856	27856	27856	27421	26860	27856	27856	28417	27856	27856	26860	17448	457839
NET PROJECT CASH FLOW	SA '000	-42993	29607	27001	28889	32100	32134	33055	31756	31927	31404	31865	25114	25675	25675	26671	17757	387635
CUMULATIVE CASH FLOW	SA '000	-42993	-13386	13614	42504	74604	106738	139793	171548	203475	234880	266745	291859	317533	343208	369878	360965	

NET PRESENT VALUE @ 14% = $138,836,724

CASE 2B

SENSITIVITY ANALYSIS		NET PRESENT VALUE			NET PROJECT CASH FLOW		
$'000		GRADE	OPERATING COST	CAPITAL COST	GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.90	99603	150795	140886	281770	396855	368172
FACTOR	1.00	134925	134925	134925	360965	360965	360965
	1.10	170248	119056	128965	440159	325074	353757

PROJECT: MT.TODD - C.I.L (MTODFB07) FINANCIAL ANALYSIS TABLE 8.2C
06-Feb-89

CASE 2 C
NO TAX
GOLD PRICE - $US/ 550.00 troy oz.

RESERVE TONNAGE 20,018,912 t
PLANT CAPACITY 3,000,000 tpa
COST SCALE-UP FACTOR 1.275
PLANT MAINT. CAPITAL 0.43 $/t

WASTE TONNAGE 18,337,948 t
COSTS AT NOV. 1988
ROYALTIES EXCLUDED
INFLATION NOT INCLUDED

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
PRODUCTION																	
HEAP LEACH PRODUCTION																	
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	1641	3200	3274	3200	3200	3000	303								18338
C.I.L PRODUCTION																	
Ore Type 3	'000 t	650	3000	3000	3000	3000	3000	3000	1369	0	0	0	0	0	0	0	20019
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
GOLD REVENUE																	
RECOVERY OF GOLD																	
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900							900								
GOLD PRODUCED	oz.	20046	96762	96762	102704	111192	112889	116285	52287	0	0	0	0	0	0	0	708927
EXCHANGE RATE	$US/$A	0.72	0.66	0.66	0.67	0.68	0.69	0.70	0.71	0.72	0.73	0.74	0.75	0.75	0.75	0.75	
GOLD PRICE	$A/oz.	763.89	833.33	833.33	820.90	808.82	797.10	785.71	774.65	763.89	753.42	743.24	733.33	733.33	733.33	733.33	
TOTAL PRODUCTION REVENUE	$A '000	14625	80635	80635	84309	89935	89984	91366	41201	0	0	0	0	0	0	0	572691
EXPENDITURE																	
OPERATING COSTS																	
Operating Costs - Unit Rates																	
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																	
Mining Cost	$A '000	2246	9214	11584	11696	13292	13292	12930	3971	0	0	0	0	0	0	0	78226
Treatment Cost (incl. Admin.)	$A '000	5499	25380	25380	25380	25380	25380	25380	11581	0	0	0	0	0	0	0	169360
Infrastructure	$A '000	85	390	390	390	390	390	390	178	0	0	0	0	0	0	0	2602
Rehabilitation	$A '000	65	300	300	300	300	300	300	137	0	0	0	0	0	0	0	2002
SUB-TOTAL OPERATING COST	$A '000	7895	35284	37654	37766	39362	39362	39000	15867	0	0	0	0	0	0	0	252190

TABLE 8.2C CONTINUED

(MTODFB07)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
CAPITAL COST																	
Salvage	SA '000																0
Process Plant	SA '000	38196	996	996	996	996	996	996									44172
Infrastructure	SA '000	6707															6707
Mine Development	SA '000	2223															2223
Owner's Capital	SA '000	1255			561												1816
Working Capital	SA '000	1063							-1063								0
Plant Maint. Capital	SA '000	280	1290	1290	1290	1290	1290	1290	589	0	0	0	0	0	0	0	8608
Cost Scale-Up Factor	1.275																
SUB-TOTAL CAPITAL COST	SA '000	63397	2641	2641	3356	2641	2641	2641	-605	0	0	0	0	0	0	0	80995
TOTAL EXPENDITURE	SA '000	71292	37925	40295	41122	42003	42003	41641	15262	0	0	0	0	0	0	0	333186
NET PROJECT CASH FLOW	SA '000	-56667	42710	40341	43187	47932	47982	49726	25939	0	0	0	0	0	0	0	239506
CUMULATIVE CASH FLOW	SA '000	-56667	-13957	26384	69571	117503	165485	215210	241149	241149	241149	241149	241149	241149	241149	241149	

NET PRESENT VALUE @ 14% = $127,309,196

CASE 2C
SENSITIVITY ANALYSIS

$'000		NET PRESENT VALUE GRADE	OPERATING COST	CAPITAL COST	NET PROJECT CASH FLOW GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.90	91071	143426	134700	183880	266368	249084
FACTOR	1.00	127309	127309	127309	241149	241149	241149
	1.10	163548	111192	119918	298418	215930	233214

PROJECT: MT.TODD - C.I.L (MTODFB98) FINANCIAL ANALYSIS TABLE 8.2D
06-Feb-89

CASE 2 D
NO TAX
GOLD PRICE - $US/ 550.00 troy oz.

RESERVE TONNAGE 30,028,368 t
PLANT CAPACITY 3,000,000 tpa
COST SCALE-UP FACTOR 1.275
PLANT MAINT. CAPITAL 0.43 $/t

WASTE TONNAGE 29,412,969 t
COSTS AT NOV. 1988
ROYALTIES EXCLUDED
INFLATION NOT INCLUDED

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
PRODUCTION																	
HEAP LEACH PRODUCTION																	
Ore Type 1 C/O= 0.6	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
Ore Type 2	'000 t	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0	0
Head grade	gm./t	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	1.00	
WASTE	'000 t	520	1641	3200	3274	3200	3200	3000	3000	3000	3000	2378					29413
C.I.L PRODUCTION																	
Ore Type 3	'000 t	650	3000	3000	3000	3000	3000	3000	3000	3000	3000	2378	0	0	0	0	30028
Head Grade	gm./t	1.09	1.14	1.14	1.21	1.31	1.33	1.37	1.35	1.36	1.39	1.42	1.29	1.29	1.29	1.29	
GOLD REVENUE																	
RECOVERY OF GOLD																	
Ore Type 1	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 2	%	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	75.00	
Ore Type 3	%	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	88.00	
GOLD LOCK UP/CLEAN UP	oz.	-900										900					
GOLD PRODUCED	oz.	20046	96762	96762	102704	111192	112889	116285	114587	115436	117982	95554	0	0	0	0	1100199
EXCHANGE RATE	$US/$A	0.72	0.66	0.66	0.67	0.68	0.69	0.70	0.71	0.72	0.73	0.74	0.75	0.75	0.75	0.75	
GOLD PRICE	$A/oz.	763.89	833.33	833.33	820.90	808.82	797.10	785.71	774.65	763.89	753.42	743.24	733.33	733.33	733.33	733.33	
TOTAL PRODUCTION REVENUE	$A '000	14625	80635	80635	84309	89935	89984	91366	88765	88180	88891	71689	0	0	0	0	869014
EXPENDITURE																	
OPERATING COSTS																	
Operating Costs - Unit Rates																	
Mining Cost/tonne of waste	$A/t	1.52	1.52	1.52	1.52	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	1.81	
Mining Cost/tonne of ore	$A/t	2.24	2.24	2.24	2.24	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	2.50	
Treatment Cost (incl. Admin.)	$A/t	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	8.46	
Infrastructure	$A/t	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	0.13	
Rehabilitation	$A/t	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	0.10	
Operating Cost -Summary																	
Mining Cost	$A '000	2246	9214	11584	11696	13292	13292	12930	12930	12930	12930	10251	0	0	0	0	123295
Treatment Cost (incl. Admin.)	$A '000	5499	25380	25380	25380	25380	25380	25380	25380	25380	25380	20121	0	0	0	0	254040
Infrastructure	$A '000	85	390	390	390	390	390	390	390	390	390	309	0	0	0	0	3904
Rehabilitation	$A '000	65	300	300	300	300	300	300	300	300	300	238	0	0	0	0	3003
SUB-TOTAL OPERATING COST	$A '000	7895	35284	37654	37766	39362	39362	39000	39000	39000	39000	30919	0	0	0	0	384242

TABLE 8.2D CONTINUED

(MTODF808)

	UNIT	YEAR 1	YEAR 2	YEAR 3	YEAR 4	YEAR 5	YEAR 6	YEAR 7	YEAR 8	YEAR 9	YEAR 10	YEAR 11	YEAR 12	YEAR 13	YEAR 14	YEAR 15	TOTAL
CAPITAL COST																	
Salvage	$A '000																0
Process Plant	$A '000	38196	996	996	996	996	1992	996	996	996	996						48156
Infrastructure	$A '000	6707															6707
Mine Development	$A '000	2223															2223
Owner's Capital	$A '000	1255			561				561								2376
Working Capital	$A '000	1063										-1063					0
Plant Maint. Capital	$A '000	280	1290	1290	1290	1290	1290	1290	1290	1290	1290	1023	0	0	0	0	12912
Cost Scale-Up Factor	1.275																
SUB-TOTAL CAPITAL COST	$A '000	63397	2641	2641	3356	2641	3637	2641	3356	2641	2641	-51	0	0	0	0	92278
TOTAL EXPENDITURE	$A '000	71292	37925	40295	41122	42003	42999	41641	42356	41641	41641	30867	0	0	0	0	476519
NET PROJECT CASH FLOW	$A '000	-56667	42710	40341	43187	47932	46986	49726	46409	46539	47250	40821	0	0	0	0	392495
CUMULATIVE CASH FLOW	$A '000	-56667	-13957	26384	69571	117503	164489	214214	260623	307163	354413	395234	395234	395234	395234	395234	

NET PRESENT VALUE @ 14% = $176,828,407

CASE 2D

SENSITIVITY ANALYSIS		NET PRESENT VALUE			NET PROJECT CASH FLOW		
$'000		GRADE	OPERATING COST	CAPITAL COST	GRADE	OPERATING COST	CAPITAL COST
SENSITIVITY	0.90	130931	197270	184602	308332	433658	404188
FACTOR	1.00	176828	176828	176828	395234	395234	395234
	1.10	222726	156387	169055	482135	356810	386280

9.0 RECOMMENDATIONS FOR A FEASIBILITY STUDY

9.1 COMPARISON OF PROCESS ROUTES

Heap Leaching Process

Analysis of testwork carried out to-date yields little evidence that an overall recovery of 60% Au by heap leaching can be improved upon. The Telfer mine operates an ROM heap leach at this recovery level, however the mining and haulage cost has been incurred, and the project bears no infrastructure, process or comminution costs.

The financial analysis indicates that a substantial increase in Gold price would be required to make a heap leach operation viable at Mt. Todd. Metallurgically the ore is not amenable to this process, probably because at millimetre order sizings the Gold is not liberated, and the host rock exhibits very poor internal permeability to leach solutions.

It is recommended that heap leaching should be reserved for opportunities where waste material is available that could not be economically treated by C.I.P. processing. In this situation a long cycle leach process need only bear the cost of site preparation, pad construction and catchment control (plus a minor outlay in pumps and irrigation).

It is of interest that the heap leach project planned for the Potosi tailings in Bolivia is using second hand equipment, and operating at Silver head grades of 150 -240 g/t at 75% recovery. This equates roughly to a Gold head grade of ~4 g/t. Throughput is very small, and a ball mill is utilised to achieve a heap feed size of 600 microns.

C.I.P./C.I.L. Process

The C.I.P./C.I.L. process route is financially more viable for the low gold price at the prefeasibility stage. The design of the process plant was carried out for 24 hours leach time at 53 micron grind and 90% recovery. Review of testwork at a later date indicated that these parameters should change to 48 hours leach time at 75 micron grind and 88% recovery. Further testwork is being commissioned to confirm these parameters, and the financial analysis in this final report reflects the increase in capital cost required ($1.2 million less for mill, $1.937 million more for leach capacity, a nett increase of $0.737 million).

It is also of note that the C.I.P./C.I.L. desorption area was designed for 90% recovery from a head grade of 1.5 g/t, and is thus more expensive than required. Similarly a 15 metre tailings thickener has been costed, whereas settling testwork indicates that a 12 metre (high rate) unit will be adequate.

As discussed previously, the selection of a SAG/ball mill grinding circuit may not be the optimum circuit for comminution of Mt. Todd ore. Billiton Australia Gold Pty Ltd have expressed reservations with regard to SAG milling, and it is recommended that, prior to commencement of a feasibility study of the C.I.P./C.I.L process option, further testwork, analysis and cost estimation of comminution options be carried out. Reservations are primarily associated with the question of the particle size at which impact breakage ceases and attrition grinding commences in the SAG mill.

Accordingly, it is recommended that C.I.L./C.I.P. be the preferred process option.

9.2 RECOMMENDATIONS

A. Geotechnical

Mt. Todd C.I.P. Study

Recommendations for Further Geotechnical Investigation

1. Plant Area

   For final design of plant area foundations, especially mill foundations, more detailed characterisation and proving of rock will be necessary. In order to ensure non-resonant foundations it is necessary to establish a range of dynamic modulus values for the founding rock as close as possible to the final location of the mills.

2. Raw Water Dam

   To access seepage losses from the proposed raw water dam, it will be necessary to quantify losses by permeameter testing over the dam floor. To optimize dam wall design it will be necessary to quantify the available clayey materials.

3. Tailings Dams

   Investigation will be necessary at the proposed tailings dam sites to determine

   (a) the adequacy of dam foundations,

   (b) the base permeability characteristics, and

   (c) the availability and construction requirements of suitable dam wall construction materials, particularly for contaminated water retention dams.

   This data will confirm the suitability of the sites with respect to minimizing seepage to groundwater, and will enable design of dams to be optimised to minimize losses through the walls.

B. Feasibility Study

It is recommended that a feasibility study be carried out to define the financial viability of a C.I.P./C.I.L. process for treatment of Mt. Todd ore. This study should follow on from:

1. The second program of leaching and adsorption testwork.
2. The study of comminution options.
3. The final mine feasibility study.
4. The recommended program of geotechnical work.
5. Any further infrastructure studies which are required (i.e. power, bridge, communications and accommodation refinements).

10.0 CRITICAL PATH ANALYSIS

10.1 GENERAL

Project development has been analysed using TIMELINE software. The schedule is structured around the following general features:

1. The dates are, at this stage, arbitrary. A May 1 design start is assumed.

2. It is not considered realistic that civil works (site, tailings and raw water dam) can be carried out during the wet season. Actual design start date will affect the schedule as it relates to the wet season.

3. A three week period has been set aside prior to commencement of design for finalization of scope of work.

4. First Gold pour coincides with the completion of commissioning.

5. It is assumed that the tender, evaluation and mobilization process for contract mining would have a prior start on the process plant design.

6. It is assumed that Government and Statutory approvals are handled by the Client, with further engineering information being advised at the appropriate time in support of applications.

10.2 SCHEDULE

A project schedule (Figure 10.1) has been prepared for construction of the process plant and this programme assumes:

- A single contractor for design, procurement and construction management activities,

- The contractor is provided with sufficient flexibility in decision making to allow timely and responsible execution of all activities.

On this basis, total execution time between project commencement and Date of Practical Completion as shown in the attached schedule is 42 weeks. The execution time is constrained by the delivery period for the agitators. it may be possible to reduce this period following finalization of equipment supply orders and construction sub-contracts. This is discussed in more detail in 10.5.

Key activities within the project schedule include:

- Appointment of the engineering, procurement, construction management contractor,

- Specification and procurement of long lead items,

- Detailed design,

- On-site construction,

- Practical Completion, and

- Commissioning.

10.3 INFRASTRUCTURE

Power

Grid extensions are not included on the schedule as it has not been possible to obtain a firm period for extensions to the power grid. Purchase of transformers is seen as the major item. A period of six months has been utilised. This item could become critical if agitator/crusher/mill lead times are reduced.

Water

The crucial component of water supply is the raw water dam. Dam construction must be completed as close to the wet season as possible to obtain a fill for the dry season. The bore system, as currently proven, is capable of delivering only 75 m^3/hr, and thus the raw water dam is an essential source for start-up purposes.

It is assumed that pre stripping would supply necessary fill, and thus determines the dam construction schedule.

Other

Accommodation, road/bridge and communications are not incorporated in the schedule. It is recommended that these facilities be provided as early as possible in the schedule period.

10.4 MINING

Pre stripping is scheduled to commence week 14, with a duration of 10 weeks. Production mining commences week 24 and thus generates a 580,000 tonne ore stockpile prior to commissioning in week 28.

10.5 PROCESS

The process schedule incorporates three long lead items - in decreasing order:

A. Agitators - the current critical path

B. Mills

C. Crushers

It is these items, transformers aside, which determine the final phase of installation prior to, and in conjunction with, commissioning (the final mechanical, pipework and electrical installations).

The schedule has been assembled to minimise execution time, with the preparation of specifications for long lead items commencing as early as possible. The area which remains uncertain is that of dam construction which, as previously mentioned, is a function of how the schedule relates to the start of the wet season.

10.6 RECOMMENDATIONS

To realise a significant reduction in project execution time early commencement of design and procurement activities will be necessary.

Criteria for agitator selection are as follows:

1. Established grind size, leach residence time and individual tank size.

2. Pilot testing to determine viscosity effects at operation pH, and ultimately a scale up factor for agitator type and power draw (by the manufacturer).

Overall site power requirements will need to be determined as early as possible. This could be achieved if the crushing, grinding and agitator selection processes were brought forward as prior design work. The transformer size can be determined and an order placed immediately on project commitment. Second hand and standard size transformers may be a viable option as discussed below.

This approach would also be necessary to reduce the grinding circuit lead time, which is very similar to that for agitators and transformers. A rod mill/ball mill circuit would be the simpler, quicker and more reliable option for this purpose. The lead time for crushing would not alter if a secondary crushing or tertiary crushing stage was required.

It is recommended that:

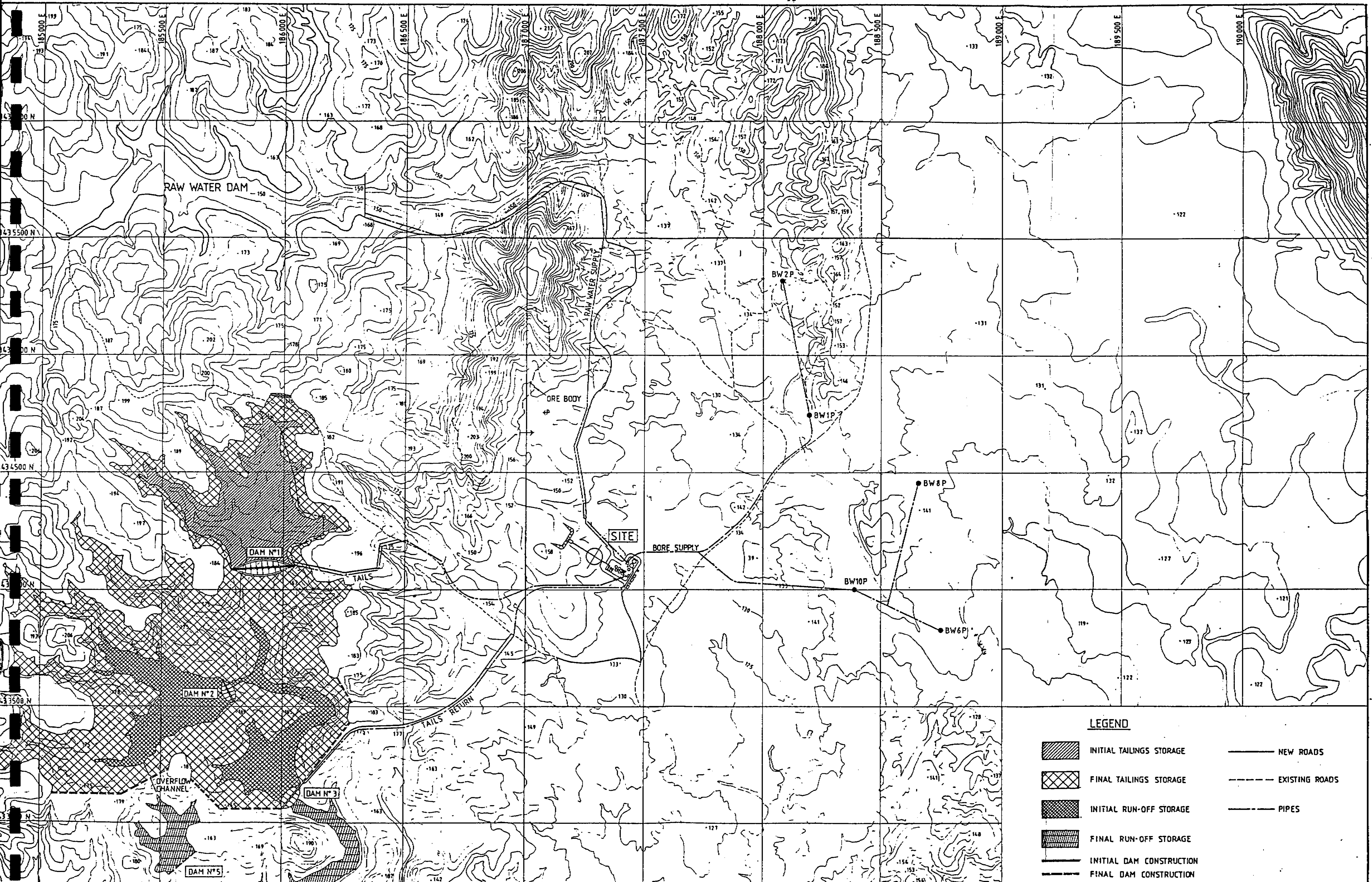
1. A program of testwork be implemented immediately to determine agitator selection criteria.

2. A study to be commissioned to assess and size the crushing and grinding circuits.

3. An estimate of total site power needs be made, with appropriate contingency. Second hand and "standard" size transformers offer reduced lead times, and a cross over parallel system with excess capability may be the most cost effective (particularly if a dual line grinding circuit is likely). Caution is advised as the cost curve for increasing transformer size has severe irregularities at different points.

4. The Northern Territory Power and Water Authority be requested to supply detailed information, on lead times for extension of the power grid to the Mt. Todd Gold Project site.

5. That appropriate consultants be engaged to refine scope and costs for infrastructure requirements such that these facilities may be provided as early as possible in the project design phase.



MINPROC ENGINEERS PTY. LTD.

67 ST. PAUL'S TERRACE, SPRING HILL QUEENSLAND 4000.
TELEPHONE: (07) 839 0363

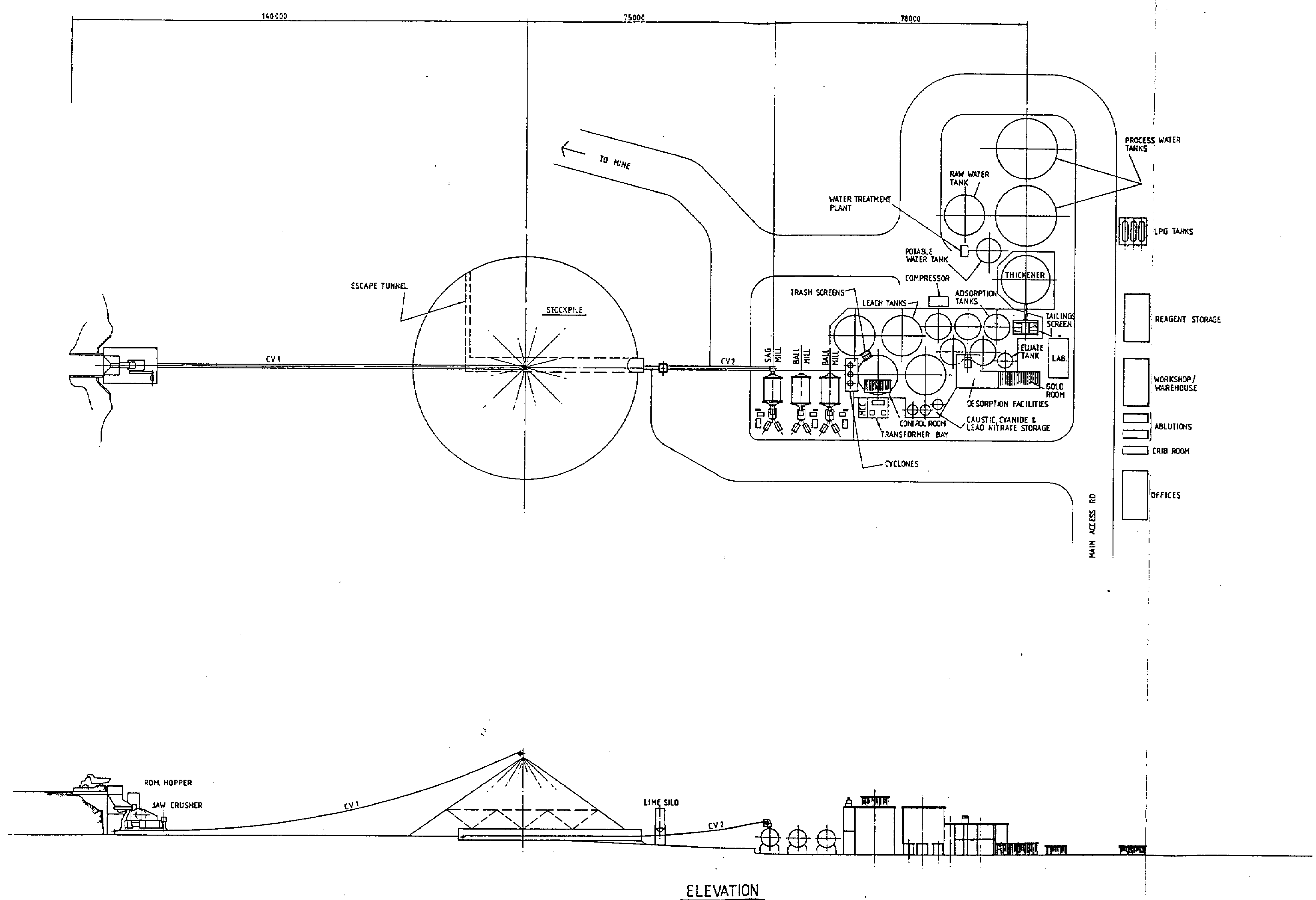
J.N FIELD 10.1.89

CLIENT: BILLITON AUSTRALIA GOLD PTY. LTD.

SIZE: AO SCALE: AS SHOWN

TITLE: MT. TODD GOLD PROJECT
2·0 M.T.P.A.
SITE LAYOUT

MTG-01-G-023



CLIENT APPROVED for CONSTRUCTION

DATE : SIGNED :

DRAWING No | REFERENCE DRAWINGS

MINPROC ENGINEERS PTY. LTD.

67 ST. PAUL'S TERRACE, SPRING HILL QUEENSLAND 4000.

TELEPHONE: (07) 839 0383

	BY	DATE
PROJECT APPR		
DESIGN APPR		
DESIGNED		
CHECKED		
DRAWN	J.N. FIELD	9-1-89

No | BY | DATE | REVISION | CHECKED | APPROV

PLANT GRID NORTH

CLIENT: BILLITON AUSTRALIA GOLD PTY. LTD

SIZE: B1 SCALE: 1:500

TITLE: MOUNT TODD GOLD PROJECT
2·0 M.T.P.A.
GENERAL ARRANGEMENT

REV No

DRAWING No: MTG-01-G-022

APPENDIX 2

Mount Todd Feasibility Study - Contract Mining Costs

A brief specification was prepared and submitted to some eight Contractors. Replies were received from six contractors:

Theiss Contractors Pty., Ltd.
Leighton Contractors Pty., Ltd.
Roche Bros. Darwin
Holdaway MacMahon
Curtain Bros. (Qld) Pty., Ltd.
TTS Construction

The prices submitted are collated over in Table 1 and the source letters are also attached for reference. The prices range reasonably well around the estimated mining price in the study after applying the average strip ratio of .92 tonnes of waste to 1 tonne of ore.

The following qualifications to the prices submitted apply:

- The prices really represent the first five years of mining. Once the favorable strip ratio ore on top of the hill is mined off and the pit commences mining, costs for waste haulage may increase.

- No electricity or water charges have been included.

- The indicative haul distances may not be achievable in practice as the pit haul road will need to exit the pit on either the ore side or the waste side with either ore or waste having to be hauled around the pit on the 160 bench. If the waste dump is re-located to the south then the one pit exit would be required and the potential problem resolved.

- There are a number of issues requiring solution in regard to the pit scheduling and individual bench tonnages shown in the mining document. Resolution of these will probably lower the waste to ore ratio overall, improve the front end project economics by taking into account the favorable early strip ratios and possibly lower the overall mining costs. These issues can be summarised as follows:

    - Some of the lower benches have more total tonnage than the benches above them.

    - The highest tonnage waste appears on the bottom bench.

    - The indicative mining schedule does not seem to take into account the favorable ratios on the initial benches. It would probably be very difficult to mine up to two million tonnes of waste in the early years of the project. The most likely optimum schedule is one that constrains total movement in the early years of the schedule to that necessary to achieve the desired mining rate of two million tonnes per annum and reasonable forward development. Alternatively it may be practical to alter the cut-off level in the early years of the project to improve project cash flows.

The table is constructed as follows:

1. Haul to dump distance - use 500 metres (overburden (c)).

2. Load to pads is for information only (ore (c)).

3. $/tonne waste by 0.92 plus $/tonne ore plus admin$/tonne = Total #.

4. 4% day work and mobilisation costs are included in Mine site Services and Mine Preproduction Costs respectively.

5. Sampling/assay and Mine Site Services costs are extracted from the existing Mine Pre-feasibility Study.

TABLE 1

MT. TODD GOLD PROJECT

CONTRACT RATES FOR MINING

Item	Unit	Roche Bros	Theiss	Holdaway Macmahon	Leighton	Curtain	TTS
Overburden							
(a) Rip and Load	bm^3			0.25	0.82	1.80	0.90
(b) Drill and blast	bM^3	1.35	1.87	1.35	1.00	1.35	1.45
(c) Haul to dump	500M	2.71	2.55	2.60	1.49	1.90	2.50
(d) Haul to dump	1000M	3.20	2.68	2.90	1.68	2.60	2.85
(e) Haul to dump	2000M	3.68	3.06	3.50	-	3.10	4.30
Ore							
(a) Drill and blast	tonne	0.60	0.51	0.70	0.57	0.55	0.75
(b) Load to ROM	tonne	1.42	0.92	1.20	0.89	1.80	1.55
(c) Load to Pads	tonne	1.40	1.03	0.75	0.92	1.20	0.90
Administration							
(a) Mobil.		332,000	210,000	120,000	331,039	450,000	150,000
(b) Demobil.		80,000	5,000	25,000	165,520	300,000	70,000
(c) Accommodation M			43,982	30,000	13,212*	30,000	18,000
Accommodation Y		511,000	527,784	360,000	687,024	360,000	216,000
Total Unit Basis							
. Waste	bM^3	4.06	4.42	4.25	2.49	3.25	3.95
	tonne	1.62	1.77	1.70	1.00	1.30	1.58
. Ore	tonne	2.02	1.43	1.90	1.46	2.35	2.30
. Admin/t (2.0M)		0.25	0.26	.18	0.34	0.18	0.11
. Total at (0.92)#		3.76	3.32	3.64	2.72	3.73	3.86
. + 4% Day Work		3.91	3.45	3.79	2.83	3.88	4.02
. + Mobilisation		.04	.02	.01	.05	.07	.02
		3.95	3.47	3.80	2.88	3.95	4.04

. AV 3.68 RANGE (2.88 - 4.04)

* Weekly Cost

. Av of TOTAL # 3.51 (2.72 - 3.86)

+ sampling/assay .21
+ Mine site services .18

TOTAL 3.90 For Comparison

. Cheapest 3.11

ROCHE BROS

MINING & CIVIL ENGINEERING CONTRACTORS

ROCHE BROS. PTY. LTD.
(Incorporated in Victoria)
REPLY TO:
1761 GRAFFIN CRESCENT,
WINNELLIE, 5789, NORTHERN TERRITORY.
P.O. BOX 2149, DARWIN N.T. 5794
TELEPHONE: (089) 47 0250
FACSIMILE: (089) 84 3505
TELEX: AA 85551

REF: 701-027

16th December 1988

The Study Manager
Mt Todd Gold Project
Minproc Engineers
P.O. Box 544
SPRING HILL QLD 4000

Dear Sir,

RE: MT TODD GOLD PROJECT - N.T.

We have pleasure in submitting our budget prices for the above project. Our prices are based on the specifications and information supplied.

Roche Bros. experience and expertise and depth of resources is well established throughout the industry.

We have a well established office and workshop complex in Darwin from which all Northern Territory projects are managed.

Existing and previous contract mining operations in the Territory and northern Western Australia include:

*	Cosmo Howley	(Current)
*	Granites	(Current)
*	Bow River	(Current)
*	Toms Gulley	(Current)
*	Moline	(Current)
*	White Devil	
*	Northern Star	
*	Golden Duke	
*	Woodcutters	
*	Argyle (Hard Rock)	
*	Argyle (Alluvial)	

We offer any or all of these as references to our capability in contract mining in the Top End.

..2/

ROCHE BROS

We thank you for the opportunity of quoting budget prices for this project,and look forward to receiving tender documents in due course.

Please contact the undersigned should you require any further information.

Assuring you of our best services.

Yours faithfully
ROCHE BROS. PTY LTD

C.H. DIEBEN
MANAGER - NORTHERN TERRITORY

Encl.
CHD/sm

ROCHE BROS

REF:701-026

MT TODD GOLD PROJECT
BUDGET SCHEDULE OF RATES

ITEM	DESCRIPTION	UNIT	RATE $
1.0	WASTE MINING		
1.1	Load oxide waste	BCM	0.67
1.2	Load Sulphide waste	BCM	0.79
1.3	HAUL WASTE To dump (inc RD & dump maintenance) up to 500m lead.		
	(a) RL 130 to 200 (Oxides)	BCM	2.04
	(b) RL 130 to 200 (Sulphides)	BCM	2.43
	(c) RL 90 to 130 (Sulphides)	BCM	2.94
	(d) RL 50 to 90	BCM	3.43
1.4	Extra over Item 1.3 for up to 1000m lead. ALL LEVELS	BCM	0.49
1.5	Extra over Item 1.3 for up to 2000m lead. ALL LEVELS	BCM	0.97
1.6	DRILL & BLAST WASTE		
	(a) Oxides	BCM	0.88
	(b) Sulphides	BCM	1.35
2.0	ORE MINING		
2.1	Load Oxide ore	BCM	0.63
2.2	Load Sulphide ore	BCM	0.81
2.3	Haul ore to R.O.M. (inc RD & S/pile maintenance) at 1000m lead.		
	(a) RL 130 to 200 (Oxide)	BCM	2.73
	(b) RL 130 to 200 (Sulphide)	BCM	3.12
	(c) RL 90 to 130 (Sulphide)	BCM	3.66
	(d) RL 50 to 90 (Sulphide)	BCM	4.19
2.4	DRILL & BLAST ORE		
	(a) Oxides	BCM	0.99
	(b) Sulphides	BCM	1.52

..2/

ITEM	DESCRIPTION	UNIT	RATE $
3.0	ADMINISTRATION COST		
3.1	MOBILISATION		
	(a) Plant & Facilities	LS	182,000
	(b) Camp	LS	150,000
3.2	DEMOBILISATION		
	(a) Plant & Facilities	LS	70,000
	(b) Camp	LS	10,000
3.3	ACCOMMODATION - Maintain & Operate 35 man singlemens camp	MAN DAY	40
4.0	MISCELLANEOUS WORKS		
4.1	Feed main primary crusher ROM S/pile (6000 T/day on double shift)	TONNE	0.44
4.2	Load crushed ore ex S/pile,haul 500m, dump & stack on leach pads,	TONNE	1.40

HOLDWAY-MACMAHON
(QLD.) PTY. LTD.

Civil Engineers &
Earthmoving
Contractors

799 Fairfield Road,
Yeerongpilly,
Qld. 4105

P.O. Box 174,
Moorooka,
Qld. 4105

Telephone: (07) 848 2061
Facsimile: (07) 892 2305
Telex: AA145196

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21 DEC 1988
ANS / /

WDT:MAB:360:1:88/256

16th December 1988

The Study Manager,
Mt.Todd Gold Project,
Minproc Engineers,
P.O. Box 544,
SPRING HILL QLD 4004

Attention: Mr. Mike Gunn

Dear Sir,

RE: MT TODD GOLD PROJECT - NORTHERN TERRITORY

We thank you for the opportunity to submit budget estimates for this project.

Attached is the completed Schedule of Rates. This schedule is based on the following:-

1. Rates submitted are at current prices and Rise & Fall would apply.
2. Rates are average for benches to RL140m only.
3. Item 1a) is an extra over the rate for loading and hauling of blasted material.
4. Items 1c), 1d) and 1e) are load and haul rates for blasted material.
5. Item 2 has not been corrected for dry tonnes.
6. Construction of haul roads not included in the mining rates.

We look forward to your further advice.

Yours faithfully,

HOLDWAY-MACMAHON (QLD.) PTY.LTD.

W.D. Thomson

W.D. THOMSON B.E. M.I.E.(Aust.)
SENIOR ESTIMATING ENGINEER

HOLDWAY-MACMAHON (QLD.) PTY.LTD.

MT TODD GOLD PROJECT

SCHEDULE OF PRICES REQUESTED

ITEM	DESCRIPTION		UNIT	RATE
1	OVERBURDEN			
	(a)	Rip ~~and load~~ overburden	b m^3	0.25
	(b)	Drill and blast overburden	b m^3	1.35
	(c)	Haul overburden to dump 500 metres away from the pit exit	b m^3	2.60
	(d)	Haul overburden to dump 1000 metres away from pit exit	b m^3	2.90
	(e)	Haul overburden to dump 2000 metres away from pit exit	b m^3	3.50
2	ORE			
	(a)	Drill and blast ore	t	0.70
	(b)	Load ore and haul to ROM stockpile	t	1.20
	(e)	Load crushed ore and haul to pads 500 metres away	t	0.75
3	ADMINISTRATION COSTS			
	(a)	Mobilisation		120,000
	(b)	Demobilisation		25000
	(c)	Monthly Accommodation Cost (not to be included in above rates)		~~14000~~

30,000

Verbal quote
4 Jan 89 on
query to Holdaway.

Jamieson

THIESS
CONTRACTORS PTY. LTD.
(INCORPORATED IN QUEENSLAND)

16th December 1988

The Study Manager
Mt Todd Gold Project
Minproc Engineers
P O Box 544
SPRING HILL QLD 4000

Dear Sir

RE: MT TODD GOLD PROJECT, N.T. - BUDGET ESTIMATE

We have pleasure in submitting our Schedule of Rates Budget Estimate for the above project. Our budget estimate has been prepared in accordance with the documents and drawings supplied with your letter of 8th December 1988.

The Schedule of Rates is submitted in the form of a computer print out. The quantities used were calculated on the basis of a 3 year contract period with ore extraction at the rate of 2,000,000 tonnes per annum. The waste volume was calculated from the bench quantities supplied by your Mr J. Jamieson by phone on 14th December 1988.

Our price is calculated in December 1988 dollars with no allowance for escalation.

Our price is based on the Mining Fuel Rebate of $0.18899/litre applying to all diesel fuel used on this contract.

As we propose using a hydraulic excavator for all excavation, we have not allowed for any ripping and have assumed all material will be drilled and shot at appropriate patterns and powder factors. Our drill and blast rates are based on our interpretation of the limited information available and are subject to confirmation.

For the purposes of calculation, we have assumed all waste at 2.5T/bm3 and all ore at 2.65T/bm3.

Should you require further information or clarification of this submission please contact the undersigned.

Yours faithfully
THIESS CONTRACTORS PTY LTD

D.G. McLelland

for W G Turner
MANAGER - ENGINEERING

QUEENSLAND OFFICE:
146 KERRY ROAD,
ARCHERFIELD,
BRISBANE QLD 4108

POSTAL ADDRESS:
P.O. BOX 199,
ARCHERFIELD 4108

TELEPHONE (07) 275 8500
TELEX AA41236
FAX (07) 277 8840

DER NO.C225
TODD GOLD PROJECT.

ITEM	DESCRIPTION	QUANTITY	UNITS	RATE	TOTAL
.	MT TODD GOLD PROJECT.				
.	#####################				
1	OVERBURDEN.				
.	-----------				
1.A	RIP & LOAD OVERBURDEN	1391200.00	BM3	Included in 1C, 1D & 1E	
1.B	DRILL & BLAST OVERBURDEN	1391200.00	BM3	1.07	1,488,584
1.C	HAUL OVERBURDEN TO DUMP 500m FROM PIT EXIT	1391200.00	BM3	2.55	3,547,560
1.D	HAUL OVERBURDEN TO DUMP 1000m FROM PIT EXIT		BM3	2.68	RATE ONLY
1.E	HAUL OVERBURDEN TO DUMP 2000m FROM PIT EXIT		BM3	3.06	RATE ONLY
2	ORE.				
.	----				
2.A	DRILL & BLAST ORE	6000000.00	TONNE	0.51	3,060,000
2.B	LOAD & HAUL ORE TO ROM STOCKPILE	6000000.00	TONNE	0.92	5,520,000
2.E	LOAD & HAUL CRUSHED ORE 500m TO PADS	6000000.00	TONNE	1.03	6,180,000
3	ADMINISTRATION COSTS.				
.	---------------------				
3.A	MOBILISATION		LS		210,000
3.B	DEMOBILISATION		LS		5,000
3.C	MONTHLY ACCOMMODATION COST (NOT INCL IN RATES	36.00	MONTH	43902.00	1,580,472

			TENDER TOTAL		21,591,616

--### FINAL PAGE ###--

CIVIL AND
MECHANICAL ENGINEERING
BUILDING CONSTRUCTION AND DESIGN
PROJECT MANAGEMENT

19 Lang Parade
Milton, Queensland
Australia 4064

P.O. Box 288
Toowong 4066

Telephone: (07) 870 3355
Fax: (07) 870 1451
Cables: LEIGHTON, BRISBANE
Telex: AA41024

RECEIVED
20 DEC 1988
ANS / /

LEIGHTON CONTRACTORS PTY. LTD.

(Incorporated in New South Wales)

Ref: TRC/kmj/TF03

16 December 1988

The Study Manager
Mt Todd Gold Project
Minproc Engineers
PO Box 544
SPRING HILL QLD 4000

FAXED
16 DEC 1988

Attention: Mr Jeff Jamieson

Dear Sir,

RE: MT TODD GOLD PROJECT
NORTHERN TERRITORY

Please find enclosed our Schedule of Budget Prices for the mining of the above works.

In our estimate we have made certain assumptions which are outlined on the attached schedules.

We trust this is to your satisfaction and we await your further advice.

Yours faithfully,
LEIGHTON CONTRACTORS PTY LTD

T R COGILL
Engineering Manager

SCHEDULE OF ASSUMPTIONS

1. The average waste dump haul is 1000m from the edge of the pit, level slope.

2. The average ore haul is 500 metres from the edge of the pit, slightly downhill to the ROM stockpile.

3. For the first three years, the following rates should apply;

Item 1 (d)	Haul Overburden to Dump	$1.49
Item 2 (b)	Load ore and haul to ROM stockpile	$0.83

Leighton Contractors Pty Ltd

JO'Rogue

MT TODD GOLD PROJECT

SCHEDULE OF PRICES REQUESTED

ITEM	DESCRIPTION		UNIT	RATE
1	OVERBURDEN			
	(a)	Rip and load overburden	m^3	\$0.82
	(b)	Drill and blast overburden	m^3	\$1.00
	(c)	Haul overburden to dump 500 metres away from the pit exit		N/A 1.45
	(d)	Haul overburden to dump 1000 metres away from pit exit		\$1.68
	(e)	Haul overburden to dump 2000 metres away from pit exit		N/A
2	ORE			
	(a)	Drill and blast ore	t	0.57
	(b)	Load ore and haul to ROM stockpile	t	0.89
	(e)	Load crushed ore and haul to pads 500 metres away	t	0.92
3	ADMINISTRATION COSTS			
	(a)	Mobilisation	Item	331039
	(b)	Demobilisation	Item	165520
	(c)	Monthly Accommodation Cost (not to be included in above rates) (Capacity 42 No)	Week	13212

LEIGHTON

ADDRESS ALL CORRESPONDENCE TO:
P.O. BOX 488,
GARBUTT, Q. 4814.

TELEPHONE: (077) 746199
TELEX: AA47304
FAX:
ADMIN (077) 74-6450
MANAGEMENT: (077) 74-6446
PURCHASING: (077) 74-6257

CURTAIN BROS. (QLD.) PTY. LTD.

CIVIL ENGINEERING, CONTRACT MINING AND GENERAL CONTRACTING

891 INGHAM ROAD, BOHLE ESTATE, TOWNSVILLE.
QUEENSLAND, AUSTRALIA.

14 December 1988

Our ref. : M1-1-L.001

Minproc Engineers Pty Ltd
PO Box 544
SPRING HILL QLD 4004

Attention: Mr Mike Gunn

Dear Sir,

MT TODD GOLD PRICES
BUDGET PRICES

Attached is our completed schedule for the above.

We amplify some aspects of our rates as follows: -

.. Our rates have no allowance in them for infra-structure works; i.e. clearing, construction of haul roads, etc. They are purely rates for mining

.. We have assumed, in our drill & blast rates, that benches in both waste and ore will be drilled to 5 m.

.. Our budget figures for mob and demob are based on taking all required equipment, camp, facilities, etc. from Townsville. At the Tender stage, we would look closer at the availability of camps, etc. nearer to the job site.

Trust the attached is of assistance.

Yours faithfully,
CURTAIN BROS. (QLD.) PTY. LTD.

B. T. Gaylor

B.T. GAYLOR,
Contracts Engineer

MT TODD GOLD PROJECT

SCHEDULE OF PRICES REQUESTED

ITEM		DESCRIPTION	UNIT	RATE
1		OVERBURDEN		
	(a)	Rip and load overburden	m^3	$1.80
	(b)	Drill and blast overburden *	m^3	$1.35
	(c)	Haul overburden to dump 500 metres away from the pit exit		$1.90
	(d)	Haul overburden to dump 1000 metres away from pit exit		$2.60
	(e)	Haul overburden to dump 2000 metres away from pit exit		$3.10
2		ORE		
	(a)	Drill and blast ore **	t	$0.55
	(b)	Load ore and haul to ROM stockpile	t	$1.80
	(e)	Load crushed ore and haul to pads 500 metres away	t	$1.20.
3		ADMINISTRATION COSTS		
	(a)	Mobilisation		$450,000
	(b)	Demobilisation		$300,000
	(c)	Monthly Accommodation Cost (not to be included in above rates)		$30,000

* Dry holes. Add $0.50/m³ for wet holes

** Dry holes. Add $0.20/t. " " "

Head Office: 35-53 Morehead Street, Townsville
Address all correspondence to Box 5422, TMC, Townsville, QLD. 4810
Telephone (077) 222777 Fax (077) 211387
Telex 47012 Telegraphic Address "Transerve"

TTS CONSTRUCTION

An Operating Division of TTS TRANSPORT PTY. LTD.

December 16, 1988.

Minproc Engineers Pty Ltd
P.O. Box 544
SPRING HILL. 4004.

Attention: Mike Gunn

Dear Sir

RE: MT. TODD GOLD PROJECT

In response to your request of December 8, 1988, we have pleasure in submitting our budget prices as per attached schedule.

You will note we have amended items on the schedule to provide clarification.

1.0 As there is no allowance to load overburden under Item 1(b), by hauling under 1(c) (d) or (e) we have modified 1(a) to a "rip and push" operation and included "load" in items 1(c), 1(d) and 1(e).

2.0 Item 2(b) is qualified to allow hauling to stockpile up to 1000 metres away from pit exit.

3.0 Item 1(c) is assumed to have an average haul distance of 400 metres from load point to pit exit.

4.0 Item 1(d) is assumed to have an average haul distance of 800 metres from load point to pit exit.

5.0 Item 1(e) is assumed to have an average haul distance of 1200 metres from load point to pit exit.

6.0 Equipment selection could vary from that indicated in your document.

We trust the above assists in your study and would be pleased to supply further information as required.

Yours faithfully

SMclay.

S. McLay
TTS CONSTRUCTION.

Head Office: 35-53 Morehead Street, Townsville
Address all correspondence to Box 5422, TMC, Townsville, QLD. 4810
Telephone (077) 222 777 Fax (077) 211 387
Telex 47012 Telegraphic Address "Transerve"

TTS CONSTRUCTION

An Operating Division of TTS TRANSPORT PTY. LTD.

MT. TODD GOLD PROJECT

SCHEDULE OF PRICES REQUESTED

ITEM	DESCRIPTION		UNIT	RATE
1	OVERBURDEN			
	(a)	Rip and Push overburden	m^3	0.90
	(b)	Drill and blast overburden	m^3	1.45
	(c)	Load and haul overburden to dump 500 metres away from the pit exit	m^3	2.50
	(d)	Load and haul overburden to dump 1000 metres away from pit exit	m^3	2.85
	(e)	Load and haul overburden to dump 2000 metres away from pit exit	m^3	4.30
2	ORE			
	(a)	Drill and blast ore	t	0.75
	(b)	Load ore and haul to ROM stockpile 1000 metres away	t	1.55
	(c)	Load crushed ore and haul to pads 500 metres away	t	0.90
3	ADMINISTRATION COSTS			
	(a)	Mobilisation		$150,000.00
	(b)	Demobilisation		$ 70,000.00
	(c)	Monthly Accommodation Cost (not to be included in above rates)		$ 18,000.00/mth

APPENDIX 3

MTDC2 MT TODD GOLD PROJECT - CIP/CIL STUDY - DESIGN CRITERIA

CATEGORY		UNITS	DATA		
SCHEDULES					
MINING	TONNES PA	TONNES	. 2000000 ORE		
	WEEKS PA		52		
	DAYS PW		6		
	SHIFTS PD		2		
	HOURS PS		10		
	TONNES PW	TONNES	38462	c	
	TONNES PD	TONNES	6410	c	
CRUSHING	TONNES PW	TONNES	38462	c	
	DAYS PW		6		
	SHIFTS PD		2		
	HOURS PS		10		
	% AVAILBY		80		
DESIGN	T.P.H.	TONNES/HR	401	c	
MILLING	TONNES PW	TONNES	38462	c	
	DAYS PW		7		
	SHIFTS PD		2		
	HOURS PS		12		
	% AVAILBY		92.50		
NOMINAL	T.P.H.	TONNES/HR	248	c	
DESIGN T.P.H.		TONNES/HR	250		
STRIPPING	DAYS PW		7		
	STRIPS PD		1		
DESIGN BATCH SIZE		TONNES C	3.31 PER STRIPc		
BULLION	Kg Au PA	Kg	2270	c	
	Oz Au PA	Oz	72995	c	
ORE CHARACTERISTIC					
GRADE	Au	g/T	1.29		
	Ag	g/T	1.50		
	S.G.(DRY)		2.48 W	DIST. %	14
			2.67 T		22
			2.78 P		64
	S.G. AV		2.71		
	MOISTURE	%	******** check		
	BULK SG		1.45 W		
			1.48 T		
			1.48 P		
	U.C.S.	MPa	24.00 W		
			29.10 T		
			57.10 P		
	IMPACT	kWh/T	4.10 W		

CRUSHER		2.60 T	
INDEX		7.80 P	
ABRASION		0.0180 W	
INDEX		0.0046 T	
		0.0660 P	
ANGLE OF	DEGREES	37.00 (FINE)	
REPOSE			
FREE SiO2	%	********	
BOND B.M. W.I.	kWh/T	11.70 W	
		17.20 T	
		24.70 P	
R.M. W.I.	kWh/T	******* USE B.M.	
CRUSHING			
DESIGN T.P.H.	TONNES/HR	401	c
R.O.M. SIZE d100	m.m.	1000	
TRUCK SIZE	TONNES	85	
R.O.M.BIN CAPACITY	TONNES	170	c
C.O.S. CAPACITY L	HOURS	24	
	TONNES	9615	c
DESIGN PRODUCT d80 SIZE	m.m.	200	
F.O.S. CAPACITY L	HOURS	48	
	TONNES	11880	c
GRINDING CIRCUIT			
DESIGN T.P.H.	TONNES/HR	250	c
FEED d80 SIZE	m.m.	200	
PRODUCT d80 SIZE	m.m.	0.075	
RECIRCULATING LOAD		UF/OF	
SAG MILL	%	200	
BALL MILL	%	150	
CLASSIFICATION			
SAG MILL PRODUCT			
DENSITY	%SOL. WT.	45	
PULP SG	T/M3	1.40	
	%SOL. VOL	23	
TPH PULP	TONNES/HR	556	
VOL FLOW	M3/HR	398	
BALL MILL PRODUCT			
DENSITY	%SOL. WT.	42	
PULP SG	T/M3	1.36	c

	%SOL. VOL	21	c	
TPH PULP	TONNES/HR	595	c	
VOL FLOW	M3/HR	437	c	

LEACHING/ADSORPTION				
DESIGN T.P.H.	TONNES/HR	250	c	
DENSITY	%SOL. WT.	42		
SOLIDS FLOWRATE	M3/HR	437	c	
SOLUT.N FLOWRATE	M3/HR	345		
SOL.N FEED GRADE	g/T	0.82		
TRASH SCREEN				
APERTURE	MICRONS	1000		
C.I.P. HYBRID				
LEACH TIME TOTAL	HR	43.00		
ADSORB	HR	5 FROM MODEL		
TOTAL	HR	48.00	c	
NO. OF STAGES TOTAL		13		
LEACH		8		
ADSORB		5		
TANK VOL TOTAL	M3	20993 (LIVE)	c	
LEACH	M3	18806	c	
ADSORB	M3	2187	c	
INTERTANK SCREEN		AIRSWEPT,FLOODED,W'WIRE		
APERTURE	MICRONS	833		
LEACH RECOVERY				
Au	%	88 check		
Ag	%	40 " "		
CARBON TYPE		PICA G210A5 (AMDEL)		
SIZE	ASTM	6X12		
DESIGN TAIL GRADE Au	g/T	0.15		
DESIGN SOL.N TAIL Au	g/T	0.01		
DESIGN CARBON				
Au LOAD	g/T	1935 c		
Ag "	g/T	*******		
STRIPPED				
Au "	g/T	50		
Ag "	g/T	*********		
CONCNTN	Kg/M3	14.18	c	
MASSFLOW	Kg/HR	140.83	c	
TOTAL INVENTORY	Kg	31000 ADSORB	FROM MODEL	
STAGE EFFICIENCY	%	61	" "	
GOLD INVENTORY	Kg	28	" "	
RES.TIME PER STAGE	HR	44.02	c	
TRANSFER		CONT.S AIRLIFT		
REMOVAL		RECESSED IMP. PUMP		
RECOVERY SCREEN				

APERTURE	MICRONS	800	
THICKENING			
DESIGN T.P.H.	TONNES/HR	250	c
F DENSITY	%SOL. WT.	39	
PULP S.G.	T/M3	1.33	c
	%SOL. VOL	19	c
TPH PULP	TONNES/HR	641	c
VOL FLOW	M3/HR	483	c
AV SETTLING RATE			
COMPRESSN	M/HR		
BULK	M/HR		
SOL UPFLO	M/HR		
MAX %SOL	%SOL. WT.		
DESIGN %SOL	%SOL. WT.		
FLOCC USE	g/T	50 MAX	35 MIN
DESORPTION/ELECTROWIN/REFINE			
STRIPPING SYSTEM		AARL,COLD ACID WASH	
CARBON BATCH SIZE	TONNE C	3.31	c
NO. OF STRIPS PW		7	
STRIPS PD		1	
METAL STRIPPED PW			
Au	Kg	43.66	c
Ag	Kg	23.08	
TOTAL	Kg	67	c
ELECTROWIN			
NO. CELLS		1	
NO. CATHS		9	
CURRENT	AMPS	700	
BARREN ELUATE	PPM	10 MAX	
ELECTROREFINE			
NO. CELLS		1	
NO. CATHS		10	
CURRENT	AMPS	500	
CARBON REGEN. RATE	Kg/HR	250	
REAGENT USAGE			
C.I.P.			
NaCN	Kg/T	0.80	
LIME	Kg/T	1.10	
NaOH	Kg/T	0.20	
Pb(NO)3	Kg/T	0.40 MAX	
FLOCC	Kg/T	0.05	

CARBON	T/YR	31.00	
G MEDIA	Kg/T	1.50	
DESORB			
NaCN	Kg/T C	25.00	2% W/V
NaOH	Kg/T C	20.00	2% W/V
HCl	Kg/T C	85.00	3% W/V

WATER BALANCE			
GRIND PRODUCT			
T.P.H.	TONNES/HR	250	c
DENSITY	%SOL. WT.	42	
PULP SG	T/M3	1.36	c
	%SOL VOL	21	c
PULP TPH	TONNES/HR	595	c
VOL FLOW	M3/HR	437	c
WATER TPH	TONNES/HR	345	
TAIL			
DENSITY	%SOL. WT.	39	
WATER	TONNES/HR	391	c
THICKENER U/F			
DENSITY	%SOL. WT.	60	
WATER LOSS	TONNES/HR	167	c
RECLAIM	TONNES/HR	25	15 %REC F
TOTAL WATER LOSS	TONNES/HR	142	
RETURN	TONNES/HR	249	c
(TO GRINDING)			
GRIND MAKEUP	TONNES/HR	96	c
DESORB USE	TONNES/HR	46	c
OTHER	TONNES/HR	2 check	
TOTAL	TONNES/HR	144	c
	TPD	3448	c
	TPW	24136	c
	TPA	1255072	c
TAILINGS DAM			
TONNES		20000000	C
MINE LIFE	YEARS	10	
DENSITY	%SOL. WT.	77 check	
SG TAIL	T/M3	1.95	C
DAM VOLUME	M3	13343766	C

MT TODD GOLD PROJECT CIP/CIL WET SAG OPTION

	SAG	BM	BM
ANNUAL THROUGHPUT	2000000	1000000	1000000
OPERATING HOURS	8000	8000	8000
Tonnes per hour	250.0	125.0	125.0
MILL TYPE: ROD [R],BALL [B] ?	B	B	B
ROD MILL WORK INDEX	36.96	24	24
BALL MILL WORK INDEX	33.6	24	24
FEED SIZE ,F80(um)	200000	3000	3000
PRODUCT SIZE ,F80(um)	2000	75	75
MILL DIAM. INSIDE LINERS (m)	5.3	4.5	4.5
(ft)	17.4	14.8	14.8
MILL LENGTH (m)	6.87	8.31	8.31
MILL POWER (kW)	2065	3104	3104
WET GRINDING [Y]/N ?	Y	Y	Y
OPEN CIRCUIT Y/[N] ?	N	N	N
STEEL/RUBBER LINERS S/[R] ?	S	R	R
NEW/WORN LINERS N/[W] ?	N	N	N
GRATE DISCHARGE Y/[N] ?	Y	N	N
Charge (fraction of volume)	0.17	0.43	0.43
Critical Speed %	74	74	74
Solids S.G	2.7	2.7	2.7
EFFICIENCY FACTORS			
EF1 Dry Grinding	1.00	1.00	1.00
EF2 Open Circuit Milling	1.00	1.00	1.00
EF3 Diameter Efficiency	0.91	0.91	0.91
EF4 Oversized Feed	23.16	1.01	1.01
EF5 Ball Mill Fine Grinding	1.00	1.00	1.00
EF6 H/L Rr. Rod Milling	1.00	1.00	1.00
EF7 Low Rr. Ball Milling	1.00	1.00	1.00
EF8	1.00	1.00	1.00
EF9 Rubber Liners	1.00	1.10	1.10
EF10 Worn Liners	1.00	1.00	1.00
EF11 Grate Discharge	1.16	1.00	1.00
EF TOTAL	1.06	1.01	1.01
Reduction Ratio (Rr)	100	40	40
Optimum Feed Size (Fo) :			
Rod Mill (um)	9489	11776	11776
Ball Mill (um)	2372	2944	2944
Basic kw/tonne	7.42	23.33	23.33
Basic Grinding Power (kW)	1855.0	2916.4	2916.4
Total Efficiency Factor EFT	1.06	1.01	1.01
Total Power Required (kW)	1967	2956	2956
(at Pinion) (HP)	2636	3962	3962
Transmission Losses %	5	5	5
Total Power Required (kW)	2065	3104	3104
(at Motor) (HP)	2768	4160	4160
BALL SIZE (mm)	120	90	90 USING ARMCO CALC
SLUMP FACTOR	1.17	0.73	0.73
Power/Tonne Balls	16.70	11.20	11.20
Weight of balls (tonnes)	117.77	263.84	263.84
Required Length of Mill (m)	6.87	8.31	8.31
Required Length of Mill(ft)	22.5	27.3	27.3

ball size calculation (ARM

INPUTS	G100	MICRONS	3000
	WI	kWHRS/T	25
	%CRIT SPD	%	.74
	MILL D	M	4.5
	%R.L.	UF/OF	150
OUTPUTS			
	RPM CRIT		19.93947
	MAX BALL	M.M.	82.26662

STREAM TPH SOLS TPH WATER % SOLIDS PULP SG TPH PULP M3/HR PULP

ENTER SG SOLIDS 2.71 %RL = 200.00

COF %SOL 45.00 TPH 250.00

% MOIST 1.00 CUF WATER 0.40 Min 0.2

SPLIT

SAG MILL MT TODD GOLD PROJECT

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	%Sol Vol
NEW FEED	250.00	2.53	99.00	2.66	252.53	94.78	97.34
WATER ADD		2.00	0.00	1.00	2.00	2.00	0.00
MILL FEED	750.00	208.28	78.27	1.98	958.28	485.03	57.06
MILL DISC	750.00	208.28	78.27	1.98	958.28	485.03	57.06
WATER ADD		301.10	0.00	1.00	301.10	301.10	0.00
SPILLAGE	0.06	0.00	100.00	2.71	0.06	0.02	100.00
CYC FEED	750.06	509.38	59.55	1.60	1259.44	786.16	35.21
CYC U/F	500.00	203.75	71.05	1.81	703.75	388.25	47.52
CYC O/F	250.06	305.63	45.00	1.40	555.69	397.90	23.19

C1 4.76 FEED kPa 60

C2 1.04 ROPE LIM 61.80 %SOLv CUF

C3 0.98 VF D mm 937

d50c DES 500 MICRONS INLET Dmm 662 MAX

CYC DIAM 2342 mm MAX D 574 MIN

Q/CYC 3986.97 M3/HR

NO. CYC.S 0.20

APEX D 185 mm LOW 234 mm

HIGH 820 mm

			feed kPa								
nominate lower sizes		mm	40	60	80	100	120	140	160	180	200
read no. of	1	1270	0.8	0.7	0.6	0.5	0.5	0.4	0.4	0.4	0.4
cyclones at	2	1050	1.2	1.0	0.8	0.8	0.7	0.6	0.6	0.6	0.5
feed kPa	3	900	1.6	1.3	1.2	1.0	0.9	0.9	0.8	0.8	0.7
	4	750	2.4	1.9	1.7	1.5	1.4	1.3	1.2	1.1	1.1
	5	660	3.0	2.5	2.1	1.9	1.8	1.6	1.5	1.4	1.4
	6	151	58.1	47.4	41.1	36.7	33.5	31.0	29.0	27.4	26.0

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	

ENTER SG SOLIDS 2.71 %RL = 200.00
COF %SOL 40.00 TPH 250.00
% MOIST 1.00 CUF WATER SPLIT 0.40 Min 0.2 %Sol Vol

SAG MILL MT TODD GOLD PROJECT

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	%Sol Vol
NEW FEED	250.00	2.53	99.00	2.66	252.53	94.78	97.34
WATER ADD		2.00	0.00	1.00	2.00	2.00	0.00
MILL FEED	750.00	254.59	74.66	1.89	1004.59	531.34	52.09
MILL DISC	750.00	254.59	74.66	1.89	1004.59	531.34	52.09
WATER ADD		370.56	0.00	1.00	370.56	370.56	0.00
SPILLAGE	0.06	0.00	100.00	2.71	0.06	0.02	100.00
CYC FEED	750.06	625.15	54.54	1.52	1375.21	901.92	30.69
CYC U/F	500.00	250.06	66.66	1.73	750.06	434.56	42.46
CYC O/F	250.06	375.09	40.00	1.34	625.15	467.36	19.74

C1 3.45 FEED kPa 60
C2 1.04 ROPE LIM 59.94 %SOLv CUF
C3 0.98 VF D mm 1532
d50c DES 500 MICRONS INLET Dmm 1083 MAX
CYC DIAM 3830 mm MAX D 938 MIN
Q/CYC 10663.57 M3/HR
NO. CYC.S 0.08
APEX D 209 mm LOW 383 mm
HIGH 1340 mm

nominate fewer sizes read no. of cyclones at feed kPa		mm	feed kPa 40	60	80	100	120	140	160	180	200
	1	1270	0.9	0.8	0.7	0.6	0.5	0.5	0.5	0.4	0.4
	2	1050	1.4	1.1	1.0	0.9	0.8	0.7	0.7	0.6	0.6
	3	900	1.9	1.5	1.3	1.2	1.1	1.0	0.9	0.9	0.8
	4	750	2.7	2.2	1.9	1.7	1.6	1.4	1.4	1.3	1.2
	5	660	3.5	2.8	2.5	2.2	2.0	1.9	1.7	1.6	1.6
	6	151	66.6	54.4	47.1	42.1	38.5	35.6	33.3	31.4	29.8

STREAM	TPH SOLS	TPH WATER	% SOLIDS	PULP SG	TPH PULP	M3/HR PULP	
ENTER	SG SOLIDS	2.71		%RL =	150.00		
	COF %SOL	42.00		TPH	125.00		
	% MOIST	45.00		CUF WATER SPLIT	0.35	Min 0.2	%Sol Vol
REVERSE CIRCUIT							
NEW FEED	125.00	102.27	55.00	1.53	227.27	148.40	31.08
WATER ADD		2.00	0.00	1.00	2.00	2.00	0.00
MILL FEED	187.50	94.99	66.37	1.72	282.49	164.18	42.14
MILL DISC	187.50	94.99	66.37	1.72	282.49	164.18	42.14
WATER ADD		68.43	0.00	1.00	68.43	68.43	0.00
SPILLAGE	0.06	0.00	100.00	2.71	0.06	0.02	100.00
CYC FEED	312.56	265.70	54.05	1.52	578.26	381.03	30.27
CYC U/F	187.50	92.99	66.85	1.73	280.49	162.18	42.66
CYC O/F	125.06	172.70	42.00	1.36	297.76	218.85	21.09

C1	3.36		FEED kPa	180	
C2	0.76		ROPE LIM	60.67	%SOLv CUF
C3	0.98		VF D mm	101	
d50c DES	60	MICRONS	INLET Dmm	72	MAX
CYC DIAM	254	mm		.62	MIN
Q/CYC	80.97	M3/HR			
NO. CYC.S	4.71				
APEX D	69	mm	LOW	25	mm
			HIGH	89	mm

			feed kPa								
nominate lower sizes		mm	40	60	80	100	120	140	160	180	200
read no. of	1	760	1.1	0.9	0.8	0.7	0.6	0.6	0.6	0.5	0.5
cyclones at	2	660	1.5	1.2	1.0	0.9	0.9	0.8	0.7	0.7	0.7
feed kPa	3	508	2.5	2.0	1.8	1.6	1.4	1.3	1.2	1.2	1.1
	4	305	6.9	5.6	4.9	4.4	4.0	3.7	3.5	3.3	3.1
	5	254	9.9	8.1	7.0	6.3	5.7	5.3	5.0	4.7	4.4
	6	151	28.2	23.0	19.9	17.8	16.3	15.0	14.1	13.3	12.6

APPENDIX 4

ORETYPE:D18J

EACH ORE TYPE - ONE OR TWO SAMPLES

1. **GRIND**

106 microns d80	)	
75	)	stainless steel mill & media
53	)	
44	)	

Procedure

Agitation Leach
- 48 hours
- 2, 6, 12, 18, 24, 32, 40, 48.
- aerate only if indicated by O_2 uptake test.
- 0.07% CN (700 mg/l)
- 40% solids by weight with compensation for liquor extractions.
- pH 10.5 - 11.0
- monitor
  - D.O. *) especially for
  - CN *) transition
  - pH & temperature
- Assay
  - Au)
  - Ag)
  - Cu) feed, solution &
  - Bi) residue
  - Pb)
  - Zn)
  - Fe)

2. **REPRODUCE**
- 4 tests at selected size
- procedure as in 1.

3. **BATCH LEACH TEST**
- 2 or 3 tests for model.
- procedure attached.

4. **PREG-ROBBING**

IF REQUIRED
- Preaeration effect) 2 tests each
- Lead nitrate)

5. **OXYGEN UPTAKE**
- (Pending results of phase 1 of CIP testwork).
- Procedure already supplied and result will determine preaeration time if required (D.O. versus time) and technique (ie. sparge, aerator, turbine or H_2O_2).
- Au in solution over time compared with D.O. over time.
- Stress gentle agitation, as surface area to volume ratio is large on small scale test.

6. **ADSORPTION**

- sighter tests to determine carbon concentration required (1, 3, 5, 7 g/l) by stage efficiency.
- batch sequential tests - 3 stages at 3-4 litres/stage. Full analysis of carbon (ie. Au, Ag, Cu, Bi, Zn, Pb, Fe, Ca, Mg) and 24 hour duration.
- example reports attached.

The features of the test data which require investigation are:

1. Grind size - 75 or 53 micron?
2. High cyanide consumption, particularly where initial leaching rate reduced with transition ore. Have no evidence so far to rule out oxygen deficiency for this effect as well.
3. Preg robbing - reprecipitation of Gold seen in some tests.
4. Carbon concentration and Gold loading. (so far carbon concentration too high, which effects loadings).
5. Aeration requirement.

7. **CIL TEST**

Agitation leach with carbon in pulp at the 2 hour point. Procedure as in 1.

NOTE

The order of testwork may require rearrangement if O_2 uptake tests show any anomaly. If so, a series of aeration requirement leach tests will be necessary, prior to 1., 2. & 3.

- no aeration
- 3 hours aeration
- full aeration
- H_2O_2 addition

Aeration can be costly, and its requirements need to be established prior to design.

3. EVALUATION OF MODEL PARAMETERS

To evaluate the model parameters, laboratory scale tests are conducted and data collected. These data are then fitted to the assumed model equations, using a least squares of error procedure, which generates the parameter values of the assumed model that most closely match the laboratory data. Note that if the assumed model is a poor fit to the data, the least squares generated parameters will be of little use.

3.1 Leach Model Parameters

The leach model (2.1) to be used in the present work is rewritten below:

$$\frac{d[Au]_o}{dt} = k_o\,([Au]_o - [Au]_{o,f})^2 \qquad (3.1)$$

After integration, and the assumption that $[Au]_{o,i} >> [Au]_{o,f}$, (3.1) gives:

$$[Au]_o = \frac{[Au]_{o,i}\,(1 + k_o\,t\,[Au]_{o,f})}{(1 + k_o\,t\,[Au]_{o,i})} \qquad (3.2)$$

Equation (3.2) forms the basis of a curve fitting program that evaluates k_o and $[Au]_{o,f}$ from a data series of leach time versus $[Au]_o$, generated from a batch leach laboratory test.

3.1.1 The Batch Leach Test

In the batch leach test, a baffled, stirred 4 litre container is used. A typical data set for a quartz ore is given in Table 3.1 (Costello, 1985).

Table 3.1 a typical data set for a batch leach test.

Pulp solids concentration	50%
Ore specific gravity	2.6
Water specific gravity	1.0
Pulp specific gravity	1.44
Pulp volume (initial)	4 l
pulp mass	5.778 kg
Ore mass	2.889 kg

Water mass	2.889 kg
Pulp volume per 100 g ore sample	138 ml
Lime addition (100% NaCN)	0.5 kg t^{-1}
Cyanide addition (100% NaCN)	0.5 kg t^{-1}
Initial cyanide concentration	0.5 g l^{-1}
Final cyanide concentration	0.3 g l^{-1}
Initial pH	9.7
Final pH	9.5 (varies according to sample)
Natural pulp pH	7.8
Oxygen concentration before chemical addition	6 mg l^{-1}
Oxygen demand	<3 mg l^{-1} h^{-1}

The reaction system is started and approximately 138 ml of sample pulp is withdrawn at 0.5, 1, 2, 4, 8, 12, 16, 20, and 24 hours. The solution is filtered and assayed for gold, pH, and NaCN. The residue is washed and dried and assayed for gold.

3.1.2 Test Results

The data set of time and solids gold concentration form part of, or can be calculated from the results of the batch leach test. These data form the input to the leach curve fitting program LCHFIT.

This program calculates the best fit of the parameters $[Au]_{o,f}$ and k_o in equation (3.2), according to a non-linear, least squares error curve fitting procedure. The program makes use of the FORTRAN subroutines presented by Pres, Flannery, Teukolsky and Vetterling (1986). These subroutines use the Levenberg-Marquardt method of least squares curve fitting.

One feature common to most of the data sets analysed is that data is not collected at even intervals through the sampling period: initially, data is collected every 2-4 hours up until t=8 hours, and often there is a gap of 16 hours before the next reading is collected at t=24 hours, which is often the final reading. The least squares program weights each data point by the same amount, therefore a large error in the 24 hour concentration will be tolerated if the 4 data points up to 8 hours are fitted with small errors.

However, the 24 hour data point is the most significant metallurgically, and therefore conflict exists between the program weighting of the data and the metallurgical weighting. This may be dealt with in two ways. Firstly, the testing laboratory may be requested to sample at regular 2 or 4 hour intervals up until the 24 hour interval, thereby giving a more even distribution of data. Secondly, the LCHFIT program can weight the test data using several different methods.

The standard analysis weights each data point the same, such that each data point is of equal importance. The second mode weighs the data according to the time that the data point was taken - that is, the 8 hour data point is eight times as important as the 1 hour data point. In the third mode, the final data point alone is weighted relative to the other data points in the set. The relative importance of this point can be set anywhere between 2 and 99 times. These weighting procedures result in much smaller errors on the weighted data points, relative to the unweighted points.

BATCH SEQUENTIAL. C/P ADSORPTION.

Test Identity
Carbon type :Haycarb YAO [6x12]
Carbon weight : 18.80 gram [4 gpl]
Slurry % Solids : 44.70 [w/w]
Solution weight : 3638.0 Equiv vol 3638.0
Slurry sample vol: 80.0 Equiv soln 61.4
Ore s.g. : 2.67
Solution s.g. : 1.00

RESULTS
==========

CYCLE	SOLUTION VOLUME mls	ELAPSED TIME (hr) cycle	ELAPSED TIME (hr) cumulative	Solution Au mg/l	Solution Ag mg/l	Solution Cu mg/l	Carbon Calc'd Load Au g/t	Carbon Load Cumul' Au g/t
1	3638	0.00	0.00	1.67	0.45	210	0	0
	3577	0.50	0.50	0.40	0.21	203	242	242
	3515	1.00	1.00	0.12	0.17	212	52	294
	3454	2.00	2.00	0.04	0.12	204	15	309
2	3638	0.00	2.00	1.67	0.45	210	0	309
	3577	0.50	2.50	0.58	0.27	212	207	516
	3515	1.00	3.00	0.23	0.21	213	65	581
	3454	2.00	4.00	0.07	0.18	218	29	611
3	3638	0.00	4.00	1.67	0.45	210	0	611
	3577	0.50	4.50	0.69	0.29	217	186	797
	3515	1.00	5.00	0.29	0.24	212	75	872
	3454	2.00	6.00	0.10	0.21	214	35	907
	3392	3.00	7.00	0.07	0.19	213	5	912
	3331	4.00	8.00	0.05	0.18	216	4	916
	3270	20.00	24.00	0.03	0.14	226	3	919

Regression Output:For Six Hours

Constant	5.712		
Std Err of Y Est	0.176	intercept b=	302.54
R Squared	0.891		
No. of Observations	9.000	coeff r=	0.944
Degrees of Freedom	7.000		

X Coefficient(s) 0.576 Fleming n= 0.576
Std Err of Coef. 0.076

Fleming k= 241.1

a185

EXAMPLE

SIGHTER TESTS.

CARBON CONCENTRATION	2 gpl		4 gpl		8 gpl		16 gpl	
Elapsed time hrs	Soln Au g/t	%Au Loaded	Soln Au g/t	%Au Loaded	Soln Au g/t	%Au Loaded	Soln Au g/t	%Au Loaded
0	0.64	0.0	0.64	0.0	0.64	0.0	0.64	0.0
0.5	0.39	40.8	0.24	63.6	0.07	89.4	0.00	100.0
1	0.28	58.3	0.15	77.6	0.02	97.0	0.00	100.0
2	0.20	70.8	0.05	92.7	0.00	100.0	0.00	100.0
3	0.12	82.8	0.02	97.2	0.00	100.0	0.00	100.0
4	0.10	85.9	0.02	97.2	0.00	100.0	0.00	100.0
5	0.08	88.8	0.00	100.0	0.00	100.0	0.00	100.0
6	0.06	91.7	0.00	100.0	0.00	100.0	0.00	100.0
24	0.02	97.3	0.00	100.0	0.00	100.0	0.00	100.0

Regression analysis for kinetics of gold loading.

time	6 Hours	6 Hours (three)	2 Hours	6 Hours
FLEMING k	256	293 (214)	162	172
FLEMING n	0.33	0.19 (0.24)	0.08	0.00

Loaded Carbon				
Au g/t	232.0	121.4	71.6	39.8

Golder Associates
GEOTECHNICAL ENGINEERS

MT TODD GOLD PROJECT

MINE INFRASTRUCTURE

SITE INVESTIGATION

88638252(C) JANUARY 1989

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TABLE OF CONTENTS

1. INTRODUCTION

Golder Associates were requested by Minproc Engineers Pty Ltd, acting on behalf of Billiton Australia Gold Pty Ltd, to carry out site investigation work for the mine infrastructure development, incorporating the following items:

- **Crushing Plant**

  \- Two large crushers with associated screens, agglomeration drum, and dump hopper. Administration buildings and workshop buildings are proposed immediately south of the crusher line. An absorption/desorption plant is also proposed in this area with unit foundation loads of the order of 100kn.

- **Leaching Stockpiles**

  \- Stockpiles on the initial leach pads (Nos. 1 & 2) are to be constructed in two lifts to a height of 12m. Future leach pads (Nos. 3, 4 and 5) will be constructed on the alluvial flood plain approximately 1km east of the proposed crusher line and leach pads 1 & 2.

- **Process Ponds**

  \- Two fully lined process ponds are proposed south of leach pad No. 2. The ponds will have 5m high embankments.

- **Raw Water Dam**

  \- A 10m high earth embankment is proposed, located approximately 3km north-west of the crusher line.

- **Stow Creek Crossing**

  \- A new access road is proposed off Edith Falls Road into the site, and a culvert crossing will be required at Stow Creek.

Investigation work was carried out by excavation of 21 test pits on 5 and 6 December, 1988. This report sets out details of field work and laboratory testing together with engineering comment and geotechnical design parameters.

2. THE SITE AND LOCAL GEOLOGY

The Mt Todd gold deposits (named Batman and Robin) are located approximately 50km north-east of Katherine. The main deposit (Batman) is located within a group of hills rising 60m above an alluvial flood plain to the east. The proposed crusher line is located at the south-eastern base of these hills with leach pad Nos. 1 & 2 on slightly sloping ground covered with light scrub and medium height trees.

The proposed area of leach pads 3, 4, and 5 is a flat flood plain covered with sparse vegetation and low to medium height trees.

The proposed raw water dam is located within a steep sided gully which was boulder strewn but dry at the time of investigation.

At the location where the proposed access road crosses Stow Creek the creek banks are very steep and rise 2 to 3m above water level. The area to be traversed by the road is generally flat and covered with low to medium height trees.

Information supplied by on-site geologists indicates the whole area to be underlain by greywacke rock, which is generally medium strong to strong and close to surface level at the base of the Batman hill gold deposit.

3. FIELD WORK

The location of the various sites are shown on the site locality plan, Fig. 1. A large excavator (Kobelco K912A) was used to excavate 21 test pits. The location of 18 of these test pits are given on Fig. 2. Test pits 19 and 20 were excavated on the banks of Stow Creek at the proposed crossing site and Test Pit 21 was located 200m along the proposed access road to the east of Stow Creek.

All test pits were logged by an experienced geotechnical engineer and samples taken for laboratory testing.

4. GROUND CONDITIONS

Detailed descriptions of the strata encountered are given in the attached Test Pit Reports using terminology defined in appended notes. The ground conditions are summarised for the various sites as follows:

- **Crusher Plant/Administration and Workshop Buildings**

  - Uniform ground conditions were encountered at the base of the hills comprised of medium dense and dense silty sands and gravels to a maximum depth of 1.0m overlying weak to medium strong rock. The rock was generally fragmented with clean tight joints.

- **Leach Pads Nos. 1 & 2 / Process Ponds**

  - Ground conditions did not vary markedly from above, with the silty sand/gravel strata extending to a maximum depth of 0.8m. A layer of clayey gravel was encountered in Test Pit 13 between 0.6m and 1.2m. The underlying rock was generally very weak to weak.

- **Leach Pads 3, 4 and 5**

  - Although located on an alluvial flood plain, the alluvial materials (silty sand and gravel) overlying rock were still medium dense to dense and extended to a maximum depth of only 1.7m in Test Pit 16. The underlying rock ranged in strength from very weak to medium strong.

- **Raw Water Dam**

  - The soil overlying rock in this area was more cohesive than in the general plant area and is described as rock fragments in a clay matrix. This stratum extended to a maximum depth of 2.5m in Test Pit 5, located some 500m west of the proposed dam embankment in the floor of the gully. On the sides of the gully (Test Pits 2 and 3) rock was encountered at approximately 0.5m. The rock was generally medium strong to strong with clean tight joints and apparently of low permeability. A visual assessment of moisture content profile down the sides of the test pits indicated that water flowing through the gully does not penetrate below 0.5m.

- **Stow Creek Crossing/Access Road**

- Test Pit 19 on the western bank of Stow Creek encountered stiff alluvial clays and silts to 2.9m overlying very dense sands and gravels to the terminal depth of 5m. In contrast Test Pit 20 on the eastern bank encountered silty clay to only 0.6m overlying rock. Test Pit 21 encountered stiff alluvial clays and silts to the terminal depth of 2.5m. It is most probable that the creek has cut a number of channels in the area through the rock and the old channels have been filled with the cohesive alluvial deposits.

5. LABORATORY TESTING

Particle size distribution tests were carried out on samples of the silty sand/gravel encountered in the majority of the test pits and on a sample of clayey silt taken from the western bank of Stow Creek. Atterberg limits determinations were also carried out on the latter sample. The results are summarised on an attached sheet and grading curves are appended.

The silty sand/gravel contained up to 60% gravel sized particles with 20% sand and 20% silt. The cohesive alluvial deposits on the western bank of Stow Creek were comprised predominantly of low plasticity silt sized particles with 20% fine grained sand.

6. ENGINEERING COMMENT

6.1 Crushing Plant/Administration and Workshop Buildings

Foundations to all plant items along the proposed crusher line could be formed either in the medium dense to dense silty sand/gravel strata; or on the underlying weak to medium strong rock using allowable bearing pressures of 300kPa and 1000kPa respectively. The 1000kPa value is a conservative figure based on weak rock and could be increased if necessary for stronger rock. The rock is fragmented and therefore should be relatively easy to excavate if working from an open face. In a confined excavation (eg. service trench) the rock will be difficult to excavate, as indicated during excavation of the test pits using a large excavator.

It is understood that the administration and workshop buildings, and the absorption/desorption plant will be relatively flexible structures imposing low foundation loads. Strip or pad footings to these buildings could be formed at shallow depth in the medium dense to dense silty sand/gravel stratum using an allowable bearing pressure of 300kPa. Weak to medium strong rock would be encountered at shallow depth (about 1.0m) underlying the buildings and the comments given above on excavation conditions would apply.

6.2 Leach Pad Nos. 1 & 2 / Process Ponds

Some cut to fill earthworks will be necessary in order to form a level pad for the ore stockpiles. The silty sand and gravel layer will be readily excavated and would be suitable for re-use as filling. A compaction level equivalent to 90% Modified compaction is recommended. Fill material should be placed in loose layers no greater than 300mm thick prior to compaction.

The proposed 12m high stockpiles would induce negligible settlement in the natural or compacted foundation materials, and could be formed as steep as practical for leaching purposes, without threatening the stability of foundation materials.

It is understood that excavation depths for the process ponds will be up to 3m. Rock is anticipated below approximately 1m. As noted under Section 6.1, the rock is fragmented and should be readily excavated when working from an open face, using a large excavator or a bulldozer (say up to D6 in size) to rip the rock. In a confined excavation the rock could be very difficult to excavate; possibly requiring the use of air tools or blasting. Excavated rock to be used as general filling either to form leach pads or process pond embankments should be broken down to exclude fragments greater than 200mm.

Embankments to the fully lined process ponds could be formed using the near surface silty sands and gravel (mixed with broken down rock if necessary), compacted to a dry density ratio equal to 90% of Modified compaction. For a 5m high embankment, a battered slope of 1.5 horizontal to 1 vertical is recommended. Embankments formed from silty sand and gravel would be subject to erosion and hence external slopes should be protected. The fragmented greywacke encountered generally below 1.0m would be adequate for this purpose.

6.3 Leach Pads 3, 4, and 5

Similar ground conditions were encountered under proposed leach pads 3, 4 and 5 as encountered under leach pad 2, hence the comments given under section 6.2 would apply.

6.4 Raw Water Dam

The overburden soil encountered in the floor of the gully was comprised of rock fragments in a clay matrix. This material, when compacted in layers to a dry density ratio equal to 95% of Standard compaction (this dictates a slightly higher placement moisture content than Modified compaction), would form a low permeability earth embankment. It will be necessary to determine the extent, and hence available quantity of cohesive material within the gully floor area prior to final embankment design.

Prior to commencement of embankment construction, the embankment foundation area should be stripped of all vegetation and loose boulders, and then proof rolled to remove any soft zones. For a 10m high embankment, battered side slopes of 1.5 horizontal to 1 vertical are recommended. It will be necessary to protect embankment slopes against erosion, preferably by the use of vegetation on external slopes.

The in-situ soil and underlying rock have been visually assessed as being of very low permeability. Hence only minor seepage would be anticipated through the dam floor and embankment wall provided care is taken to use predominantly clayey material in embankment construction; excluding rock fragments and boulders larger than 200mm. In order to quantify the anticipated seepage volumes it would be necessary to establish permeability values for the in-situ fragmented rock and the clay embankment. This facet is important if the plant process is dependent on water supply from the dam.

6.5 Stow Creek Crossing/Access Road

It is understood that the Stow Creek crossing will be formed by placing a number of large culverts in the existing creek bed and covering with soil won locally.

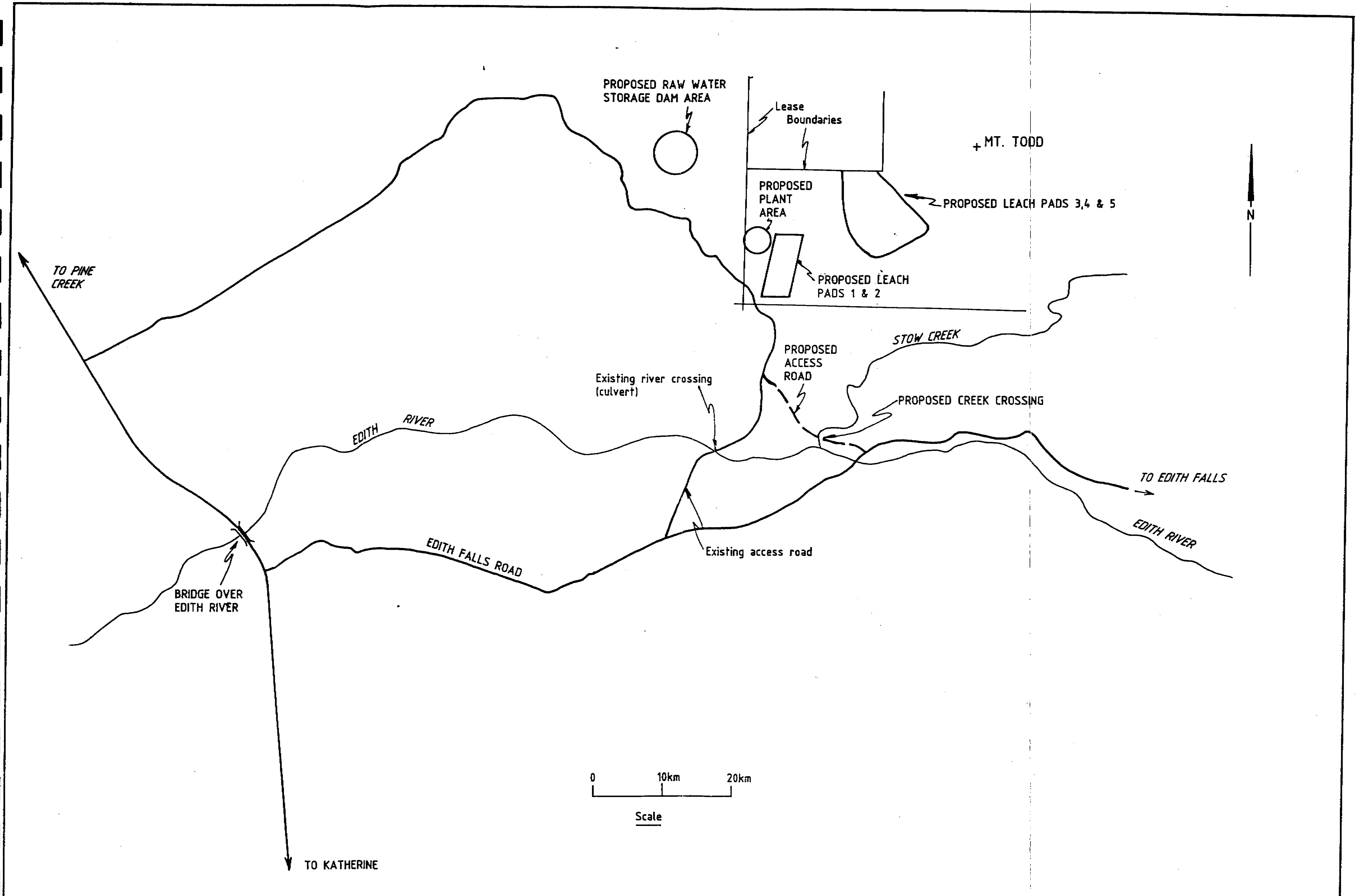
The clayey silt encountered on the western bank of Stow Creek would be highly erodible and hence this material should not be used to fill over culverts. There was evidence of a number of small gravel pits in the area to the east of Stow Creek and it is recommended that either gravel from these pits, or silty sands and gravels such as encountered in the leach pad areas be used for this purpose. (Naturally these materials should be placed in compacted layers). In addition, large rock rip-rap should be placed upstream of the culverts to protect against current erosion during high velocity flow periods.

The worst material likely to be encountered as subgrade material for the proposed access road is the clayey silt/silty clay such as encountered in Test Pits 19 to 21. Based on the laboratory testing carried out, a CBR value of 16% is indicated for this material. It is noted that the material has a high silt content and hence would be impossible to compact when wetted above optimum moisture content. Adequate drainage should be installed along the sides of the road to protect against erosion of the subgrade.

GOLDER ASSOCIATES PTY LTD

per :

David K. Nolan



GOLDER ASSOCIATES CONSULTING GEOTECHNICAL AND MINING ENGINEERS	MT. TODD GOLD PROJECT SITE LOCALITY PLAN	Job No 88638252 Scale AS SHOWN Date DEC. 1988	Fig. 1

APPENDIX

TEST PIT REPORT SHEETS

DESCRIPTION OF TERMS USED IN CLASSIFICATION

LABORATORY TEST RESULTS

TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE PROPOSED RAW WATER DAM
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 1, 2 and 3
DATE 5th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
1	CLAY AND ROCK FRAGMENTS Strong to very strong rock fragments (up to 400mm) in a brown clay matrix	GL			
	TEST PIT TERMINATED AT 1.0m (unable to penetrate)	1.0			
2	CLAY AND ROCK FRAGMENTS Medium strong red rock fragments (up to 300mm) in a stiff clay matrix	GL			
	ROCK (Greywacke) Medium strong, highly fractured rock with tight clean joints - grading strong (highly fractured) below 2.0m approximately	0.6	D	1.0	
	TEST PIT TERMINATED AT 2.5m (unable to penetrate)	2.5			
3	CLAY AND ROCK FRAGMENTS Weak to medium strong rock fragments (up to 300mm) in a stiff red clay matrix	GL			
	ROCK (Greywacke) Medium strong, red, highly fractured - grading strong to very strong, highly fractured (tight, clean joints) below 0.9m	0.5			
	TEST PIT TERMINATED AT 1.5m (unable to penetrate)	1.5			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	TEST PIT NO.	4, and 5
SITE	PROPOSED RAW WATER DAM	DATE	5th December, 1988
LOCATION	MT TODD GOLD PROJECT	PROJECT NO.	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
4	<u>CLAY AND ROCK FRAGMENTS</u> Weak grey rock fragments in a stiff clay matrix	GL			
	<u>ROCK</u> (Greywacke) Medium strong, grey, highly fractured clean tight joints - ironstained in places	0.5			
	<u>TEST PIT TERMINATED AT 2.5m</u> (near pentration refusal) Note: water not penetrating below 0.5m - very damp above 0.5m then dry	2.5			
5	<u>CLAY AND ROCK FRAGMENTS</u> Alluvial gravel (100-200mm - rounded) in a red clay matrix - tight and difficult to excavate	GL			
	<u>CLAY AND ROCK FRAGMENTS</u> - as above with colour change to grey	1.5			
	<u>ROCK</u> (Greywacke) Strong, grey highly fractured, clean tight joints	2.2			
	<u>TEST PIT TERMINATED AT 2.8m</u> (very difficult to penetrate)	2.8			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE PROPOSED CRUSHER LINE
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 6
DATE 5th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
6	SILTY SAND/GRAVEL Apparent medium dense to dense, brown	GL			
	ROCK (Greywacke) Weak grey, highly fractured (clean tight joints)	0.8			
	ROCK (Greywacke) Weak to medium strong, grey, highly fractured (clean tight joints)	2.5			
	TEST PIT DISCONTINUED AT 4.5m (very difficult to penetrate)	4.5			

EQUIPMENT Excavator Kobelco K912A
SUPERVISOR D Nolan
GROUND WATER Not encountered
REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE PROPOSED CRUSHER LINE
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 7, 8 and 9
DATE 5th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
7	SILTY SAND/GRAVEL Apparent medium dense to dense, brown	GL			
	ROCK (Greywacke) Weak, grey, fragmented (clean tight joints) - grading weak to medium strong below 2.5m approximately	0.5			
	TEST PIT TERMINATED AT 3.5m (very difficult to dig)	3.5			
8	SILTY SAND/GRAVEL Apparent dense red brown	GL			
			D	0.50	
	ROCK (Greywacke) Weak, grey, fragmented (clean tight joints)	0.9			
			D	2.50	
	TEST PIT TERMINATED AT 3.5m (difficult to dig)	3.5			
9	SILTY SAND/GRAVEL Apparent medium dense to dense red-brown	GL			
	ROCK (Greywacke) Weak, grey fragmented (clean tight joints) - grading weak to medium strong below 2.5m approximately	1.0			
	TEST PIT TERMINATED AT 3.0m	3.0			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	10
SITE	ADMINISTRATION/WORKSHOP BUILDING	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
10	<u>SILTY SAND/GRAVEL</u> Apparent medium dense to dense brown - gravel size up to 50mm	GL			
	<u>ROCK</u> (Greywacke) Weak grey, fragmented (clean tight joints) - grading weak to medium strong below 1.8m approximately	0.8			
	<u>TEST PIT TERMINATED AT 3.0m</u> (very difficult to dig)	3.0			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD
SITE	POTENTIAL C.I.L PLANT
LOCATION	MT TODD GOLD PROJECT

TEST PIT NO.	11
DATE	6th December, 1988
PROJECT NO.	88638252
SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
11	SILTY SAND/GRAVEL Apparent medium dense to dense, red (gravel size to 100mm) - grading very dense below 0.8m	GL			
	ROCK (Greywacke) Weak to medium strong grey fragmented (clean tight joints) - grading to medium strong below 2.0m	1.4			
	TEST PIT TERMINATED AT 2.5m (unable to penetrate)	2.5			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE LEACH PAD NO. 2
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 12, 13 and 14
DATE 6th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
12	<u>SILTY SAND/GRAVEL</u> Apparent medium dense, brown (up to 50mm)	GL			
	<u>ROCK</u> (Greywacke) Weak, grey, fragmented, (clean tight joints) - grading weak to medium strong below 1.5m approximately	0.4			
	<u>TEST PIT DISCONTINUED AT 3.5m</u> (very difficult to dig)	3.5			
13	<u>SILTY SAND/GRAVEL</u> Apparent medium dense, brown	GL			
	<u>CLAYEY GRAVEL</u> Apparent medium dense to dense, red (gravel up to 50mm)	0.6			
	<u>ROCK</u> (Greywacke) Weak, grey fragmented (clean tight joints) - grading weak to medium strong below 2.5m approximately	1.2			
	<u>TEST PIT DISCONTINUED AT 3.2m</u> (difficult to dig)	3.2			
14	<u>SILTY SAND/GRAVEL</u> Apparent medium dense to dense, brown (gravel up to 50mm)	GL			
	<u>ROCK</u> (Greywacke) Very weak, grey, fragmented	0.6			
	<u>ROCK</u> (Greywacke) Weak, grey, fragmented (clean tight joints) - grading weak to medium strong at 2.6m approximately	1.3			
	<u>TEST PIT DISCONTINUED AT 3.0m</u>	3.0			

EQUIPMENT Excavator Kobelco K912A
SUPERVISOR D Nolan
GROUND WATER Not encountered
REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	15
SITE	LEACH PAD NO. 2	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
15	SILTY SAND/GRAVEL Apparent medium dense to dense, red-brown (gravel up to 50mm)	GL			
	ROCK (Greywacke) Weak to medium strong grey, highly fractured (clean tight joints) - grading medium strong below 1.5m approximately	0.8			
	TEST PIT TERMINATED AT 2.5m (unable to penetrate)	2.5			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE LEACH PADS 3 TO 5
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 16, 17 and 18
DATE 6th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
16	SILTY SAND Apparent medium dense, yellow brown - with some fine gravel below 0.5m	GL			
	SILTY SAND/GRAVEL Very dense (difficult to dig) red brown	0.8			
	ROCK (Greywacke) Medium strong, grey, highly fractured (tight clean joints)	1.7			
	TEST PIT TERMINATED AT 2.0m (unable to penetrate)	2.0			
17	SILTY SAND/GRAVEL Apparent medium dense, red-brown (gravel up to 10mm)	GL			
	ROCK (Greywacke) Very weak, grey, fragmented (clean tight joints) - grading weak to medium strong below 2.2m approximately	0.7			
	TEST PIT TERMINATED AT 2.5m	2.5			
18	SILTY SAND/GRAVEL Apparent medium dense, yellow brown (gravel up to 10mm)	GL			
	ROCK (Greywacke) Very weak, grey, fragmented (clean tight joints) - grading weak to medium strong below 1.5m approximately	0.5			
	TEST PIT TERMINATED AT 2.5m (very difficult to dig)	2.5			

EQUIPMENT Excavator Kobelco K912A
SUPERVISOR D Nolan
GROUND WATER Not encountered
REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	19 and 20
SITE	PROPOSED STOW CREEK CROSSING	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
19 West Bank	SILTY CLAY/CLAYEY SILT Stiff, brown and yellow brown - grading very stiff to hard below 0.6m - with fine sand	GL			
			D	1.0	
	SILTY SAND Very dense grey fine grained indurated bands - alluvial gravel layers (up to 100mm) below 3.8m	2.9			
	TEST PIT TERMINATED AT 5.0m	5.0			
20 East Bank	SILTY CLAY/CLAYEY SILT Stiff yellow brown, with some fine sand and gravel	GL			
	ROCK (Greywacke/Mudstone/Siltstone) Extremely to very weak grey, fragmented with ironstaining - grading weak to medium strong below 1.5m approximately	0.6			
	TEST PIT TERMINATED AT 2.5m	2.5			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	21
SITE	ACCESS ROAD - 200m EAST OF STOW CREEK	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
21	<u>SILTY CLAY/CLAYEY SILT</u> Stiff, yellow brown with a trace of gravel - grading very stiff to hard and red and yellow-brown below 0.6m - apparent cementation with some sand and alluvial gravel below 2.0m approximately	GL			
	<u>TEST PIT TERMINATED AT 2.5m</u> Shallow gravel pits on high ground in surrounding area	2.5			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

Golder Associates
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DESCRIPTION AND CLASSIFICATION OF SOIL

GENERAL

Description and classification of soil and rock are based on definitions and systems outlined in Australian Standard AS1726, SAA Site Investigation Code 1975 and its appendices A-D 1978. Description of rock fracturing is based on a system set out by the Sydney Group of the Australian Geomechanics Society, 1975.

DESCRIPTION AND CLASSIFICATION OF SOIL

Soils are classified on the basis of predominating grain size, modified by other significant grain size or sizes present (e.g. CLAYEY SAND) on the following basis:

Classification	Particle Size
CLAY	less than 0.002mm
SILT	0.002-0.06mm
SAND	
fine sand	0.06-0.2mm
medium sand	0.2-0.6mm
coarse sand	0.6-2.0mm
GRAVEL	
fine gravel	2-6mm
medium gravel	6-20mm
coarse gravel	20-60mm
COBBLES	60-200mm
BOULDERS	greater than 200mm

COHESIVE SOILS are described in terms of consistency, colour and structure with comments on minor constituents or apparent special features. Consistency is based on the shear strength of the soil, and is generally estimated from experience, measured by hand penetrometer or determined by laboratory testing. Terms used in describing consistency are set out below:

Term	Shear Strength
VERY SOFT	less than 12kPa
SOFT	12-25kPa
FIRM	25-50kPa
STIFF	50-100kPa
VERY STIFF	100-200kPa
HARD	greater than 200kPa

NON-COHESIVE SOILS are described in terms of relative density, colour, with comments on minor constituents or apparent special features. Relative density or density index is generally based on standard penetration testing (AS1289 Test F3.1), or other forms of penetration testing. Terms used in describing relative density are set out below:

Term	Relative Density	SPT "N" Values blows/300mm
VERY LOOSE	less than 15%	less than 5 blows
LOOSE	15-35%	5-10 blows
MEDIUM DENSE	35-65%	10-30 blows
DENSE	65-85%	30-50 blows

ENGINEERING CLASSIFICATION OF ROCK[1]

ROCK STRENGTH

DESCRIPTION TERM	GENERAL FIELD GUIDE	Is (50)[2] MPa	AS1726 ROCK STRENGTH	APPROX.[3] q_u MPa
Extremely Weak	Easily remoulded by hand to a material with soil properties		Extremely Low	
		0.03		0.7
Very Weak	May be crumbled and fragmented by hand		Very low	
		0.1		2.4
Weak	A piece of core 150mm x 50mm diameter may be broken by hand and easily scored with a knife. Sharp edges of the core may be friable and break during handling. Lumps of rock crumble with light hammer blow		Low	
		0.3		7
Medium Strong	A piece of core 150mm x 50mm diameter may be broken by hand with considerable difficulty but may be scored with a knife. Core may be broken with a light hammer blow.		Medium	
		1		24
Strong	A piece of core 150mm x 50mm diameter cannot be broken by hand, but can be slightly scratched or scored with a knife. Core may be broken with a blow from a hammer.		High	
		3		70
Very Strong	A piece of core 150mm x 50mm diameter may be broken with a heavy hammer blow, but cannot be scratched by a knife.		Very high	
		10		240
Extremely Strong	A piece of core 150mm x 50mm diameter is difficult to break with a hammer and rings when struck		Extremely High	

Notes:
1. Based on AS1726 - 1981 - SAA Site Investigation Code.
2. Point Load Strength Index : ISRM Committee on Laboratory Tests, Document No. 1, October 1972.
3. The approximate unconfined compressive strength (q_u) is based on an assumed ratio to the point load index of 24:1. This ratio may vary widely.

RQD: ROCK QUALITY DESIGNATION (expressed as a percent) for core recovered from borehole, defined as :

sum of sound core pieces 100mm or more long*
/ total length of section considered

* core fractured by drilling process considered unbroken

FRACTURING

DESCRIPTIVE TERM	GENERAL FIELD GUIDE
Fragmented	The core is comprised primarily of fragments of length less than 20mm and mostly of width less than the core diameter. Generally >50 breaks/metre.
Highly Fractured	Core lengths are generally 20-40mm with occasional fragments. Generally 25-50 breaks/metre.
Moderated Fractured	Core lengths are generally 30-100mm with occasional shorter and longer section. Generally 10-30 breaks/metre.
Fractured	Core lengths are generally 80-400mm with occasional shorter and longer sections. Generally 3-12 breaks/metre.
Slightly Fractured	Core lengths are generally 300-1000mm with occasional shorter and longer sections. Generally 0-3 breaks/metre.
Unbroken	The core does not contain any natural breaks.

Note: This classification applies to drill cores and refers to the spacing of all types of natural fractures along which the core is discontinuous (i.e. natural breaks). Such natural fractures include joints, bedding plane partings and other natural rock defects but exclude artificial fractures such as breaks caused by drilling and boxing core. Natural unbroken defects such as tight joints and veins along which no fracture is present are also excluded.

- Modified from draft prepared by the Sydney Group of the Australian Geomechanics Society, January 1975.

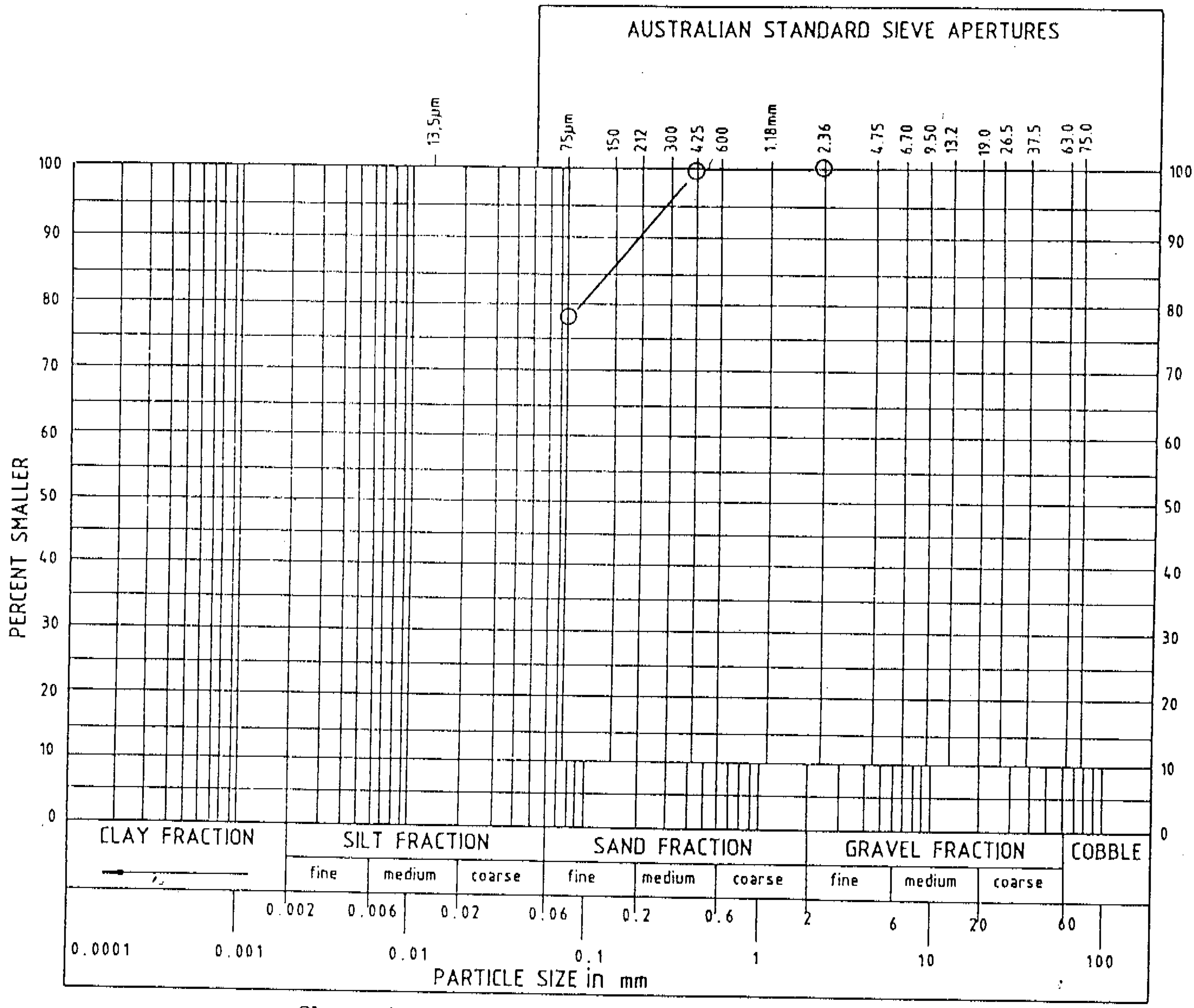
MT TODD GOLD PROJECT

MINE INFRASTRUCTURE

SITE INVESTIGATION

88638252(B)

TEST No.	DEPTH (m)	DESCRIPTION	IN-SITU CONDITION	PLASTICITY		LINEAR SHRINKAGE	ESTIMATED CBR*	GRADING		
			Moisture Content %	Liquid Limit %	Plasticity Index %	%	%	pass 2.36mm %	pass .425mm %	pass .075mm %
TP8	0.5	Silty sand and gravel	2.5	-	-	-		43	35	25
TP19	1.0	Clayey silt with some fine sand	2.6	23	6	2.0	16	100	100	79



DESCRIPTION Clayey silt with some fine sand

TEST TYPE AS1289 B1.1, C1.2, C2.1, C3.1, C4.1, C6.1

PRE-TREATMENT NIL

NATURAL MOISTURE CONTENT 2.6%

LIQUID LIMIT 23%

PLASTIC LIMIT 17%

PLASTICITY INDEX 6%

LINEAR SHRINKAGE 2.0% Mould length 125mm

RESULTS OF CLASSIFICATION TESTS

SITE MT TODD GOLD PROJECT

CLIENT MINPROC ENGINEERS PTY LTD

EXCAVATION No. TP19

DEPTH 1.0m

SIGNATORY [signature]

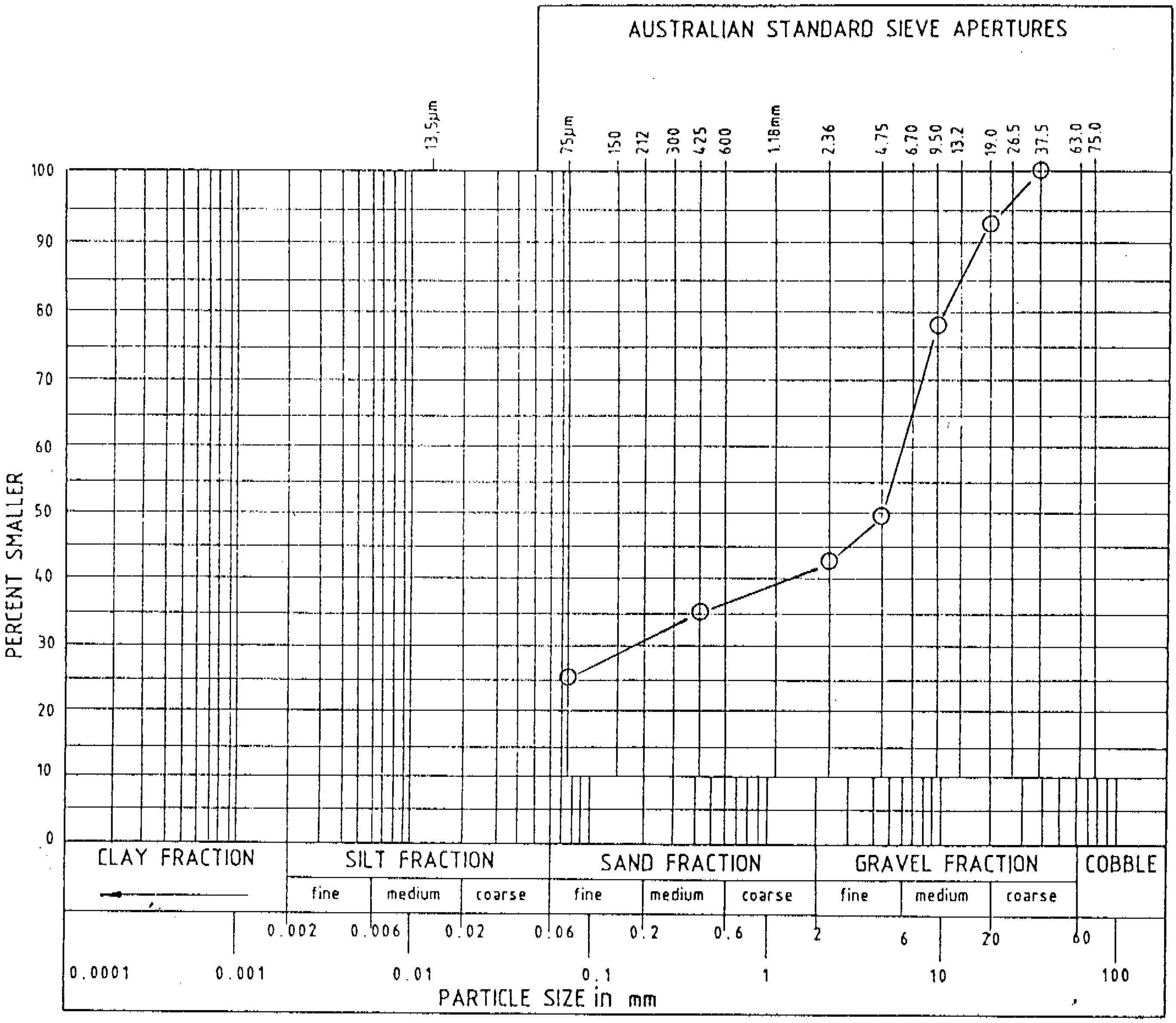
DATE 20/1/89

NATA

This Laboratory is registered by the National Association of Testing Authorities, Australia. The test(s) reported herein have been performed in accordance with its terms of registration. This document shall not be reproduced except in full.

REPORT No. N88-537/1
JOB No. 88638252
DATE 14.12.88
LABORATORY BRISBANE
LABORATORY No. E1155

Golder Associates
GEOTECHNICAL ENGINEERS



DESCRIPTION Silty sand and gravel

TEST TYPE AS1289 B1.1, C6.1

PRE-TREATMENT NIL

NATURAL MOISTURE CONTENT 2.5%

LIQUID LIMIT -

PLASTIC LIMIT -

PLASTICITY INDEX -

LINEAR SHRINKAGE - Mould length -

RESULTS OF CLASSIFICATION TESTS

SITE MT TODD GOLD PROJECT

CLIENT MINPROC ENGINEERS PTY LTD

EXCAVATION No. TP8

DEPTH 0.5m

SIGNATORY

DATE 20/1/89

REPORT No. N88-537/2

JOB No. 88638252

DATE 14.12.88

LABORATORY BRISBANE

LABORATORY No. E1156

Golder Associates

GEOTECHNICAL ENGINEERS

BILLITON AUSTRALIA GOLD PTY. LTD.
MT. TODD JOINT VENTURE
Annual Report for Period Ending 28 January, 1989

APPENDIX XIV
Preliminary Environmental Review
- Kinhill

ENVIRONMENTAL CONSIDERATIONS OF THE MT TODD PROJECT 1988

AIM:

This assessment was initiated as part of the review process of considerations for potential environmental effects of the activity undertaken in the Mt Todd Area.

The purpose of this environmental report is to identify the principal issues associated with activities to be undertaken in the area comprising the Mt Todd JV mining leases, and to put forward management suggestions designed to control these issues.
This is however, by no means a comprehensive review, as further work is to be done by Kinhill Engineering (an environmental/engineering consultancy group) who will be preparing a Preliminary Environmental Review.

This report includes; a description of the proposed project, the existing environment of the site, the potential impact arising from the project and the safeguards proposed for environmental protection.

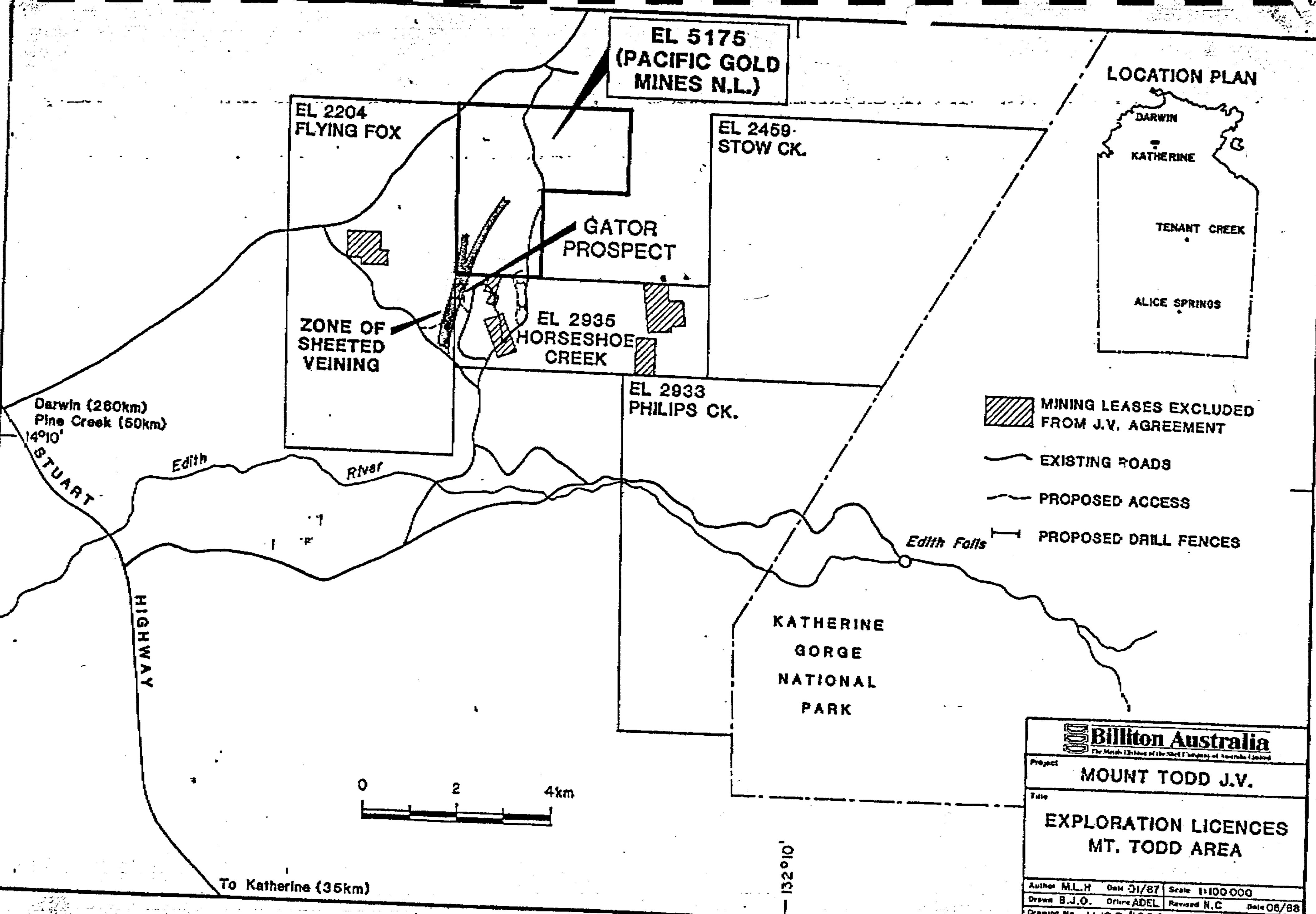
The information is based on a review of available data ie., information provided by local government and other authorities, site visit and review, as well as 2nd hand data collection - literature reviews of closely associated circumstances.

In section 4 of the "Environmental Assessment Act, 1982", states its aim as being; " ...to ensure that each matter affecting the environment which is, in the opinion of the Minister, a matter which could reasonably considered to be capable of having a significant effect on the environment, is fully examined and taken into account..."

Thus a major part of our work to date has been the liaison with government departments, e.g. The Department of Mines and Energy and The Conservation Commission, to adequately ensure that the above aim is being met.
mt todd:sjm/1

MINING TENEMENTS:

The area involved is approximately 40 km north-north west of Katherine, and approx. 47km south-east of Pine Creek. It lies 13 km east of the Stuart Highway from which it is reached via 6km of bitumen road and 7 km of gravel track/road.
Billiton Australia Gold together with Zapopan each have a 50% interest in The Mount Todd Joint Venture, with Billiton being the manager of the project. The tenements involved are located as shown on the attached figures (Fig's 1 & 2). Fig. 2 also shows the tenements currently held by Pacific Gold Mines, over which BAUG has an option to purchase. These tenements are also subject to the BAUG/Zapopan Joint Venture.

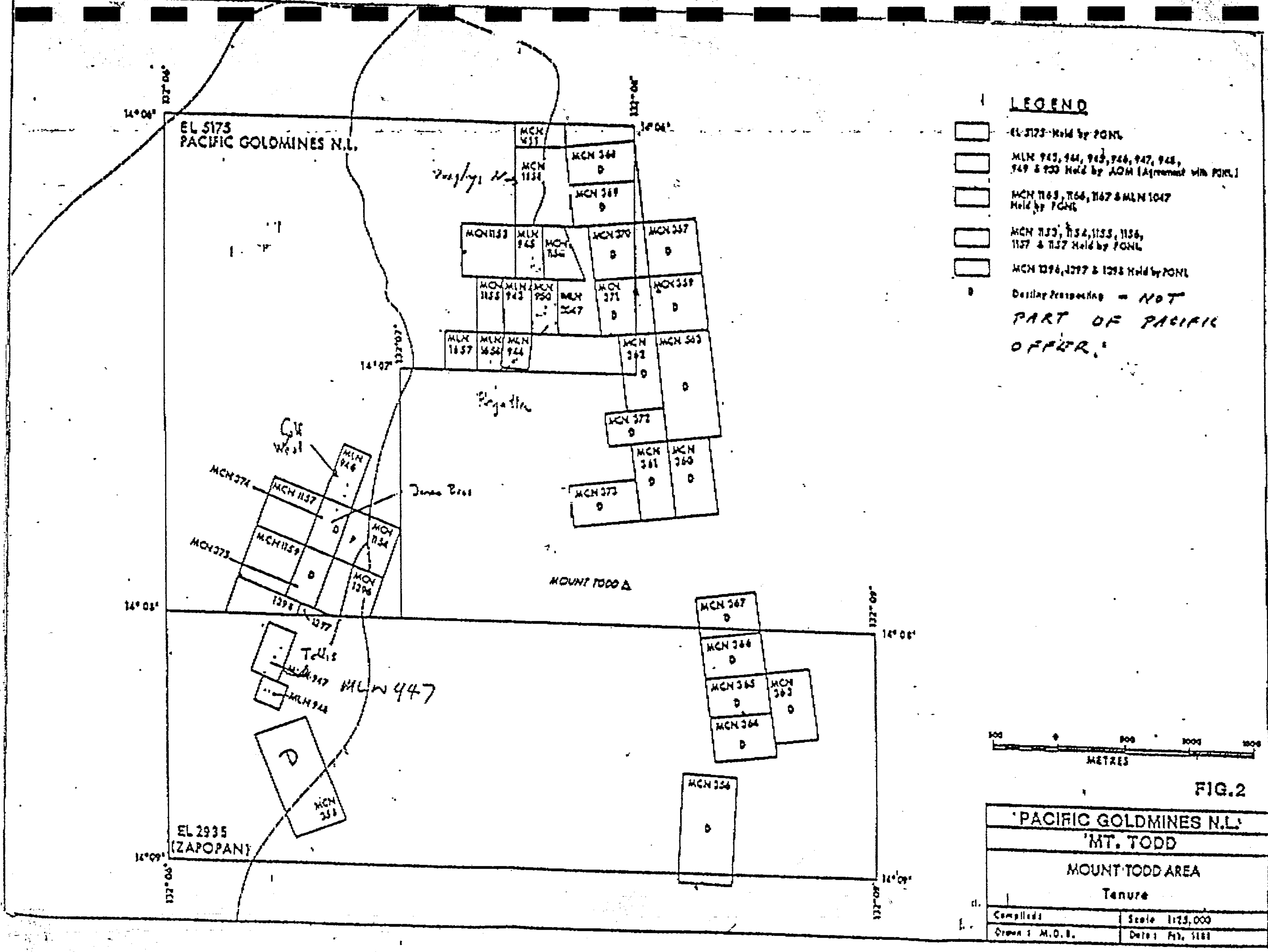


Billiton Australia

Project: MOUNT TODD J.V.

Title: EXPLORATION LICENCES MT. TODD AREA

Author M.L.H	Date 01/87	Scale 1:100 000
Drawn B.J.O.	Office ADEL	Revised N.C Date 06/88
Drawing No HJ90/1000		Fig No. 1



LEGEND
EL 5175 Held by PGNL
MLN 943, 944, 945, 946, 947, 948, 949 & 950 Held by AOM (Agreement with PGNL)
MCN 1163, 1166, 1167 & MLN 1047 Held by PGNL
MCN 1153, 1154, 1155, 1156, 1157 & 1157 Held by PGNL
MCN 1296, 1297 & 1298 Held by PGNL
D Drilling Prospecting = NOT PART OF PACIFIC OFFER.
EL 5175
PACIFIC GOLDMINES N.L.
EL 2935
(ZAPOPAN)
MOUNT TODD
MLN 947
METRES
'PACIFIC GOLDMINES N.L.'
'MT. TODD'
MOUNT TODD AREA
Tenure
Scale 1:25,000

FIG.2

EXISTING ENVIRONMENT

GENERAL DESCRIPTION:

The main area of interest, known as the Batman zone, is 222m above sea level at its highest point. Overall, the landscape in the Mount Todd area is gently undulating and heavily dissected with ephemeral streams, which flow south into the Edith River.
The terrain from Katherine to Pine Creek generally consists of plains with occasional rocky outcrops.
Eucalypt/Acacia woodland, of medium to low density predominates throughout the ridges/hills and valleys, with a higher density being associated with watercourses.

A. GEOLOGY:

The area consists of undulating foothills and strike ridges which have resulted from the erosion of Archean metasediments, granites and gneisses from lower proterozoic metasediment and igneous intrusives (Dames & Moore, 1985).
It lies to the southern extremity of the Pine creek Geosyncline.

Two formations of interest, The Burrell Creek Formation and The Tollis Formation, comprise greywacke, siltstone and shale with minor volcanics. The Burrell Creek Formation containing mostly greywacke and hornfels, while The Tollis Formation consists mainly of shales and siltstones.
The strata strike mainly north-north-west, dipping steeply to the west, although some tight folding has resulted in an easterly dip.

The most promising area of interest, the Batman Prospect, occurs within part of the Burrell Creek Formation and dips 60 to 80% to the west, whereas the mineralised quartz veins dip steeply to the east at a low angle to the strike. The strata are mainly shale/siltstone on the eastern side and greywacke on the western side of the prospect.

B: GROUNDWATER

The known occurrences of groundwater in the project vicinity are in fractured shale bedrock.

In this area, shales of the Tollis Formation, such as at Robin Prospect appear to be productive aquifers, with local occurences of groundwater in the Burrell Creek Formation. The groundwater is contained in secondary porosity arising from brittle fracture of the rocks within the formation.

Drilling at the Batman Prospect indicates that small supplies of groundwater lie at the base of oxidisation and that a larger supply occurs in fracture zones. The water salinity has been measured in drilling samples at around 440 mg/l TDS.

Rockwater have found from their analysis that the groundwater is fresh,and should prove suitable for human consumption as well as process use. No above-limit arsenic concentrations have been found in it, at the sites sampled in the present project area, excepting for higher arsenic values (0.064 mg/l), which were measured in groundwater north of Mt Todd, where there is also a potential for a high iron content (exceeding 10mg/l). This compares with the WHO standards of 0.3 mg/l (working level) for iron and 0.05 mg/l for arsenic.

Depths to water table have been found to be shallow at between 15-30 meters below surface.

C. CLIMATE:

The Climate of the area is tropical savannah with two seasons - a warm dry period from May to September, and a hot wet summer period from October to April. As seen by figure 3, virtually all rain falls in the summer wet season, mostly during November to March.

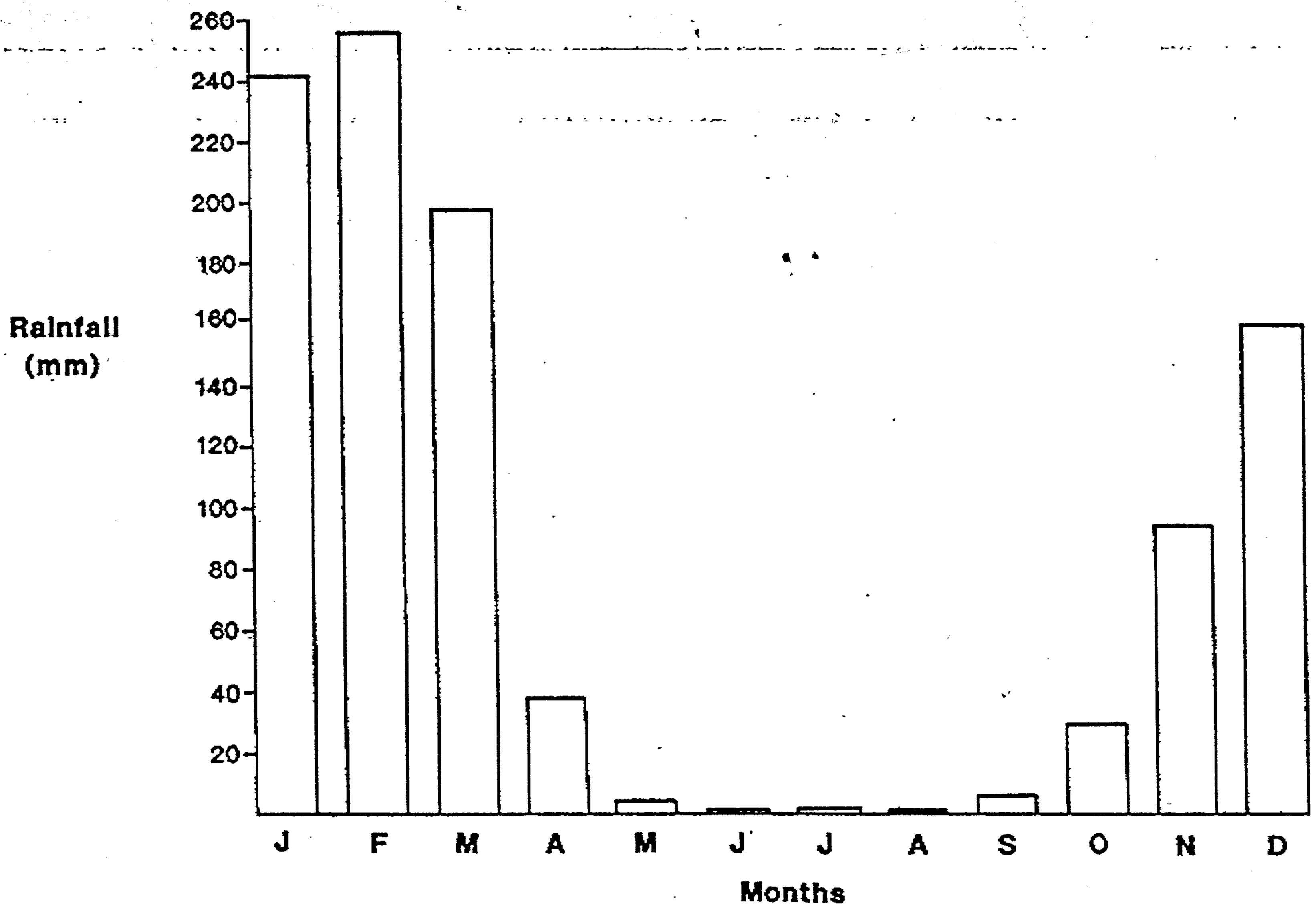


Figure 3 :*Mean monthly rainfall*

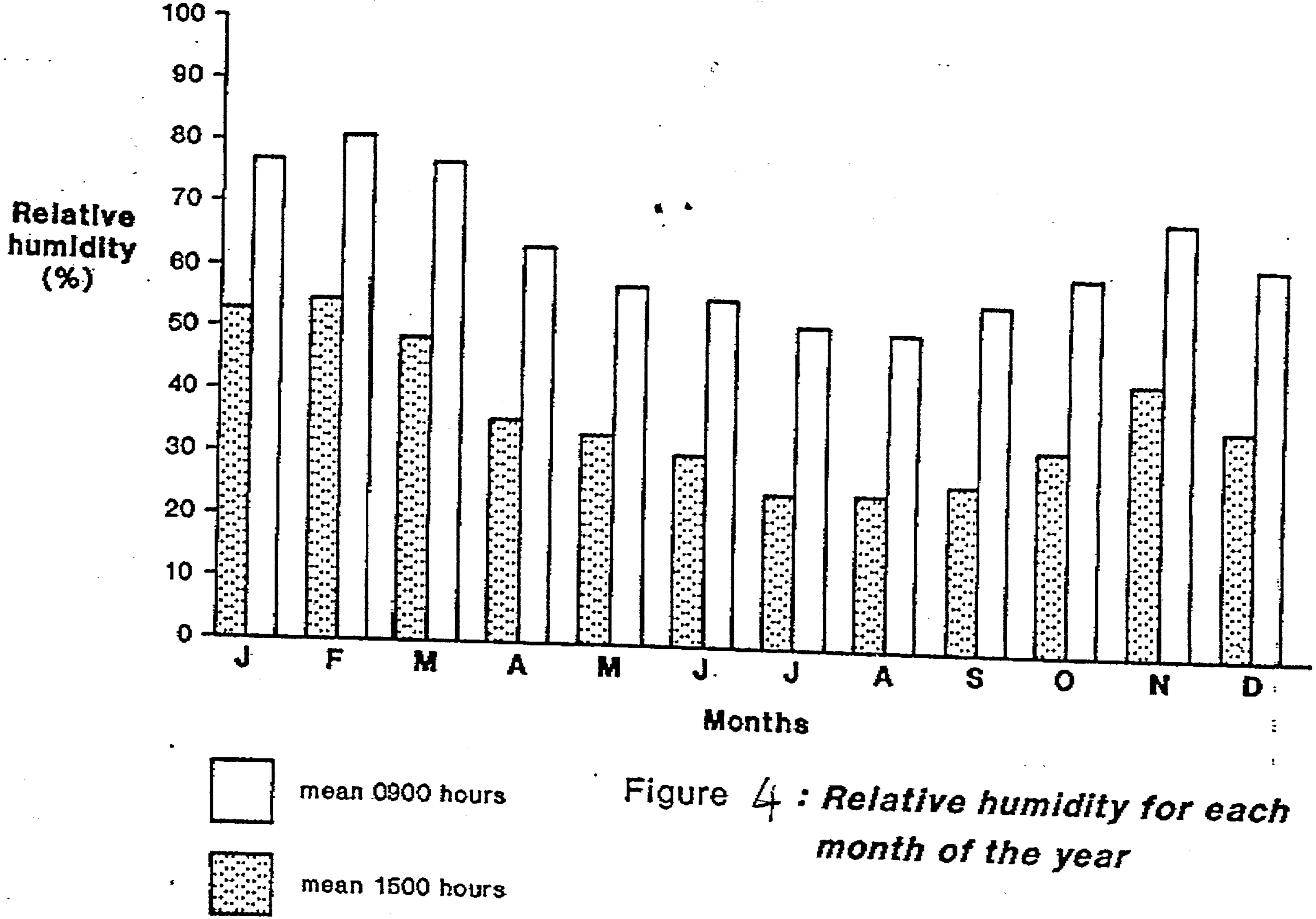


Figure 4 : *Relative humidity for each month of the year*

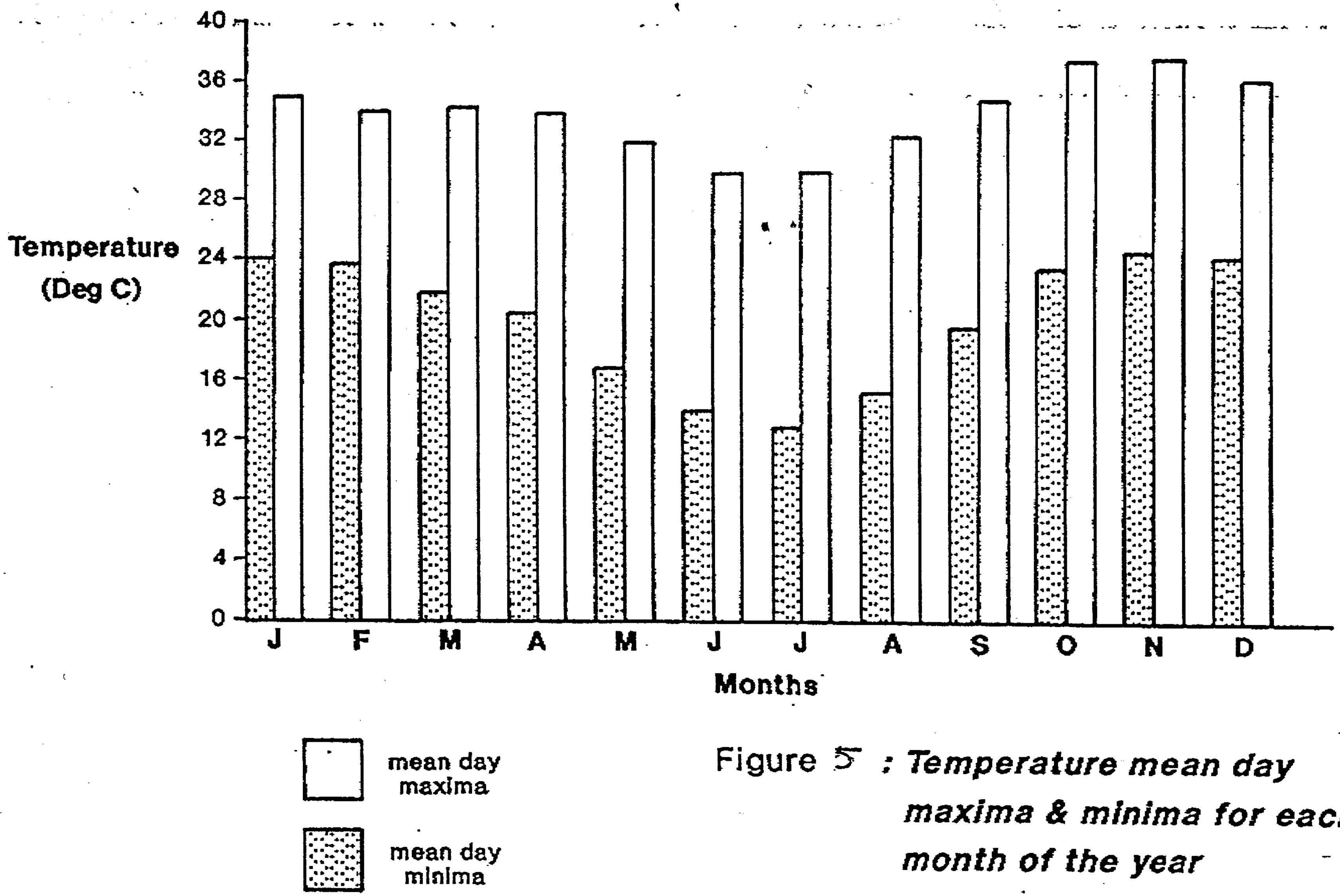
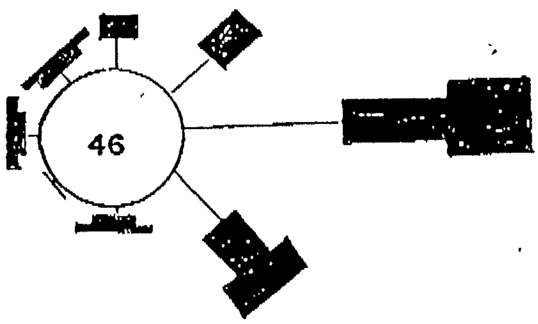


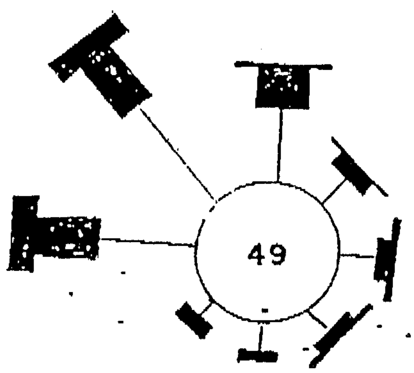
Figure 5 : *Temperature mean day maxima & minima for each month of the year*

WIND SPEED AND DIRECTION (WIND-ROSE) FOR KATHERINE

Dry Season (March – September)
Composite (0900 hours)



Wet Season (October – February)
Composite (0900 hours)



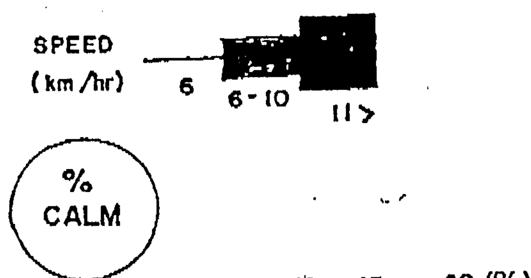
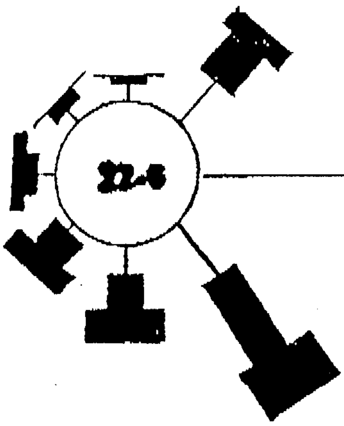


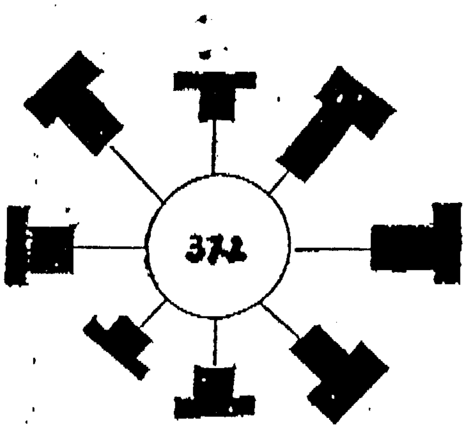
Figure 6

WIND SPEED AND DIRECTION (WIND-ROSE) FOR KATHERINE

Dry Season (March – September)
Composite (1500 hours)



Wet Season (October – February)
Composite (1500 hours)



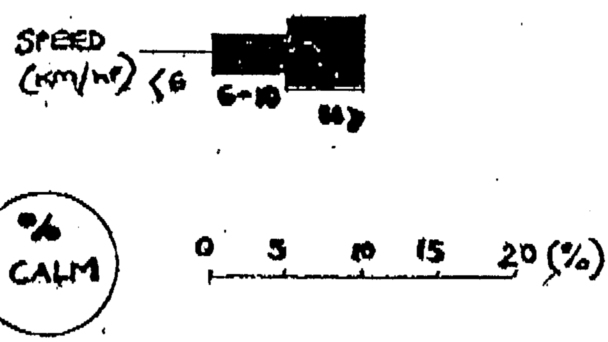


Figure 7

Average rainfall decreases southwards from more than 1,500mm at Darwin to 1,040mm per annum at Katherine.
The mean average rainfall for the area is 800-1200mm per annum and the mean average rainfall 2000-2250mm per annum (Dames and Moore,. 1985).
The average evaporation rate is 2,570mm per year.

Figure 5 shows that air temperatures are uniformly warm throughout the year, with daily maxima ranging from 30^{0}C in June/July to 38^{0}C in October/November; whilst mean daily minima range from 13.2^{0}C June/July to 24^{0}C November/January.
As seen by figure 4, mean humidity (9000 hours reading) is relatively high from December to February, (up to 81%) and around 52% in the winter months.

Figure 6 and 7 illustrate that easterly winds predominate from March to September. From October to February the winds are variable, with north-westerly winds being most frequent from November to January. May and June tend to be the windiest months, with a higher frequency of speeds over 20 km/hr.
During the wet season average winds are low, there is a high percentage of calms but some strong movements usually associated with thunder storms.

D. SOILS

Lithosols are dominant of areas of greater relief. Alluvial valleys and lower side slopes have lateritic podzols, yellow earths and soddic soils. The valleys or low relief areas consist of unconsolidated soils of aeolian (windblown) or alluvial origin.
The skeletal, rocky soils at Batman are of poor fertility, with very little topsoil due to the finer particles being washed out during the wet season. The shallow soils are underlain by framented rock, with weathered rock size increasing down the soil profile. The soils are unconsolidated and poorly developed on the slopes, with the erosion risk being very high ie. no clay to bind the soil, thus the vegetation cover would be an important protection factor during the wet season.

E: VEGETATION

Eucalypt woodland comprising mainly of Eucalytus bleeseri and E. tetradonta and tussock grasses. Closed forest adjacent to water courses.

As seen by past workings and disturbed areas, the vegetation in the vicinity seems well able to naturally regenerate, with many areas now being under regrowth of vegetation types similar to that of undisturbed areas.

A vegetation map of a scale of 1:10,000 or larger should be prepared by the consultants. This will involve both aerial-photo interpretation and ground truthing (vegetation survey).

F: FAUNA

The area surrounded by the Fergusson and Edith Rivers is noted on the register held by the Conservation Commission as a well known habitat for the Chestnut Quilled Rock Pidgeon (Petrophassa rufipennis). This rare species has a limited distribution in the Northern Territory occurring only to the east of the Stuart Highway. It frequents sandstone escarpments nesting among rocks, and feeds on the seeds of grasses and herbaceous plants, travelling short distances (approx. 100m) to drink (Frith, 1979).

Some rare bat species have also been mentioned as being significant in the environs of the Katherine area, although none as yet have been recorded in the Mt Todd area.

A fauna survey is required to determine those species of significant status.

G: SOCIO - ECONOMIC

The major industries in the Katherine and surrounding area are tourism, mining, fishing and grazing. The mine itself would be an added economic benefit to the community, providing direct and indirect benefits.

A workforce of approximately 80 will be involved in the mine. Of these most will be recruited from outside the Katherine area. It is currently envisaged that accomodation will be provided at Katherine.

LAND-USE:

The area in the vicinity of the proposed mine is crown land, which in the past has been under various pastoral leases. The area has a history of mining, mainly open-cut extraction of gold and tin. There are scattered open cut excavations and stockpiled overburden dumps which are very visible and have just been abandoned without any restoration work. Natural re-generation seems to be occuring around the open-cuts in some cleared areas.

ASTHETIC, RECREATIONAL AND HERITAGE VALUES:

The area has no significant or unique asthetic value, due to disturbance, clearing, pastoral uses and overburden dumps. However, (according to the Conservation Commission), there is some recreational use of the area, for gemstone and fossil collection. Topaz is noted as the primary gemstone of interest.
As the mining operations would occupy only a small area there would be little interference with these recreational uses.

Wolfram Hill, northeast of the prospect, is the nearest area of heritage value. No impact is foreseen as the area is considered to be remote from envisaged operations.

NATIONAL PARKS:

The Katherine Gorge National Park 7km due east of the Mount Todd area is the nearest park in the vicinity. As it is also some distance away, little impact is anticipated.

ABORIGINAL SITES:

Two sacred sites have been identified outside and to the south of the Mt. Todd JV area. Liasion with the Sacred Sights Authority has been established and it does not appear that there are any major problems in this regard.
There is an Aboriginal Land Claim covering the area concerned, however it is understood that the portion of the claim covering the project has not been recommended for grant. Two further land claims have subsequently been lodged over the same area as the original claim.

PROJECT DESCRIPTION:

GENERAL:

Although planning of the mine is at an early stage, it is envisaged that the area of Batman would be developed as an open-cut mine, operating on a 24 hour basis.

The ore is of low grade and conceptually suited to heap leaching. It is intended that testwork will address two major avenues for each of the ore types a) heap leaching and b) beneficiation followed by Carbon-in-leach (CIL).

The estimated water requirement is 0.75×10^6 cu m/yr (2,740 cu m/d), and a larger supply might be sought to allow for ancillary uses and contingency.

DESIGN:

A heap leaching pad, opencut mine, associated overburden dump, pregnant pond, metallurgical treatment plant and associated storages, workshops, stores and offices would form the basis of activity and design of the area, along with other support facilities such as process water reservoir, electricity generating unit, borewater sites etc.
These are to be addressed in further detail along with the proposed waste facilities - waste oil, sewage, kitchen garbage etc.

ORE BODY - CONTENT:

This is obviously of utmost importance and will be addressed by Kinhill when adequate data is avaiable.

mt todd:sjm/7

ENVIRONMENTAL ISSUES

FIRE:

Fire behaviour in this region is largely influenced by the density of the grasses in the understorey, a hilly topography and the very seasonal nature of the rainfall. During the dry season, it is expected that fires would be of high intensity and spread rapidly due to the relatively uniform nature of grass layer and slope of terrain. Burning is at presently managed by setting fire to the grasslands towards the end of the wet season.

DUST & NOISE POLLUTION:

These will be limited and confined to the immediate area of operations due to the distance between the mining activity and the nearest townships of Katherine and Pine Creek. There is little likelyhood of any impact on these communities.

VEGETATION:

The more obvious impacts of mining in the area are those on the vegetation, through both minimal clearing as well as disturbance, trampling, and the increase of dust emissions from increased usage of roads and disturbed surfaces (ie. reduction of light for photosynthesis).
Another is the possible increase in the frequency of weed species due to the clearance and disturbance of the natural vegetation.

FAUNA:

The increase in road traffic, human presence and disturbance, are likely to cause dislocation of fauna. Mobile fauna however will return, and emplacement of logs can provide habitats for ground dwelling fauna.

mt todd:sjm/8

The destruction of habitat and resources especially for the rock-pidgeon and other species will be of minimum impact due to the regionally common nature of the habitats. Due to the extensive grazing land-use ie. already disturbed land, it is unlikely that the project would have any additional adverse effect on the fauna.
Traditional aboriginal use of plants and animals (if in fact this occurs) should not be largely effected as the species are widespread throughout the area.

Planning to avoid sensitive areas, where possible, will favour the conservation of any rare or endangered species found in the area.

MINE PRACTICES AND WATER MANAGEMENT:

One of the major issues to be addressed is the possibility of contamination by cyanide and other poisonous/harmful elements, which may be utilised in the mining and treatment processes.
Safe design and practices for the heap leach pad will ensure that both environmental and economic conditions are satisfied. For Exampleoptimum retention of the cyanide/gold liquor, both prevents contamination of the environment and ensures that as much gold is recovered as is possible.

Excessive rainfall in the wet season is a major consideration in the design of overflow or stormwater ponds which connect to the pregnant and barren liquor ponds. To ensure total catchment of stormwater it is necessary to review rainfall records over the previous 50-100 years, to obtain the highest recorded storm event (Mcrabb, 1988).
Using the available data from the Katherine area (appendix 1), the maximum amount of rainfall over the highest recorded 24-hour period is 140mm. Rockwater, who have obtained figures from the CSIRO, and have records going back 100 years as against 31 years, have a high daily duration of 133mm. Thus, the pond needs to be sized to contain rainfall from the highest recorded storm event, in this case between 133-140mm.

LINER LEAKAGE AND CYANIDE POLLUTION:

The principal site of solution loss is through the pad liner during the construction phase. Leakage is caused by ripping, puncturing or inefficient gluing/welding.
A detection system can be laid below the liner in the sand or material used for liner protection. This consists of slotted agricultural pipe which is laid to run with the slope to a collection well.
This detection system is a good monitoring system,however if cyanide/gold liquor leakage is detected, little can be done to solve the problem. The best method therefore is a positive approach; ie. strict supervision of gluing/welding of this liner and when ore is dumped onto it.
A sand or tailing layer is usually placed on both sides of the liner to reduce rock impact.
mt todd:sjm/9

The liner thickness used by Billiton, is estimated to be at least 1.0mm thick, as compared to the requirement of 0.5mm thick.
Apart from liner construction/supervision, monitoring wells located downslope from the pad and sunk to water table or depth that will accumulate water following rain, are recommended (McCrabb, 1988). This allows the monitoring of water quality below the pad.

DECOMMISSIONING:

When the gold has been recovered, it is necessary to neutralise the cyanide within the system. This is achieved by firstly removing all traces of cyanide from the heap by continuous flushing via the sprinkler system. The heap will then constitute a lower body of rock, with all cyanide bearing solution being contained in the pond area. Neutralisation procedures can then be applied to the pond (aeration and U.V)

The heap can be covered with waste rock to reduce its permeability and to extend its watershed outside the liner drain area. Drainage from the heap can be re-directed to flow directly to the stormwater pond. As it is difficult to initiate reclamation works for a heap which is contained within the operating mine, the approach of incorporating the heap into the long term reclamation plans for the mine should be condsidered. Again this involves covering the heap with overburden allowing the heap to be incorporated into the mines existing or future earthworks. These earthworks can then be rehabilitated at the cessation of mining.

WILDLIFE PROTECTION:

A general requirement would be to stockproof the entire heap leach area with fencing.
Birds are attracted to the water source of a heap leach area during dry periods when alternative sources are not abundant.
At heap leach operations in North coastal Australia, birds have been found drinking from the ponds and bathing under sprinklers and consequently dying on site (McRabb, 1988).

mt todd:sjm/9.1

McRabb, 1988; also stated that this is best avoided by providing an alternative water source a short distance away, and that it should remain undisturbed. This has merit, especially if the Quilled Rock Pidgeon and other rare or endangered species was found to frequent the area.

PEST SPECIES:

Mosquitoes form the main issue here, in that, the three most commonly found species found in sewage ponds and their effluents, namely Clunex annulirostris, C.quinquefasciatus and Anopheles hilli, transmit a variety of dangerous viruses and diseases. The adult A. hilli species, is also capable of transmitting malaria.
Thus practices ensuring that breeding sites are eliminated is an importand consideration.

MANAGEMENT SECTION:

The status and significance of the vegetation and fauna are related to the degree to which past land-use has altered the original vegetation, the extent to which similar vegetation occurs elsewhere, and the occurrence of protected and rare species. The proposed development is in an area which has had a history of disturbance through small-claim mining, larger scale open-cut extraction, and grazing; and is not a new development in an otherwises unaltered area. The major direct effect of the proposed mine would involve the clearing of a mostly disturbed area of deciduous woodland.

The upgrading and use of existing tracks, and restricted clearing, would all help in minimising disturbance in the area.
Top-soil i.e. the first 100cm will be separately removed and stockpiled from excavated areas, and subsequently replaced upon completion of construction. Conservation of toplayer will carry over seed and small rootstock. Compacted areas can be ripped to permit rainwater infiltration and root penetration, prior to topsoil return.
On cessation of use, if desired, temporary tracks can be rehabilitated.
Rapid rehabilitation of disturbed areas would then be carried out in order to minimise loss of habitat and environmental impact.

The regrowth of vegetation on old diggings in the area suggests that natural re-vegetation takes place rapidly, and that this vegetation is similar to the nature of vegetation of undisturbed areas. Thus the vegetation map/survey will show what are the appropriate vegetation types for stabilising embankments, probably as described by Kinhill, 1984, 'mixed woodland of low hill slopes'.

The Overburden dump should be graded to a minimum slope to inhibit erosion but ensure adequate drainage. Progressive shaping of dump to enable progressive rehabilitation should be initiated from an early stage.

mt todd:sjm/11

Effective supervision during clearing, construction and rehabilitation should be undertaken. Personnel should recieve instruction on safety practices and the importance of fire.
As the area is just outside of the 50km radius of Katherine PO (fire protection zone), fires may be lit except within areas that have been declared as fire danger areas.
If a fire is required for the purpose of clearing land, burning firebreaks or for any other purpose, a permit must be obtained from a Fire Warden or Fire Control Officer.

To ensure that mosquitoes are controlled, planned modification of the environment which physically removes water in which mosquitoes develop or that makes the water unsuitable for breeding should occur,e.g. filling in small depressions with sand or earth, or in the case of dams or water storage, alternating the water level to kill mosquito larvae (Whelan,1983).

Monitoring programs should be formulated in such a way so as to provide baseline water level and water quality data to observe the effects of mining activity on the existing groundwater regime.

Overall, continued liaison with the relevant authorities and other interested parties is to be maintained, so as to obtain advice as well as ensuring that suitable standards are upheld.

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5. McCrabb, M.J. Environmental Engineering Aspects of Heapleaching in Australia in Economics and Practice of Heap Leaching in Gold Mining, Workshop Conference Proceedings. Cairns, 3-6 August, 1988.

6. Mining and the Environment - A Professional Approach. July 1987. Brisbane, Qld. AIMM.

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8. Whelan, P. 1983. The Physical and cultural Control of Mosquitoes. Department of Health and Community Services.

Appendix 1.

BUREAU OF METEOROLOGY
DARWIN REGIONAL OFFICE

TABULATION OF CLIMATIC AVERAGES AND EXTREMES

STATION: Katherine (COMPOSITE)
RECORDS COMMENCED: 1957

RAINFALL (mm)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Ann
Mean Monthly	244	257	198	39	5	2	2	1	7	30	95	160	1040
Rain Days	17	17	12	3	1	0	0	0	1	3	8	12	74
High Monthly	550	493	516	180	40	50	48	11	42	138	214	391	1575
Low Monthly	51	58	12	0	0	0	0	0	0	0	4	43	558
High 24-hour	115	107	134	100	19	33	34	11	35	52	121	140	

TEMPERATURE (Deg C)

	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Ann
Mean Day Max	35.0	34.3	34.5	34.0	32.1	30.0	30.1	32.5	35.4	37.7	38.0	36.5	34.2
Mean Day Min	24.0	23.7	22.9	20.4	17.1	14.1	13.2	15.5	19.6	23.6	24.7	24.4	20.3
Mean Monthly	29.5	29.0	28.7	27.2	24.6	22.1	21.7	24.0	27.5	30.7	31.4	30.5	
High Mean Max	37.0	37.8	38.4	37.4	34.7	32.4	31.9	35.0	36.8	39.5	39.7	39.6	
Low Mean Max	32.0	31.7	31.6	30.9	29.0	27.6	27.7	30.4	34.0	35.2	34.2	33.1	
High Day Max	41.1	40.5	39.2	38.3	36.0	36.1	35.2	37.3	39.4	41.7	45.6	43.3	
Low Day Max	36.1	25.9	22.8	22.7	18.3	21.5	18.3	24.2	27.0	26.6	17.8	25.0	
No Days > 35	17	13	14	10	1	0	0	4	20	29	27	25	160
No Days > 30	30	27	29	29	25	17	16	27	30	31	30	30	321
High Mean Min	27.7	24.5	24.4	23.8	21.0	18.4	17.7	19.1	23.2	25.6	26.0	27.9	
Low Mean Min	22.8	22.7	20.6	14.6	12.6	9.4	9.9	11.8	15.9	20.3	22.8	22.4	
High Day Min	27.8	27.8	26.5	26.5	25.6	24.0	24.5	25.2	27.2	27.9	30.0	29.4	
Low Day Min	17.2	16.7	13.8	10.7	7.2	3.4	2.8	5.3	9.8	11.0	17.4	17.3	

RELATIVE HUMIDITY (%)

Mean 9.00 am	77	81	77	64	58	56	52	52	51	56	61	70	63
Mean 3.00 pm	53	55	49	36	34	31	27	25	25	27	33	44	37

THUNDER DAYS

Mean	3	3	2	0	0	0	0	0	0	1	3	4	16

CLEAR DAYS (<2/8)

Mean	1	1	2	9	14	19	21	22	15	9	4	2	119

CLOUDY DAYS (>6/8)

Mean	19	17	15	7	5	2	2	1	3	5	8	14	98

Appendix 2

BUREAU OF METEOROLOGY - SURFACE WIND ANALYSIS

PERCENTAGE OCCURRENCE OF SPEED VERSUS DIRECTION BASED ON 29 YEARS OF RECORDS

FIRST YEAR : 1957 LAST YEAR : 1985 NUMBER OF MISSING OBSERVATIONS (AS PERCENTAGE OF MAXIMUM POSSIBLE) : 6.42 %

STATION : 014902 KATHERINE POST OFFICE WAS 014030 14 28 S, 132 16 E 105.0 M ELEV

JANUARY 0900 HOURS LST — SPEED (KM/HR) — CALM 53

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	5	3	*					8
NE	3	1	*					4
E	1	1	*	*	*			3
SE	2	*	*					2
S	2	*	*	*				2
SW	2	1	*					2
W	6	4	1	1				12
NW	9	5	2	*	*			16
ALL	29	16	4	1	*			

NO. OF OBS. 853

FEBRUARY 0900 HOURS LST — SPEED (KM/HR) — CALM 52

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	4	3	*	*				8
NE	3	1		*				4
E	3	1		*				4
SE	2	*		*				2
S	2	1	*					2
SW	1	1	*					2
W	8	4	1					13
NW	6	4	1	*				12
ALL	29	15	3	1				

NO. OF OBS. 788

MARCH 0900 HOURS LST — SPEED (KM/HR) — CALM 56

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	3	1	*					5
NE	5	2	1	*	*	*		8
E	6	3	1	*		*		11
SE	3	2	*			*		5
S	2	1	*	*				3
SW	1	1	*	*				2
W	3	2	1	1				6
NW	3	2	1	*				6
ALL	25	15	4	2	*	*		

NO. OF OBS. 861

APRIL 0900 HOURS LST — SPEED (KM/HR) — CALM 49

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	3	*	*					3
NE	5	4	2	1				11
E	10	7	3	[illegible]	*			22
SE	4	3	2	*	*			9
S	1	*	1	*	*			3
SW	*							*
W	1	1						1
NW	1	*						1
ALL	25	16	8	2	*			

NO. OF OBS. 828

JANUARY 1500 HOURS LST — SPEED (KM/HR) — CALM 38

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	3	1	1	*	*	*		5
NE	2	3	1	*		*		6
E	3	3	1	*		*		7
SE	3	2	1	*				5
S	3	2	1	*				6
SW	3	1	1	*				5
W	6	5	1	*	*			13
NW	6	5	3	1	1			15
ALL	29	21	9	2	1			

NO. OF OBS. 831

FEBRUARY 1500 HOURS LST — SPEED (KM/HR) — CALM 47

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	3	2	1	*	*	*		7
NE	2	2	1	*		*		5
E	3	2	2	*	*			7
SE	2	2	1	*	*	*		5
S	3	2	1	*	*	*		6
SW	3	3	*					6
W	6	4	3	*	*			13
NW	6	3	2	1	*			13
ALL	28	19	10	2	1	*		

NO. OF OBS. 772

MARCH 1500 HOURS LST — SPEED (KM/HR) — CALM 30

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	2	1	1	*	*			4
NE	5	3	3	1	1	*		12
E	5	7	5	1	*	*		19
SE	3	4	2	*	*			10
S	3	3	2	*	*			9
SW	2	2	*					5
W	3	1	2	1				6
NW	2	2	1	*				5
ALL	25	24	16	4	1	*		

NO. OF OBS. 841

APRIL 1500 HOURS LST — SPEED (KM/HR) — CALM 19

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	1	1	*	*	*			2
NE	5	5	4	1	*	*		15
E	9	17	9	3	*			38
SE	4	5	4	1				14
S	2	2	1	1	*			7
SW	*	1	*		*			1
W	1	1	*	*				2
NW	1	*	*					1
ALL	22	32	19	6	1	*		

NO. OF OBS. 812

* OCCURRED BUT LESS THAN 0.5 PERCENT

PRODUCED BY M.I.S.S. 29/ 7/86

FRAME A12

FIRST YEAR

STATION :

JANUARY — SPEE[illegible] — CALM 7

DIRN	1 TO 5	6 TO 10	11 TO 20
N	3	4	1
NE	7	1	1
E	3		
SE	3	2	
S	2		
SW	4	9	1
W	8	10	6
NW	16	4	2
ALL	51	31	11

JANUARY — CALM 5

DIRN	1 TO 5	6 TO 10
N	9	4
NE	11	3
E		3
SE	5	1
S	9	4
SW	9	3
W	4	3
NW	4	7
ALL	55	26

* OCCURR[illegible]

BUREAU OF METEOROLOGY - SURFACE WIND ANALYSIS

PERCENTAGE OCCURRENCE OF SPEED VERSUS DIRECTION BASED ON 29 YEARS OF RECORDS

FIRST YEAR : 1957 LAST YEAR : 1985 NUMBER OF MISSING OBSERVATIONS (AS PERCENTAGE OF MAXIMUM POSSIBLE) : 6.42 %

STATION : 014902 KATHERINE POST OFFICE WAS 014030 14 28 S, 132 16 E 135.0 M ELEV

MAY 0900 HOURS LST — SPEED (KM/HR) — CALM 37

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	1	1	*		*			2
NE	4	5	1	1	*			12
E	13	11	5	3	*			32
SE	4	4	3	1	*			12
S	1	1	1	*	*			4
SW	*	*	*					1
W	*	*						*
NW	*	*		*				*
ALL	24	22	11	4	1			

NO. OF OBS. 847

JUNE 0900 HOURS LST — SPEED (KM/HR) — CALM 40

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	1	*		*				1
NE	3	4	2	*	*	*		10
E	12	11	7	1	1	*		32
SE	5	5	2	*	*			12
S	1	1	1	*	*			3
SW	*	*	*	*				1
W	*	*						*
NW	*	*						*
ALL	22	22	12	3	1	*		

NO. OF OBS. 821

JULY 0900 HOURS LST — SPEED (KM/HR) — CALM 45

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	*	1	*					1
NE	3	4	*	*				8
E	14	9	5	1	*			30
SE	5	4	2	1	*			13
S	1	*	*					2
SW		*	*					*
W	*	*						*
NW	*							*
ALL	25	19	8	2	*	*		

NO. OF OBS. 850

AUGUST 0900 HOURS LST — SPEED (KM/HR) — CALM 50

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	2	*	*		*			3
NE	6	3	*		*			6
E	10	8	5	1	*		*	25
SE	4	3	1	*	*	*		9
S	1	1						3
SW	*	1						1
W	1							1
NW	1	*	*	*	*			2
ALL	23	16	8	2	1	*	*	

NO. OF OBS. 849

MAY 1500 HOURS LST — SPEED (KM/HR) — CALM 19

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	1	1	*	*				2
NE	4	7	3	2	*	*		16
E	11	14	7	3	1	*		36
SE	4	8	4	1	*			17
S	2	3	1	*	*			6
SW	*	1						1
W	*	*						1
NW	*	*		*				1
ALL	23	35	15	6	2	*		

NO. OF OBS. 826

JUNE 1500 HOURS LST — SPEED (KM/HR) — CALM 21

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	1	1	*					2
NE	3	6	4	2	*	*		15
E	10	13	8	3	*	*		35
SE	6	8	3	1		*		18
S	2	3	2	*	*			7
SW	1	1	*	*				3
W	*	1						1
NW	*	*			*			*
ALL	23	32	17	6	1	1		

NO. OF OBS. 796

JULY 1500 HOURS LST — SPEED (KM/HR) — CALM 21

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	*	*	*					1
NE	2	7	3	1				14
E	9	18	7	2	*			36
SE	5	8	3	1	*			17
S	2	2	2	*	*			7
SW	1	*	*					3
W	*	*	*					1
NW	1	*						1
ALL	22	36	16	4	1			

NO. OF OBS. 825

AUGUST 1500 HOURS LST — SPEED (KM/HR) — CALM 26

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	1	1	*					2
NE	3	7	2	*	*			13
E	9	14	9	3	*			35
SE	4	7	2	3	*			14
S	2	2	2	1	*			7
SW	*	*	*	*				1
W	1	1	*					2
NW	1	*	*					1
ALL	23	32	16	3	1			

NO. OF OBS. 829

* OCCURRED BUT LESS THAN 0.5 PERCENT

PRODUCED BY M.I.S.S. 29/ 7/86

FRAME 812

BUREAU OF METEOROLOGY - SURFACE WIND ANALYSIS

PERCENTAGE OCCURRENCE OF SPEED VERSUS DIRECTION BASED ON 29 YEARS OF RECORDS

FIRST YEAR : 1957 LAST YEAR : 1985 NUMBER OF MISSING OBSERVATIONS (AS PERCENTAGE OF MAXIMUM POSSIBLE) : 6.42 %

STATION : 014902 CATHERINE POST OFFICE WAS 014030 14 28 S, 132 16 E 105.0 M ELEV

SEPTEMBER 0900 HOURS LST — CALM 48

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	4	2	1					6
NE	3	3	1	*	*	*		8
E	7	6	3	1	*	*	*	18
SE	2	3	1	1		*		7
S	1	*	*	*				2
SW	*	*			*			1
W	2	1	*		*			3
NW	3	2	2	*				7
ALL	23	18	8	3	*	*	*	

NO. OF OBS. 827

OCTOBER 0900 HOURS LST — CALM 45

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	6	3	1	*	*			10
NE	3	3	*	*				6
E	4	2	1	*				8
SE	2	1	1	1				5
S	1	*	*	*	*			2
SW	*	*	*					1
W	4	3	1	*				8
NW	8	6	1	1				16
ALL	27	19	6	3	*			

NO. OF OBS. 855

NOVEMBER 0900 HOURS LST — CALM 44

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	6	3	1	*				10
NE	2	2	1	*				5
E	3	2	*	1				7
SE	2	1	*	*				3
S	1	1	*	*				3
SW	1	1	*					2
W	5	4	2	*				11
NW	10	5	1	*				16
ALL	31	18	5	2				

NO. OF OBS. 828

DECEMBER 0900 HOURS LST — CALM 52

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	5	2	*					8
NE	3	1	*					4
E	3	1	1	*				5
SE	1	1	*	*				2
S	1	1	*		*			2
SW	1	1	*					2
W	8	4	1	*				14
NW	7	4	1					12
ALL	29	14	4	*	*			

NO. OF OBS. 828

SEPTEMBER 1500 HOURS LST — CALM 22

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	2	1	*	*			*	3
NE	4	7	3	1	*			14
E	8	13	7	3	*			31
SE	4	5	4	1	*	*		13
S	2	2	1	*		*		6
SW	1	1	*	*				2
W	2	1	*					3
NW	2	1	*		*			3
ALL	24	30	15	6	1	*	*	

NO. OF OBS. 819

OCTOBER 1500 HOURS LST — CALM 31

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	4	2	1					6
NE	6	6	2	1	*			15
E	7	8	5	2	*			22
SE	3	3	3	*	*			9
S	3	3	1	*	*			7
SW	1	1						2
W	2	1	*	*				4
NW	2	2	1	*				4
ALL	27	25	12	6	1			

NO. OF OBS. 827

NOVEMBER 1500 HOURS LST — CALM 35

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	3	3	1	*	*			7
NE	4	5	2	1				12
E	8	5	3	1	*			17
SE	3	2	1	*				7
S	2	1	1	*	*			5
SW	2	1	*	*				3
W	3	3	1	*				7
NW	5	3	*	*				8
ALL	30	22	10	2	*			

NO. OF OBS. 806

DECEMBER 1500 HOURS LST — CALM 42

DIRN	1 TO 5	6 TO 10	11 TO 20	21 TO 30	31 TO 40	41 TO 50	51 & UP	ALL
N	3	2	*					6
NE	4	3	1	1		*		10
E	5	3	2	*	*	*		10
SE	2	2	*		*			5
S	3	1	1	*				5
SW	2	2	1	*				5
W	5	2	1					8
NW	6	3	1					10
ALL	30	19	7	1	*	*		

NO. OF OBS. 806

* OCCURRED BUT LESS THAN 0.5 PERCENT

PRODUCED BY M.I.S.S. 29/ 7/86

FRAME C12

BILLITON AUSTRALIA GOLD PTY. LTD.
MT. TODD JOINT VENTURE
Annual Report for Period Ending 28 January, 1989

APPENDIX XV
Mt. Todd Gold Project - Mine Infrastructure
Site Investigation
- Golder Associates

Golder Associates
GEOTECHNICAL ENGINEERS

MT TODD GOLD PROJECT

MINE INFRASTRUCTURE

SITE INVESTIGATION

88638252(C) JANUARY 1989

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TABLE OF CONTENTS

1. INTRODUCTION

Golder Associates were requested by Minproc Engineers Pty Ltd, acting on behalf of Billiton Australia Gold Pty Ltd, to carry out site investigation work for the mine infrastructure development, incorporating the following items:

- **Crushing Plant**

  - Two large crushers with associated screens, agglomeration drum, and dump hopper. Administration buildings and workshop buildings are proposed immediately south of the crusher line. An absorption/ desorption plant is also proposed in this area with unit foundation loads of the order of 100kn.

- **Leaching Stockpiles**

  - Stockpiles on the initial leach pads (Nos. 1 & 2) are to be constructed in two lifts to a height of 12m. Future leach pads (Nos. 3, 4 and 5) will be constructed on the alluvial flood plain approximately 1km east of the proposed crusher line and leach pads 1 & 2.

- **Process Ponds**

  - Two fully lined process ponds are proposed south of leach pad No. 2. The ponds will have 5m high embankments.

- **Raw Water Dam**

  - A 10m high earth embankment is proposed, located approximately 3km north-west of the crusher line.

- **Stow Creek Crossing**

  - A new access road is proposed off Edith Falls Road into the site, and a culvert crossing will be required at Stow Creek.

Investigation work was carried out by excavation of 21 test pits on 5 and 6 December, 1988. This report sets out details of field work and laboratory testing together with engineering comment and geotechnical design parameters.

2. THE SITE AND LOCAL GEOLOGY

The Mt Todd gold deposits (named Batman and Robin) are located approximately 50km north-east of Katherine. The main deposit (Batman) is located within a group of hills rising 60m above an alluvial flood plain to the east. The proposed crusher line is located at the south-eastern base of these hills with leach pad Nos. 1 & 2 on slightly sloping ground covered with light scrub and medium height trees.

The proposed area of leach pads 3, 4, and 5 is a flat flood plain covered with sparse vegetation and low to medium height trees.

The proposed raw water dam is located within a steep sided gully which was boulder strewn but dry at the time of investigation.

At the location where the proposed access road crosses Stow Creek the creek banks are very steep and rise 2 to 3m above water level. The area to be traversed by the road is generally flat and covered with low to medium height trees.

Information supplied by on-site geologists indicates the whole area to be underlain by greywacke rock, which is generally medium strong to strong and close to surface level at the base of the Batman hill gold deposit.

3. FIELD WORK

The location of the various sites are shown on the site locality plan, Fig. 1. A large excavator (Kobelco K912A) was used to excavate 21 test pits. The location of 18 of these test pits are given on Fig. 2. Test pits 19 and 20 were excavated on the banks of Stow Creek at the proposed crossing site and Test Pit 21 was located 200m along the proposed access road to the east of Stow Creek.

All test pits were logged by an experienced geotechnical engineer and samples taken for laboratory testing.

4. GROUND CONDITIONS

Detailed descriptions of the strata encountered are given in the attached Test Pit Reports using terminology defined in appended notes. The ground conditions are summarised for the various sites as follows:

- **Crusher Plant/Administration and Workshop Buildings**

  - Uniform ground conditions were encountered at the base of the hills comprised of medium dense and dense silty sands and gravels to a maximum depth of 1.0m overlying weak to medium strong rock. The rock was generally fragmented with clean tight joints.

- **Leach Pads Nos. 1 & 2 / Process Ponds**

  - Ground conditions did not vary markedly from above, with the silty sand/gravel strata extending to a maximum depth of 0.8m. A layer of clayey gravel was encountered in Test Pit 13 between 0.6m and 1.2m. The underlying rock was generally very weak to weak.

- **Leach Pads 3, 4 and 5**

  - Although located on an alluvial flood plain, the alluvial materials (silty sand and gravel) overlying rock were still medium dense to dense and extended to a maximum depth of only 1.7m in Test Pit 16. The underlying rock ranged in strength from very weak to medium strong.

- **Raw Water Dam**

  - The soil overlying rock in this area was more cohesive than in the general plant area and is described as rock fragments in a clay matrix. This stratum extended to a maximum depth of 2.5m in Test Pit 5, located some 500m west of the proposed dam embankment in the floor of the gully. On the sides of the gully (Test Pits 2 and 3) rock was encountered at approximately 0.5m. The rock was generally medium strong to strong with clean tight joints and apparently of low permeability. A visual assessment of moisture content profile down the sides of the test pits indicated that water flowing through the gully does not penetrate below 0.5m.

- Stow Creek Crossing/Access Road

- Test Pit 19 on the western bank of Stow Creek encountered stiff alluvial clays and silts to 2.9m overlying very dense sands and gravels to the terminal depth of 5m. In contrast Test Pit 20 on the eastern bank encountered silty clay to only 0.6m overlying rock. Test Pit 21 encountered stiff alluvial clays and silts to the terminal depth of 2.5m. It is most probable that the creek has cut a number of channels in the area through the rock and the old channels have been filled with the cohesive alluvial deposits.

5. LABORATORY TESTING

Particle size distribution tests were carried out on samples of the silty sand/gravel encountered in the majority of the test pits and on a sample of clayey silt taken from the western bank of Stow Creek. Atterberg limits determinations were also carried out on the latter sample. The results are summarised on an attached sheet and grading curves are appended.

The silty sand/gravel contained up to 60% gravel sized particles with 20% sand and 20% silt. The cohesive alluvial deposits on the western bank of Stow Creek were comprised predominantly of low plasticity silt sized particles with 20% fine grained sand.

6. ENGINEERING COMMENT

6.1 Crushing Plant/Administration and Workshop Buildings

Foundations to all plant items along the proposed crusher line could be formed either in the medium dense to dense silty sand/gravel strata; or on the underlying weak to medium strong rock using allowable bearing pressures of 300kPa and 1000kPa respectively. The 1000kPa value is a conservative figure based on weak rock and could be increased if necessary for stronger rock. The rock is fragmented and therefore should be relatively easy to excavate if working from an open face. In a confined excavation (eg. service trench) the rock will be difficult to excavate, as indicated during excavation of the test pits using a large excavator.

It is understood that the administration and workshop buildings, and the absorption/desorption plant will be relatively flexible structures imposing low foundation loads. Strip or pad footings to these buildings could be formed at shallow depth in the medium dense to dense silty sand/gravel stratum using an allowable bearing pressure of 300kPa. Weak to medium strong rock would be encountered at shallow depth (about 1.0m) underlying the buildings and the comments given above on excavation conditions would apply.

6.2 Leach Pad Nos. 1 & 2 / Process Ponds

Some cut to fill earthworks will be necessary in order to form a level pad for the ore stockpiles. The silty sand and gravel layer will be readily excavated and would be suitable for re-use as filling. A compaction level equivalent to 90% Modified compaction is recommended. Fill material should be placed in loose layers no greater than 300mm thick prior to compaction.

The proposed 12m high stockpiles would induce negligible settlement in the natural or compacted foundation materials, and could be formed as steep as practical for leaching purposes, without threatening the stability of foundation materials.

It is understood that excavation depths for the process ponds will be up to 3m. Rock is anticipated below approximately 1m. As noted under Section 6.1, the rock is fragmented and should be readily excavated when working from an open face, using a large excavator or a bulldozer (say up to D6 in size) to rip the rock. In a confined excavation the rock could be very difficult to excavate; possibly requiring the use of air tools or blasting. Excavated rock to be used as general filling either to form leach pads or process pond embankments should be broken down to exclude fragments greater than 200mm.

Embankments to the fully lined process ponds could be formed using the near surface silty sands and gravel (mixed with broken down rock if necessary), compacted to a dry density ratio equal to 90% of Modified compaction. For a 5m high embankment, a battered slope of 1.5 horizontal to 1 vertical is recommended. Embankments formed from silty sand and gravel would be subject to erosion and hence external slopes should be protected. The fragmented greywacke encountered generally below 1.0m would be adequate for this purpose.

6.3 Leach Pads 3, 4, and 5

Similar ground conditions were encountered under proposed leach pads 3, 4 and 5 as encountered under leach pad 2, hence the comments given under section 6.2 would apply.

6.4 Raw Water Dam

The overburden soil encountered in the floor of the gully was comprised of rock fragments in a clay matrix. This material, when compacted in layers to a dry density ratio equal to 95% of Standard compaction (this dictates a slightly higher placement moisture content than Modified compaction), would form a low permeability earth embankment. It will be necessary to determine the extent, and hence available quantity of cohesive material within the gully floor area prior to final embankment design.

Prior to commencement of embankment construction, the embankment foundation area should be stripped of all vegetation and loose boulders, and then proof rolled to remove any soft zones. For a 10m high embankment, battered side slopes of 1.5 horizontal to 1 vertical are recommended. It will be necessary to protect embankment slopes against erosion, preferably by the use of vegetation on external slopes.

The in-situ soil and underlying rock have been visually assessed as being of very low permeability. Hence only minor seepage would be anticipated through the dam floor and embankment wall provided care is taken to use predominantly clayey material in embankment construction; excluding rock fragments and boulders larger than 200mm. In order to quantify the anticipated seepage volumes it would be necessary to establish permeability values for the in-situ fragmented rock and the clay embankment. This facet is important if the plant process is dependent on water supply from the dam.

6.5 Stow Creek Crossing/Access Road

It is understood that the Stow Creek crossing will be formed by placing a number of large culverts in the existing creek bed and covering with soil won locally.

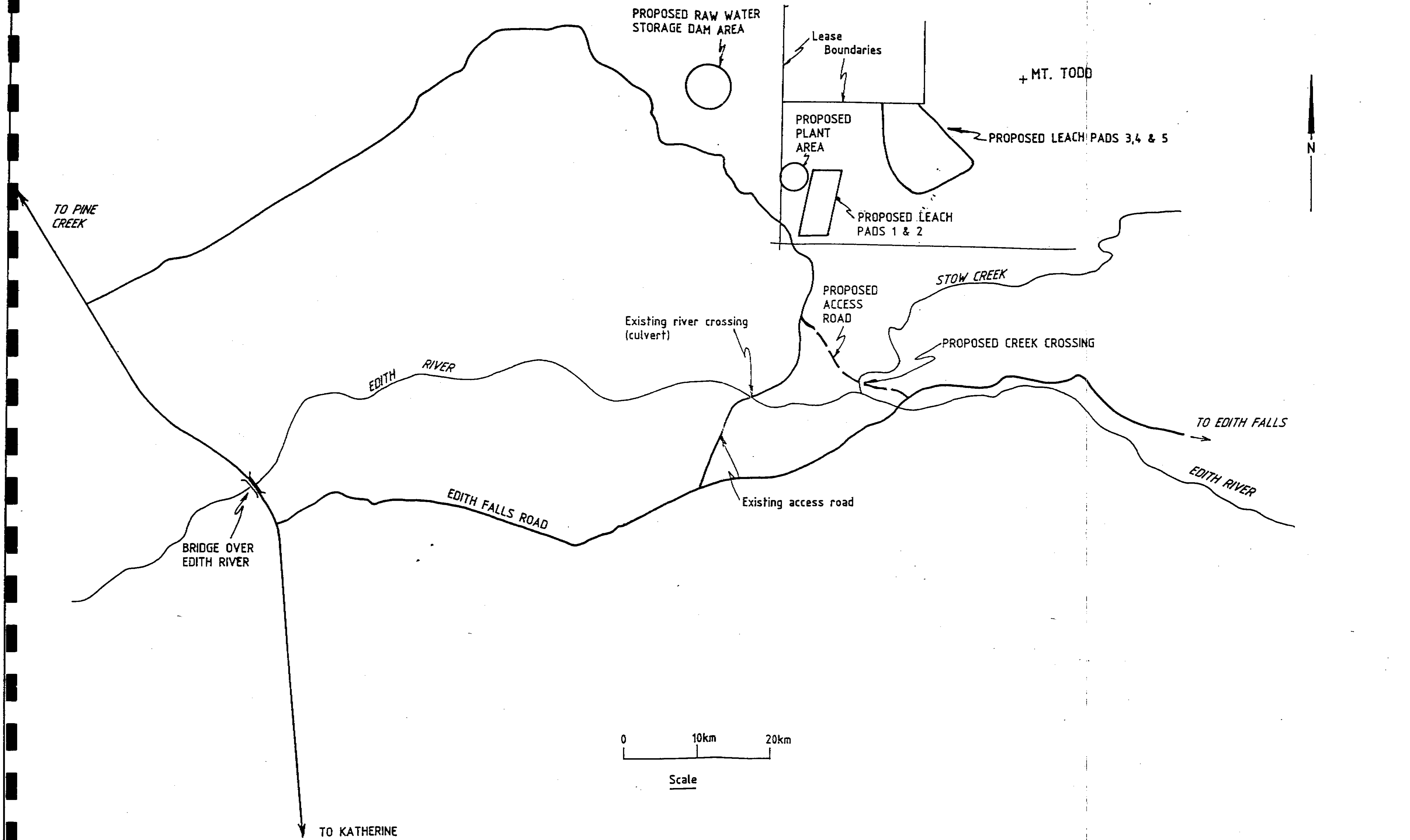
The clayey silt encountered on the western bank of Stow Creek would be highly erodible and hence this material should not be used to fill over culverts. There was evidence of a number of small gravel pits in the area to the east of Stow Creek and it is recommended that either gravel from these pits, or silty sands and gravels such as encountered in the leach pad areas be used for this purpose. (Naturally these materials should be placed in compacted layers). In addition, large rock rip-rap should be placed upstream of the culverts to protect against current erosion during high velocity flow periods.

The worst material likely to be encountered as subgrade material for the proposed access road is the clayey silt/silty clay such as encountered in Test Pits 19 to 21. Based on the laboratory testing carried out, a CBR value of 16% is indicated for this material. It is noted that the material has a high silt content and hence would be impossible to compact when wetted above optimum moisture content. Adequate drainage should be installed along the sides of the road to protect against erosion of the subgrade.

GOLDER ASSOCIATES PTY LTD

per :

David K. Nolan



GOLDER ASSOCIATES
CONSULTING GEOTECHNICAL AND MINING ENGINEERS

MT. TODD GOLD PROJECT
SITE LOCALITY PLAN

Job No 88638252

Scale AS SHOWN

Fig. 1

APPENDIX

TEST PIT REPORT SHEETS

DESCRIPTION OF TERMS USED IN CLASSIFICATION

LABORATORY TEST RESULTS

TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE PROPOSED RAW WATER DAM
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 1, 2 and 3
DATE 5th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing Type	Depth	Results
1	CLAY AND ROCK FRAGMENTS Strong to very strong rock fragments (up to 400mm) in a brown clay matrix	GL			
	TEST PIT TERMINATED AT 1.0m (unable to penetrate)	1.0			
2	CLAY AND ROCK FRAGMENTS Medium strong red rock fragments (up to 300mm) in a stiff clay matrix	GL			
	ROCK (Greywacke) Medium strong, highly fractured rock with tight clean joints - grading strong (highly fractured) below 2.0m approximately	0.6	D	1.0	
	TEST PIT TERMINATED AT 2.5m (unable to penetrate)	2.5			
3	CLAY AND ROCK FRAGMENTS Weak to medium strong rock fragments (up to 300mm) in a stiff red clay matrix	GL			
	ROCK (Greywacke) Medium strong, red, highly fractured - grading strong to very strong, highly fractured (tight, clean joints) below 0.9m	0.5			
	TEST PIT TERMINATED AT 1.5m (unable to penetrate)	1.5			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	4, and 5
SITE	PROPOSED RAW WATER DAM	**DATE**	5th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
4	<u>CLAY AND ROCK FRAGMENTS</u> Weak grey rock fragments in a stiff clay matrix	GL			
	<u>ROCK</u> (Greywacke) Medium strong, grey, highly fractured clean tight joints - ironstained in places	0.5			
	<u>TEST PIT TERMINATED AT 2.5m</u> (near pentration refusal) Note: water not penetrating below 0.5m - very damp above 0.5m then dry	2.5			
5	<u>CLAY AND ROCK FRAGMENTS</u> Alluvial gravel (100-200mm - rounded) in a red clay matrix - tight and difficult to excavate	GL			
	<u>CLAY AND ROCK FRAGMENTS</u> - as above with colour change to grey	1.5			
	<u>ROCK</u> (Greywacke) Strong, grey highly fractured, clean tight joints	2.2			
	<u>TEST PIT TERMINATED AT 2.8m</u> (very difficult to penetrate)	2.8			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

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U — undisturbed sample
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PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE PROPOSED CRUSHER LINE
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 6
DATE 5th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
6	<u>SILTY SAND/GRAVEL</u> Apparent medium dense to dense, brown	GL			
	<u>ROCK</u> (Greywacke) Weak grey, highly fractured (clean tight joints)	0.8			
	<u>ROCK</u> (Greywacke) Weak to medium strong, grey, highly fractured (clean tight joints)	2.5			
	<u>TEST PIT DISCONTINUED AT 4.5m</u> (very difficult to penetrate)	4.5			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

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U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD
SITE	PROPOSED CRUSHER LINE
LOCATION	MT TODD GOLD PROJECT
TEST PIT NO.	7, 8 and 9
DATE	5th December, 1988
PROJECT NO.	88638252
SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing Type	Depth	Results
7	SILTY SAND/GRAVEL Apparent medium dense to dense, brown	GL			
	ROCK (Greywacke) Weak, grey, fragmented (clean tight joints) - grading weak to medium strong below 2.5m approximately	0.5			
	TEST PIT TERMINATED AT 3.5m (very difficult to dig)	3.5			
8	SILTY SAND/GRAVEL Apparent dense red brown	GL			
			D	0.50	
	ROCK (Greywacke) Weak, grey, fragmented (clean tight joints)	0.9			
			D	2.50	
	TEST PIT TERMINATED AT 3.5m (difficult to dig)	3.5			
9	SILTY SAND/GRAVEL Apparent medium dense to dense red-brown	GL			
	ROCK (Greywacke) Weak, grey fragmented (clean tight joints) - grading weak to medium strong below 2.5m approximately	1.0			
	TEST PIT TERMINATED AT 3.0m	3.0			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
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U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE ADMINISTRATION/WORKSHOP BUILDING
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 10
DATE 6th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
10	SILTY SAND/GRAVEL Apparent medium dense to dense brown - gravel size up to 50mm	GL			
	ROCK (Greywacke) Weak grey, fragmented (clean tight joints) - grading weak to medium strong below 1.8m approximately	0.8			
	TEST PIT TERMINATED AT 3.0m (very difficult to dig)	3.0			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

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U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	11
SITE	POTENTIAL C.I.L PLANT	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
11	<u>SILTY SAND/GRAVEL</u> Apparent medium dense to dense, red (gravel size to 100mm) - grading very dense below 0.8m	GL			
	<u>ROCK</u> (Greywacke) Weak to medium strong grey fragmented (clean tight joints) - grading to medium strong below 2.0m	1.4			
	<u>TEST PIT TERMINATED AT 2.5m</u> (unable to penetrate)	2.5			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

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U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	12, 13 and 14
SITE	LEACH PAD NO. 2	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	**DESCRIPTION OF STRATA** Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
12	SILTY SAND/GRAVEL Apparent medium dense, brown (up to 50mm)	GL			
	ROCK (Greywacke) Weak, grey, fragmented, (clean tight joints) - grading weak to medium strong below 1.5m approximately	0.4			
	TEST PIT DISCONTINUED AT 3.5m (very difficult to dig)	3.5			
13	SILTY SAND/GRAVEL Apparent medium dense, brown	GL			
	CLAYEY GRAVEL Apparent medium dense to dense, red (gravel up to 50mm)	0.6			
	ROCK (Greywacke) Weak, grey fragmented (clean tight joints) - grading weak to medium strong below 2.5m approximately	1.2			
	TEST PIT DISCONTINUED AT 3.2m (difficult to dig)	3.2			
14	SILTY SAND/GRAVEL Apparent medium dense to dense, brown (gravel up to 50mm)	GL			
	ROCK (Greywacke) Very weak, grey, fragmented	0.6			
	ROCK (Greywacke) Weak, grey, fragmented (clean tight joints) - grading weak to medium strong at 2.6m approximately	1.3			
	TEST PIT DISCONTINUED AT 3.0m	3.0			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	15
SITE	LEACH PAD NO. 2	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
15	SILTY SAND/GRAVEL Apparent medium dense to dense, red-brown (gravel up to 50mm)	GL			
	ROCK (Greywacke) Weak to medium strong grey, highly fractured (clean tight joints) - grading medium strong below 1.5m approximately	0.8			
	TEST PIT TERMINATED AT 2.5m (unable to penetrate)	2.5			

EQUIPMENT Excavator Kobelco K912A

SUPERVISOR D Nolan

GROUND WATER Not encountered

REMARKS -

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U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST

A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

Golder A

TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE LEACH PADS 3 TO 5
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 16, 17 and 18
DATE 6th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing Type	Depth	Results
16	SILTY SAND Apparent medium dense, yellow brown	GL			
	- with some fine gravel below 0.5m				
	SILTY SAND/GRAVEL Very dense (difficult to dig) red brown	0.8			
	ROCK (Greywacke) Medium strong, grey, highly fractured (tight clean joints)	1.7			
	TEST PIT TERMINATED AT 2.0m (unable to penetrate)	2.0			
17	SILTY SAND/GRAVEL Apparent medium dense, red-brown (gravel up to 10mm)	GL			
	ROCK (Greywacke) Very weak, grey, fragmented (clean tight joints) - grading weak to medium strong below 2.2m approximately	0.7			
	TEST PIT TERMINATED AT 2.5m	2.5			
18	SILTY SAND/GRAVEL Apparent medium dense, yellow brown (gravel up to 10mm)	GL			
	ROCK (Greywacke) Very weak, grey, fragmented (clean tight joints) - grading weak to medium strong below 1.5m approximately	0.5			
	TEST PIT TERMINATED AT 2.5m (very difficult to dig)	2.5			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

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D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined compression strength)

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

Golder Associates

TEST PIT REPORT

CLIENT MINPROC ENGINEERS PTY LTD
SITE PROPOSED STOW CREEK CROSSING
LOCATION MT TODD GOLD PROJECT

TEST PIT NO. 19 and 20
DATE 6th December, 1988
PROJECT NO. 88638252
SURFACE LEVEL

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing: Type	Depth	Results
19 West Bank	SILTY CLAY/CLAYEY SILT Stiff, brown and yellow brown - grading very stiff to hard below 0.6m - with fine sand	GL			
			D	1.0	
	SILTY SAND Very dense grey fine grained indurated bands - alluvial gravel layers (up to 100mm) below 3.8m	2.9			
	TEST PIT TERMINATED AT 5.0m	5.0			
20 East Bank	SILTY CLAY/CLAYEY SILT Stiff yellow brown, with some fine sand and gravel	GL			
	ROCK (Greywacke/Mudstone/Siltstone) Extremely to very weak grey, fragmented with ironstaining - grading weak to medium strong below 1.5m approximately	0.6			
	TEST PIT TERMINATED AT 2.5m	2.5			

EQUIPMENT Excavator Kobelco K912A
GROUND WATER Not encountered
REMARKS -

SUPERVISOR D Nolan

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

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TEST PIT REPORT

CLIENT	MINPROC ENGINEERS PTY LTD	**TEST PIT NO.**	21
SITE	ACCESS ROAD - 200m EAST OF STOW CREEK	**DATE**	6th December, 1988
LOCATION	MT TODD GOLD PROJECT	**PROJECT NO.**	88638252
		SURFACE LEVEL	

Test Pit No.	DESCRIPTION OF STRATA Interpreted from field and laboratory testing	Depth (m)	Sampling and In-Situ Testing		
			Type	Depth	Results
21	<u>SILTY CLAY/CLAYEY SILT</u> Stiff, yellow brown with a trace of gravel - grading very stiff to hard and red and yellow-brown below 0.6m - apparent cementation with some sand and alluvial gravel below 2.0m approximately	GL			
	<u>TEST PIT TERMINATED AT 2.5m</u> Shallow gravel pits on high ground in surrounding area	2.5			

EQUIPMENT	Excavator Kobelco K912A	SUPERVISOR	D Nolan
GROUND WATER	Not encountered		
REMARKS	-		

A — auger sample
D — disturbed sample
U — undisturbed sample
SP — Scala Penetrometer
PP — Pocket Penetrometer
Su — shear strength (i.e. ½ unconfined

SCALA PENETROMETER TEST
A 9kg hammer with a 0.5m free fall is used to drive a 30° apex angle cone of 0.020m diameter, followed by smaller diameter rods. Results are recorded in blows per 100mm penetration. Where tests are extended beyond 1.5m penetration, excavation is carried out prior to continuing the test.

Golder Associates

DESCRIPTION AND CLASSIFICATION OF SOIL

GENERAL

Description and classification of soil and rock are based on definitions and systems outlined in Australian Standard AS1726, SAA Site Investigation Code 1975 and its appendices A-D 1978. Description of rock fracturing is based on a system set out by the Sydney Group of the Australian Geomechanics Society, 1975.

DESCRIPTION AND CLASSIFICATION OF SOIL

Soils are classified on the basis of predominating grain size, modified by other significant grain size or sizes present (e.g. CLAYEY SAND) on the following basis:

Classification	Particle Size
CLAY	less than 0.002mm
SILT	0.002-0.06mm
SAND	
fine sand	0.06-0.2mm
medium sand	0.2-0.6mm
coarse sand	0.6-2.0mm
GRAVEL	
fine gravel	2-6mm
medium gravel	6-20mm
coarse gravel	20-60mm
COBBLES	60-200mm
BOULDERS	greater than 200mm

COHESIVE SOILS are described in terms of consistency, colour and structure with comments on minor constituents or apparent special features. Consistency is based on the shear strength of the soil, and is generally estimated from experience, measured by hand penetrometer or determined by laboratory testing. Terms used in describing consistency are set out below:

Term	Shear Strength
VERY SOFT	less than 12kPa
SOFT	12-25kPa
FIRM	25-50kPa
STIFF	50-100kPa
VERY STIFF	100-200kPa
HARD	greater than 200kPa

NON-COHESIVE SOILS are described in terms of relative density, colour, with comments on minor constituents or apparent special features. Relative density or density index is generally based on standard penetration testing (AS1289 Test F3.1), or other forms of penetration testing. Terms used in describing relative density are set out below:

Term	Relative Density	SPT "N" Values blows/300mm
VERY LOOSE	less than 15%	less than 5 blows
LOOSE	15-35%	5-10 blows
MEDIUM DENSE	35-65%	10-30 blows
DENSE	65-85%	30-50 blows

ENGINEERING CLASSIFICATION OF ROCK[1]

ROCK STRENGTH

DESCRIPTION TERM	GENERAL FIELD GUIDE	Is (50)[2] MPa	AS1726 ROCK STRENGTH	APPROX.[3] q_u MPa
Extremely Weak	Easily remoulded by hand to a material with soil properties		Extremely Low	
		0.03		0.7
Very Weak	May be crumbled and fragmented by hand		Very low	
		0.1		2.4
Weak	A piece of core 150mm x 50mm diameter may be broken by hand and easily scored with a knife. Sharp edges of the core may be friable and break during handling. Lumps of rock crumble with light hammer blow		Low	
		0.3		7
Medium Strong	A piece of core 150mm x 50mm diameter may be broken by hand with considerable difficulty but may be scored with a knife. Core may be broken with a light hammer blow.		Medium	
		1		24
Strong	A piece of core 150mm x 50mm diameter cannot be broken by hand, but can be slightly scratched or scored with a knife. Core may be broken with a blow from a hammer.		High	
		3		70
Very Strong	A piece of core 150mm x 50mm diameter may be broken with a heavy hammer blow, but cannot be scratched by a knife.		Very high	
		10		240
Extremely Strong	A piece of core 150mm x 50mm diameter is difficult to break with a hammer and rings when struck		Extremely High	

Notes:
1. Based on AS1726 - 1981 - SAA Site Investigation Code.
2. Point Load Strength Index : ISRM Committee on Laboratory Tests, Document No. 1, October 1972.
3. The approximate unconfined compressive strength (q_u) is based on an assumed ratio to the point load index of 24:1. This ratio may vary widely.

RQD: ROCK QUALITY DESIGNATION (expressed as a percent) for core recovered from borehole, defined as :

$$\frac{\text{sum of sound core pieces 100mm or more long*}}{\text{total length of section considered}}$$

* core fractured by drilling process considered unbroken

FRACTURING

DESCRIPTIVE TERM	GENERAL FIELD GUIDE
Fragmented	The core is comprised primarily of fragments of length less than 20mm and mostly of width less than the core diameter. Generally >50 breaks/metre.
Highly Fractured	Core lengths are generally 20-40mm with occasional fragments. Generally 25-50 breaks/metre.
Moderated Fractured	Core lengths are generally 30-100mm with occasional shorter and longer section. Generally 10-30 breaks/metre.
Fractured	Core lengths are generally 80-400mm with occasional shorter and longer sections. Generally 3-12 breaks/metre.
Slightly Fractured	Core lengths are generally 300-1000mm with occasional shorter and longer sections. Generally 0-3 breaks/metre.
Unbroken	The core does not contain any natural breaks.

Note: This classification applies to drill cores and refers to the spacing of all types of natural fractures along which the core is discontinuous (i.e. natural breaks). Such natural fractures include joints, bedding plane partings and other natural rock defects but exclude artificial fractures such as breaks caused by drilling and boxing core. Natural unbroken defects such as tight joints and veins along which no fracture is present are also excluded.

- Modified from draft prepared by the Sydney Group of the Australian Geomechanics Society, January 1975.

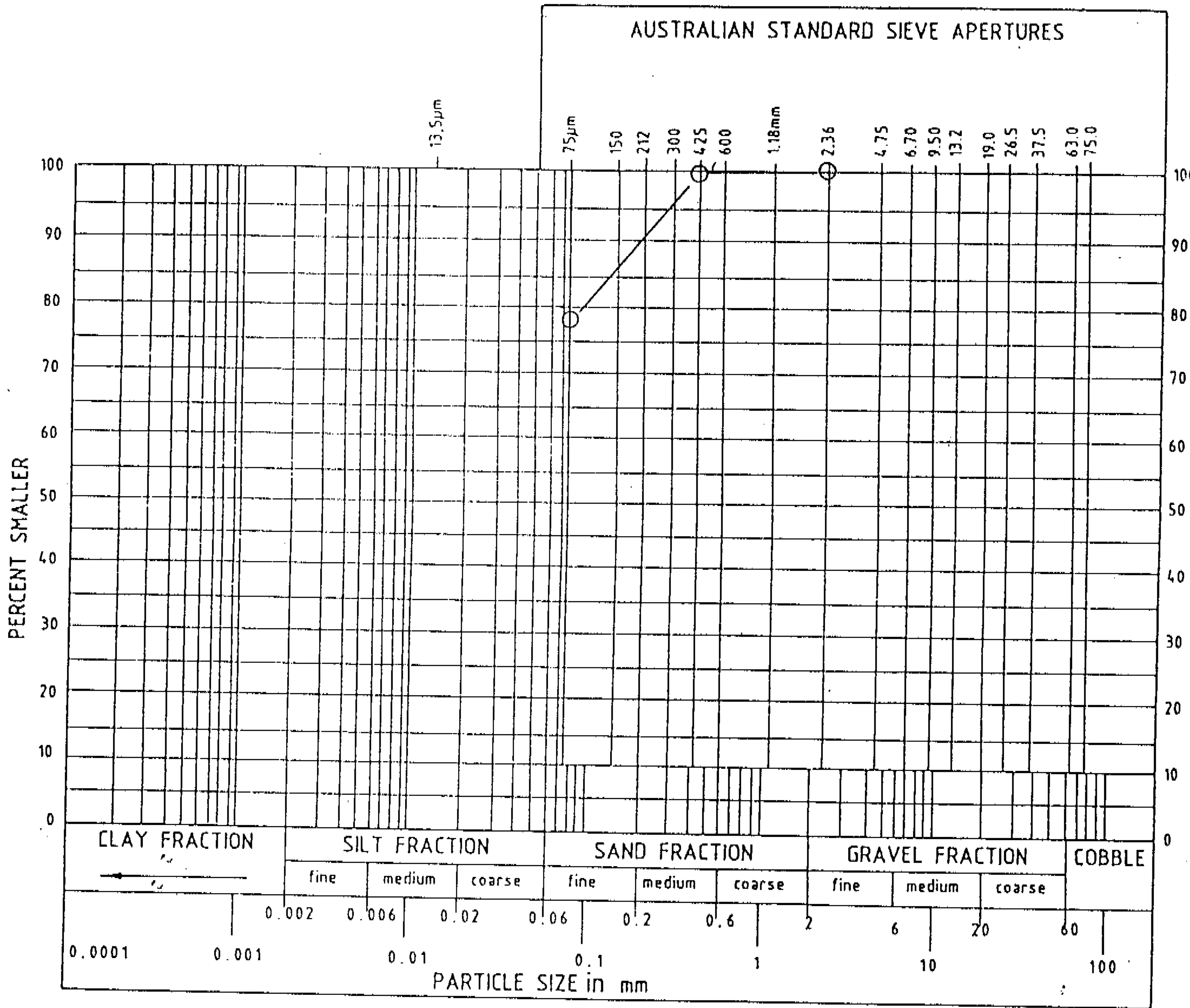
MT TODD GOLD PROJECT

MINE INFRASTRUCTURE

SITE INVESTIGATION

88638252(B)

TEST No.	DEPTH (m)	DESCRIPTION	IN-SITU CONDITION	PLASTICITY		LINEAR SHRINKAGE	ESTIMATED CBR*	GRADING		
			Moisture Content %	Liquid Limit %	Plasticity Index %	%	%	pass 2.36mm %	pass .425mm %	pass .075mm %
TP8	0.5	Silty sand and gravel	2.5	-	-	-		43	35	25
TP19	1.0	Clayey silt with some fine sand	2.6	23	6	2.0	16	100	100	79



DESCRIPTION Clayey silt with some fine sand

TEST TYPE AS1289 B1.1, C1.2, C2.1, C3.1, C4.1, C6.1

PRE-TREATMENT NIL

NATURAL MOISTURE CONTENT 2.6%

LIQUID LIMIT 23%

PLASTIC LIMIT 17%

PLASTICITY INDEX 6%

LINEAR SHRINKAGE 2.0% Mould length 125mm

RESULTS OF CLASSIFICATION TESTS

SITE MT TODD GOLD PROJECT

CLIENT MINPROC ENGINEERS PTY LTD

EXCAVATION No. TP19

DEPTH 1.0m

SIGNATORY [signature]

DATE 20/1/89

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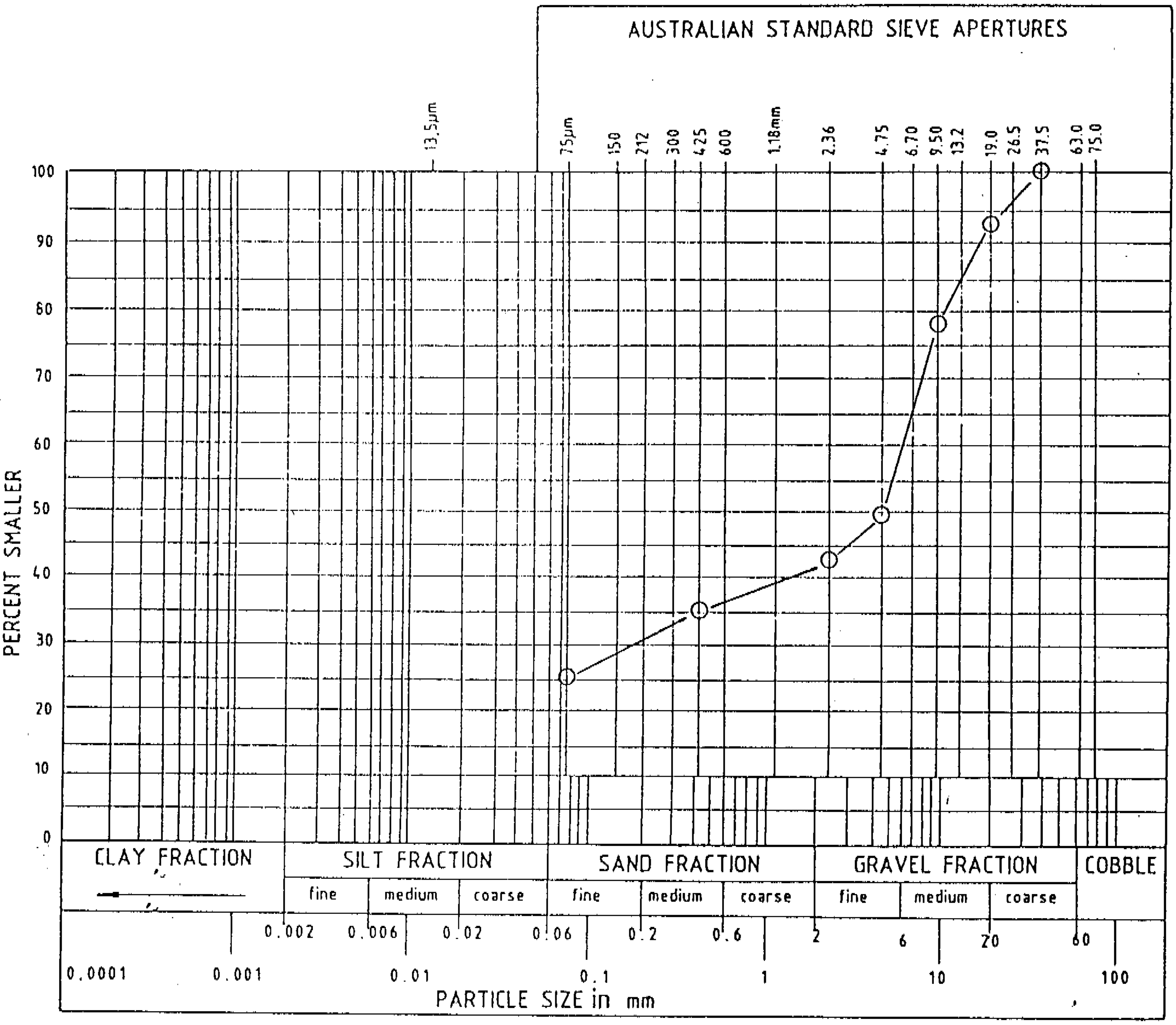
REPORT No. N88-537/1

JOB No. 88638252

DATE 14.12.88

LABORATORY BRISBANE

Golder Associates



DESCRIPTION Silty sand and gravel

TEST TYPE AS1289 B1.1, C6.1

PRE-TREATMENT NIL

NATURAL MOISTURE CONTENT 2.5%

LIQUID LIMIT -

PLASTIC LIMIT -

PLASTICITY INDEX -

LINEAR SHRINKAGE - Mould length -

RESULTS OF CLASSIFICATION TESTS

SITE MT TODD GOLD PROJECT

CLIENT MINPROC ENGINEERS PTY LTD

EXCAVATION No. TP8

DEPTH 0.5m

SIGNATORY

DATE 20/1/89

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JOB No. 88638252

DATE 14.12.88

LABORATORY BRISBANE

Golder Associates