ANGLOGOLD AUSTRALIA LIMITED

ANNUAL EXPLORATION REPORT

MINERAL LEASE (NORTH) 1109
UNION REEFS GOLD MINE

Period 01/10/2003 to 31/12/2003

DATE:	**16 March 2004**	**REPORT NO:**	**NT12672**
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SUMMARY

After consultation with the Titles Division of the Department of Business, Industry & Resource Development and with NTGS in March 2004 it has been determined that his report represents a partial Annual Report to cover the period from 01/10/2003 until 31/12/2003 for MLN1109. The Union Reefs Gold Mine workings and treatment facilities are located upon this mining tenement.

It is noteworthy that for many years the spatial position of MLN833 is in the central portion of the western wall of the Crosscourse Pit. As such, it has not been feasible to prepare a separate mining or exploration report and it is not specifically referred to in the following document for the same reason.

The Appendix 1 document entitled *MINE CLOSURE REPORT Union Reefs Gold Mine (Pine Creek)* has been previously lodged with the Mines Division at DBIRD on 09/11/03 and represents a detailed review of the operations throughout the ownership of AngloGold Australia Limited.

The primary aim of this report was to compile a single condensed summary that wholly encompasses all works carried out to date pertaining to this operation. This document is factual figures with some annotation/interpretation offered on certain aspects particularly with relation to changes in Block models, Pit Designs and the like.

Tenure

Tenement	Granted	Region	Annual Exploration Report Due
MLN 1109	16-Dec-93	Union Reefs Mine	31-Mar-04

Proposed Exploration Expenditure

It is not planned for any further exploration expenditure by AngloGold on MLN1109 in the period following the date of the announced closure. An agreement for the sale of the Union Reefs Mine to Greater Pacific Gold Ltd was announced on 14 November 2003 and it is expected that any further exploration expenditure will be incurred by Greater Pacific Gold Ltd after handover.

Warren Steward.
16 March 2004.

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APPENDIX 1

Mine Closure Report
Union Reefs Gold Mine
(Pine Creek)
November 2003

Mine Closure Report
Union Reefs Gold Mine
(Pine Creek)

Written by

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Bill Makar
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Tobermoray
Wellington
Orinoco
Alta Pit
Dam A Pit
North Waste Dump
Union North
Union North South
Dam C
West Waste Dump
Low Grade Stockpiles
Prospect Claim Pit
Lady Alice Pit
Togar Tailings
ROM
Crosscourse Pit
East Waste Dump
Plant and Admin
Crosscourse South Pit
Union South Pit
Ping Que South Pit
Tailing Storage Facilities
Big Tree Pit
Temple Pit
Millars Pit
URGM Aerial Photo September 2002

Executive Summary

The last load of ore from the Union Reefs Gold Mine was mined at 1120am on Sunday the 27th of July 2003. Remedial work continued with the contractor until the end of the month pertaining to rehabilitation.

A total of 765,827 metres of grade control were drilled, primarily on a 10m x 4m Spacing at a 60 degree angle. An Earth Saw was also used towards the end of mine life to better define low grade and mineralised waste stockpiles.

A total of 3,187 waste rock samples were taken. Approximately 25% of which returned high levels of Arsenic and or NAG results. Problematic waste from these areas was, wherever possible, directed to the core of waste dumps to ensure complete encapsulation.

Typical powder factors used at Union Reefs varied from $0.20kg/m^3$ up to $0.80kg/m^3$ for different rock types.

The drill and blast database only dates back to the 1st of October 1997. Since then, approximately 2,412,399 metres were drilled in 299,422 blast holes yielding 27,325,220 BCM using 17,062,654 kg of explosive for an average powder factor of $0.62kg/m^3$. In addition, there were a further 312,470 metres drilled in 43,810 blast holes that were either not loaded or redrilled/blown out due to hole collapse. Of the 299,422 holes blasted, 48,293 (15%) were of 5m Bench height, 203,340 (63%) were of 7.5m of Bench height, and 47,789 (12%) were of 10m bench height.

Over 180 litres per second against a vertical head up to 130m has been succesfully pumped from the lower levels of Crosscourse pit during the height of the wet seasons. This was pumped to an electric pump station (2 x 110kW pumps) situated on the Southeast corner of the ramp at the1080mRL.

Almost 17mt of ROM ore (@1.63 g/t), 1.5mt of Leach ore (@0.86g/t) and over 3.6mt of mineralised waste (@0.53g/t) was mined throughout the life of the Union Reefs Gold Mine. This, alongside 92mt of waste, produced an average strip ratio of 5.29:1

Total Drill and blast costs (from 1997 onwards) averaged $1.46 per BCM (inclusive of drilling, priming, loading, stemming, bulk explosive and accessories).

Approximately $347m worth of combined Administration, Geology, Mining, Processing, and Maintenance costs were incurred from 1996 onwards. This equates to $8.82 per BCM mined, $18.90 per tonne milled, or $432.84 per ounce produced.

Upon Reserve Reconciliation, Ore Mined was 108% of the tonnes, 87% of the grade and 94% of the ounces. Milled Tonnes represent 98% of tonnes mined, milled grade represents 102% of predicted mined grade and recovered ounces represent 99% of predicted mined ounces. Overall, reconciliation data is incomplete due to poor archiving of old data, moreover changes in reconciliation procedures. During 1997 and 1998, reconciliations were only done on ore above the 0.9g/t cut-off even though ore above 0.55g/t was stockpiled and later milled. **The final As-Mined figure is only the ore that has been reconciled against the same reserve parcel at a certain cut-off used at that time. Final As-Mined figures do not represent total ore mined and hence total ore milled. Total ore mined is presented in the mining statistics section of this report.**

Pit by Pit reconciliations are not possible due to ore being stockpiled by grade rather than pit. Moreover, ore is blended by grade to achieve the budgeted milled grade. Mine to Mill reconciliations are based on Total Mined ore (mined to the ROM) and Total Milled ore. Note that Tonnes Mined are based on volumes converted to tonnes and Tonnes Milled are calculated by weightometer readings from the mill and adjusted with the removal of moisture.

Although the majority of the easily won gold has been mined at Union Reefs there remains further potential within the existing tenements MLN 1109 and ERL 130. The remaining resource at $700/oz gold for URGM as reported in the September 2003

Reported Statement is 1.513mt @ 1.81g/t Au. 253kt @ 2.73g/t Au remains as an open pit potential at Prospecting Claim and 1.26mt @ 1.62g/t Au in two open pit deposits at Esmeralda. Further exploration at Esmeralda is likely to expand the current resource. In addition small targets may be developed on MLN 1109 with additional exploration work; northern strike extension of Lady Alice, possible a small high grade show at Wellington, remnant ore at the north end of Crosscourse. If the current gold price continues to rise, the Togar tailings and the Tobermoray alluvials may become economical.

A viable underground resource potential exists at URGM. The main underground target is below Prospecting Claim with preliminary work showing a resource of over 100kt @ 11.7g/t Au and open at depth. Revue of the mine lease deposits has highlighted that other viable underground targets may exist below Lady Alice (40kt @ 6.9g/t Au), Western Lens at the north end of Crosscourse (18kt @ 14.5g/t Au), and Union South (20kt @ 12g/t Au). All these targets are open at depth. Wellington and Millars may also prove up to contain small but high-grade resources.

Union North and Crosscourse pits have both been tailed into, sterilizing any chance of extending or deepening the pits or chasing underground potential. In Crosscourse the tailing level should be below the 1080mRL. This would have effectively sterilized any underground potential associated with the steeply plunging E Lens pod which may have proved to be a significant resource grading at better than 4g/t Au. The resource associated with the Western Lens is above the tailings. Any potential at Union North would have been sterilized as it was filled with tailings to near surface.

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1.0 Introduction

Union Reefs Gold Mine is located approximately 185km SE of Darwin, NT, at latitude 13°42'S, longitude 131°47'E and AMG coordinates 801,700E, 8,482,000N. The tenement (MLN1109) and contained resources are 100% owned and operated by AngloGold Australia Limited [(1)]

The primary aim of this report was to compile a single condensed summary that wholly encompasses all works carried out to date pertaining to this operation. This document is predominantly factual figures based with some annotation / interpretation offered on certain aspects particularly with relation to changes in Block models, Pit Designs and the like.

The primary source of information were the various access database systems including, but not limited to, Grade Control, Geotechnical, Prism Monitoring, MINEMAN, Drill and Blast, Quality Control, and METREP.

A digital copy of this report, along with the various appendices referred to hereafter, and all of the files and databases used in conjunction with this report, have been burnt to CD (wherever possible). This can be made available through Perth head office.

2.0 Geology

The Union Reefs Mining Lease (MLN 1109) is located within a corridor of Palaeoproterozoic metasediments that are flanked to the east and west by lodes of the Cullen Batholith (refer to Figure 2.1.1).

Gold mineralisation is believed to be hypothermal-mesothermal and involved a multi-staged (sedimentary preparation, metamorphic upgrading and hydrothermal mobilisation generated by heat of the batholith), long term process as represented by a number of quartz vein generations (Klominsky, et al., 1996).

2.1 Local Geology

Gold-bearing lodes of the Union Reefs District are confined to the 300m wide Pine Creek Shear. Economic mineralisation is related to a tightly interbedded sequence of weakly carbonaceous shales and greywackes of the Burrell Creek Formation. Two 'lines-of-lode' exist in which numerous historic old workings are centered. The most productive structure is known as the Union Line with a subordinate structure to the East (mine grid), known as the Lady Alice Line. The lodes are typical of those characterized as 'shear-related' but they do host small saddle reefs as well.

Gold mineralisation is associated with quartz-sulphide veining. The location and style of veining throughout the deposit is a complex interplay of structural and lithological controls. Three end member vein styles are recognised. These are:

Lode style veins, which are up to 4 m thick, commonly discontinuous, pod-like and hosted within highly sheared, dominantly shale wall rocks. Lode style mineralisation displays the largest amount of grade variability at URGM and includes localised zones of high grade gold. The majority of the old workings at URGM are located on these systems.

Stockwork vein systems, which are complex and largely restricted to greywacke dominated horizons. Stockwork veining is typically of moderate gold grade; and

Sheeted-vein systems, which are characterised by sub-parallel vein sets that typically occur in thinly interbedded sequences of shale and greywacke. Sheeted veining is typically of lower grade.

The Crosscourse zone, which hosts the majority of the gold in the URGM mineralisation, is dominated by the stockwork vein style with lode-style veins concentrated in the Ping Ques and Western Lens systems.

Coarse gold is a characteristic feature of the mineralisation at URGM and occurs as single grains and clusters up to 5 mm across. Geologists have carefully modelled 26 lodes in the mine area. These weakly sulphidic lenses range in width from 1-75m and in strike length from 30-200m. The down-dip extension of the best lode (E-Lens), is undefined but in excess of 300m. Most other lenses have a plunge component, usually to the north, of 100-150m.

The lenses are composed of quartz, carbonate, chlorite, sericite and broken or brecciated wall rock. Most major veins are bedding parallel but several linkage vein sets occur and some areas are characterized by sheeted vein sets and deformed stockwork veinlets. Boudins are common. Post mineral faulting has not had a negative effect on ore block mineability although the deformation history is complex.

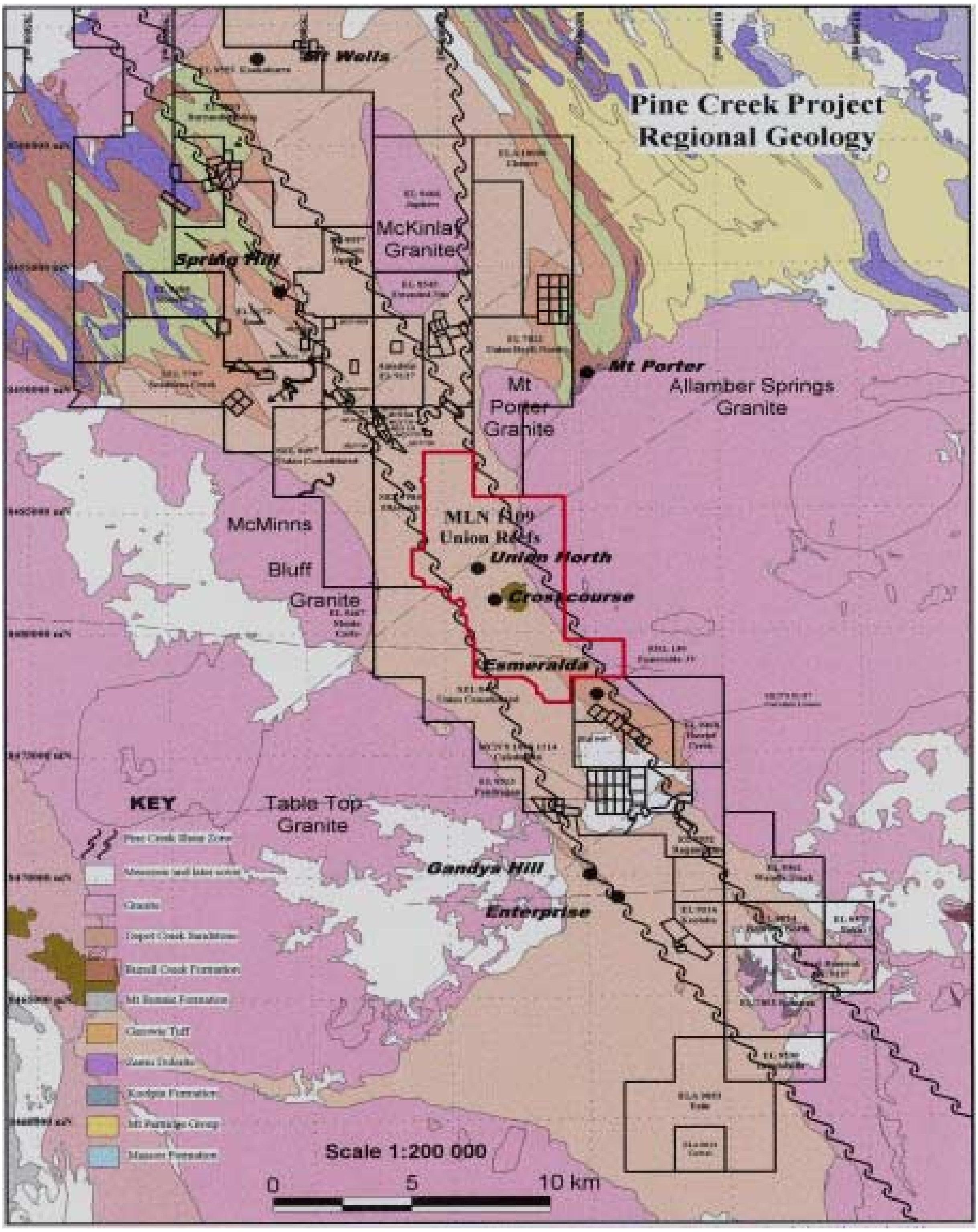
The best pathfinder mineral for gold is arsenopyrite, but pyrite, pyrhotite, sphalerite, galena and sparse copper minerals are also present. The principal styles of sulfide mineralisation include quartz-visible gold banded veining (rare), low sulfide auriferous, pyritic veining (common), weakly banded, auriferous, arsenopyrite-pyrite veining (common), low-grade, disseminated, arsenopyrite-bearing breccias or mylonitic shears (localized), and small, semi-massive base metal pods composed of sphalerite, galena and pyrite (erratic).

The wider quartz lode contacts are featheredge but many of the narrow structures show diffuse contacts. Visual control during mining is subtle in the stringy zones and in the areas of ramifying veinlets. Conditional Simulation techniques were used to smooth the composited data and create a mineable ore block markout during the grade control process.

Lithologically, the Union Reefs quartz vein ore is hosted within interbedded greywacke and shales. The units are variably stratified and generally dip steeply 85 degrees towards the West. Stratification is well defined with continuity down-dip. For designing blasts, the greywacke units are generally 3m to 20m thick, and the shale units are generally 1 to 30m thick. The units are generally interbedded and intercalated.

The greywackes are relatively competent, of high strength (100Mpa) but are closely bedded (300mm) resulting in fair quality rock. By contrast, the chloritic, phyllitic shales are relatively incompetent of moderate strength (65Mpa) and finely (5mm) sub vertically foliated, sheared and laminated resulting in poor quality rock. The smooth planar to undulating foliation present exhibits very low shear strength with peak shear strength in the range of less than 30Kpa.

Figure 2.1.1: Regional Geology (From Kevin Chen Chows Geology Report)



2.2 Grade Control

The majority of Grade control was completed by Reverse Circulation Drilling. Initially, drilling was done on a 10m x 4m pattern at an angle of 60 degrees. Mid way through the mine life, this was changed to a 10m x 4.5m pattern drilled at 55 degrees to reduce drilling costs. In the final years, grade control drilling reverted back to the original 10m by 4m pattern (at 60 degrees) to reduce chance of hole deviation. Main grade control drilling contractors used were Ausdrill, Gomex Drilling, Hughes Drilling, and Stanley Drilling. Changes in mining bench heights from 5m, 7.5m then finally 10m led to corresponding changes in grade control drilling depths. The first metre of each bench was not sampled to reduce chance of it becoming contaminated once disturbed by mining. The non-sampled metre was then compensated for by drilling an extra metre from the bench above. Minor use of blast hole sampling was done in areas restricted by topography. Grade Control drilling never exceeded 26m (two 10m vertical benches) and drilling to this depth was minimised wherever possible to reduce the chance of hole deviation.

Grade control drilling (RC) was used in conjunction with an earth saw on Mineralised Waste Stockpiles 1 and 2 to test the grade and justify re-handling to the ROM. Table 2.2.1 below illustrates total Grade Control meters drilled by pit.

2.2.1 Grade Control Drill Metres

Table 2.2.1: Grade Control Metres by Pit

Pit	Metres Drilled
Big Tree	2,912
Crosscourse	563,010
Crosscourse South	22,746
Dam A	2,329
Joy Hill	247
Lady Alice	54,060
Millars	12,266
Prospecting Claim	32,614
Ping Que South	12,396
Temple	1,899
Union North	70,387
Union South	16,189
Wimbledon	492
Minerialised Waste	1,280
Total	**765,827**

2.2.2 Assay Procedures

Assay procedures are outlined in Makar *et al* 2003 "URGM 2003 Resource Statement". Procedures and results of Quality Assurance and Quality Control of Grade Control samples are outlined in the report by De Lange 2003 "QA QC Report for Grade Control Assay Data Yearly Report".

2.2.3 Waste Rock

Waste rock procedures, both sample collection and assaying are outlined in the internal report by Hall 1999: "Waste Rock Management Plan – Union Reefs Gold Mine". Samples were not taken at the beginning of each pit due to the extensive oxide rock (no fresh sulphides present) and waste was presumed to be non-problematic. Samples were also not taken on the last benches of each pit design where predominantly fresh material of low strip ratio was encountered that produced very little waste, so was all consequently characterised as problematic.

A total of 3,187 waste rock samples where taken during the mine life at Union Reefs. Of those samples, 800 (25%) returned a problematic result (Arsenic levels equal or greater than 1000ppm and/or NAG result higher than Type 1 (or higher than 1). On Average, Arsenic readings returned a value of 537ppm. Results are outlined in Table 2.2.2 below.

Table 2.2.2: NAG Results

Sample Group	**Samples**	**%**
Total No. of Samples	3187	100%
Total No. of Problematic Samples	800	25%
Problematic Samples		
Samples Greater than or equal to1000ppm As	335	11%
Samples Higher than NAG Type 1	562	18%
Samples Greater or equal to 1000ppm As and greater than NAG Type 1	97	3%

All problematic waste was, wherever possible, directed to the core of waste dumps.

3.0 Geotechnical Considerations

The main geotechnical considerations that concern the stability of any pit are the presence, degree, and direction of major fault zones, joint sets, lineations, and fractures. The Uniaxial Compressive Strength (UCS), and stress fields (including orientation), along with rock type (and properties) are also important. Discontinuities in the rock mass comprise of natural breaks or planes of weakness such as bedding and foliation planes, joints, faults and shears. The orientation, spacing, continuity and shear strength of these discontinuities influence overall rock mass strength and behaviour during blasting and excavation.

In the Union Reefs mine jointing crossing bedding is well developed but lacks continuity and is truncated. Flat dipping shears are clearly visible in certain areas and are spaced at 5 to 10m and dip approximately 20 degrees to the east. Planar failure along the flat dipping shears has initiated toppling failures in the past and is probable (40 –80 % likelihood) in most future areas of mining. Therefore, in these recognised areas, care should be taken when optimising pit shells, designing and blasting the pit.

The presence of joint sets and sub-horizontal shears can be a geotechnical problem, particularly below water level. The less competent shale bands tend to absorb water and disintegrate, allowing freedom of movement for a sliding or toppling failure. This has been a common occurrence at URGM along the west walls of numerous pits. Another common problem has been the overburden remaining around the surface of the pit that swells during heavy rainfall periods of the wet season. The extra weight at the surface is enough to cause damage to large areas of an oxide pit.

Appendix A of the Geotechnical Appendix, contains a presentation and overview of the geotechnical activities at URGM. It contains a geological map of the area and details the geotechnical procedures used in Crosscourse.

3.1 Consultants

There have been numerous consultants guiding the geotechnical progression of Union Reefs Gold Mine. The most recent was Ian McEnhill, who assisted with development of the Prospecting Claim Pit (Appendix B). The earliest known geotechnical research was undertaken through M.P.A. Williams and Associates Pty, Ltd. *Union Reefs Project: Construction of Tailings Disposal and Water Supply Dam – Geotechnical Information.*

The changing face of Geotechnical Risk Management resulted in the inclusion of additional conclusions in the latest geotechnical report (Prospect Open Pit Slope Design Geotechnical Report by I McEnhill). These include recommendations such as group slope hazard searches, various means of notifying personnel of dangers, group discussions of potential threats and solutions, and a process of monitoring slope quality. This report followed a geotechnical drilling program and Litho-structural mapping of the areas surrounding the Prospecting Claim pit, as part of the process of the pit design.

Throughout the life of the mine, there have been ongoing issues with stability. The more recent satellite pits had issues largely attributed to oxide material at the surface. Previous problems have been at the Glacier, to the Northwest of Crosscourse, and toppling failure as in the Union North South pit. The sub-vertical bedding causes problems at various parts of the operation, especially when combined with faults at depth and steep batters. The blocky nature of these rocks reveal the differently orientated joint sets, which can cause wedge failures when exposed to mining and blasting.

Ian McEnhill has provided geotechnical assistance throughout these projects and offered advice on how to deal with the significant safety considerations. Appendix C contains reports completed by Ian McEnhill on site visits, as well as some geotechnical updates prepared by Ian McEnhill and Chris Fowers (site geotechnical person). He also completed a report on Lady Alice in 2002, (Appendix D). Appendix

E contains a report on the Western Cutback in Crosscourse, *West wall clean-up option, overall stability analysis*, 2000, by Ian McEnhill.

Pells, Sullivan and Meynink (PSM) completed a 4-stage geotechnical study on Crosscourse in 1997. Stage 1 involved an assessment of the geological structures and domains in the Crosscourse area. Stage 2 was laboratory testing of the shear strengths of the rock types. Stage 3 was a geotechnical drill program aimed at the designed pit walls and stage 4 was a pit slope design.

Prior to this, Coffey Partners International (CPI) had completed field investigations as part of a feasibility study of Crosscourse, “Geotechnical and Hydrogeological Study,” February 1993. CPI completed Rock Mass mapping of exposures in the area as well as geotechnical logging of 9 drill holes, one of which was primarily geotechnical rather than aimed at the ore zones.

PSM completed a Slope Design Review in June 2000. This study partially determined the feasibility of extending the pit below 1070m RL. The main conclusions of the pit walls (including drainage and groundwater monitoring), were with regard to slope designs and positioning of the ramp.

SRK completed a Geotechnical review on the Western Wall of Crosscourse in August 2000. Their purpose was to review the geotechnical procedures already in place, evaluate current mining options and to provide practical ideas for maximising ore extraction. They determined that the Western Wall comprised of poor quality rock, requiring 15m benches at a 55° slope, with 5m catch berms – an overall slope angle of 45°. They predicted that steepening the slope would lead to sudden and unpredictable, progressive peak shear strength failure.

Appendix F is a summary of MRMR, the Mining Rock Mass Rating, which has been used in geotechnical research/mapping at Union Reefs, *A Geomechanics Classification System for the Rating of Rock Masses in mine Design*.

3.2 Rock Mass & Discontinuity Shear Strength

Shear strength values identified for the Crosscourse pit have been used for the site pit designs. Following are the shear strength values:

Table 3.2.1: Rock Mass and Defect Shear Strengths Applied in Stability Analysis.

Feature	Friction Angle deg	Cohesive Strength (kPa)
Weathered Rock Mass	30	40
Unweathered Rock Mass	40	200
Shear Zones	24	0
Bedding	30	0
Joints	30	0

3.3 Slope Design

The pit designs have been a progressive development, learning from previous shortcomings. A basic style is used for wall angles, with variations being in the ramp location. Some consideration has been given to geotechnical features, such as the west wall of Crosscourse and the Prospecting Claim ramp. The numerous smaller pits were designed for only short-term stability. These pits were monitored closely for any movement and remedial action undertaken if necessary.

Different methods of slope analysis found there would be varying degrees of success for a small range of slope angles. A batter slope of 55° and an overall angle of 44°, with 15m benches, were found to be marginally stable by the more conservative package, and so this became the base design. The benches were 22.5m, with 7.5m blasts. Later, in the oxide pits, the bench heights typically became 10m with 3m berms every bench. This does vary, however, depending on the individual requirements of each pit. The adherence to the design plays a large part in the pit wall stability. It was clearly seen in Prospecting Claim how batters, if pulled properly, were far more stable than those that had been cut too steeply at the crest and the difference made up at the toe. Consequently batter boards were installed by survey at the crest of each bench that the excavator operators were required to follow.

A geotechnical report has been completed prior to commencement of each large pit. Table 3.3.1 below is a list of appendices and their associated pit/project.

Table 3.3.1. Pit geotechnical reports and the appropriate appendix.

Pit / Project	Author	Appendix
Crosscourse	PSM	
Crosscourse - review of Geotech model	PSM	
Crosscourse West Wall	Ian McEnhill	E
	SRK, 2000	
Crosscourse - 1010m RL	Ian McEnhill, 2001	H
	PSM	
Union North	Frank Pothitos	G
Lady Alice	Ian McEnhill	D
Prospecting Claim	Ian McEnhill	B

Appendix I contains two triangulation’s of the Crosscourse Pit surface, showing the wall angles and the location of catch berms. The videos also show the locations of major faults and where prisms have been installed.

Depending on the height of the walls (and hence their required life expectancy), a number of procedures were adhered to. Below the water table in Crosscourse, Ian McEnhill recommended that horizontal drain holes were required to help relieve the pore pressure on the walls, ensure that the final batters are pulled to the design angle (not undercut) and the walls to be scaled. Appendix J contains all the data relating to the installation of ground support in Crosscourse.

3.4 Geotechnical considerations in Blasting

Generally speaking the following geotechnical factors play a role in the outcome of any open pit batter stability

- Geological Structure
- Discontinuity / bedding spacing
- Groundwater conditions
- *Orientation and number of discontinuity sets*
- *Intact rock strength (UCS)*
- *Discontinuity / bedding shear strength*

At Union Reefs there is very little variability in intact rock strength (UCS) and discontinuity shear strength within the weathered and unweathered zones. Discontinuity orientation and the number of discontinuity sets are also generally constant. Therefore, the last three factors (italicised), rarely fluctuate, consequently their influence on batter stability *generally* remains fixed, so will not be considered any further in this report. Table 3.4.1 below lists the other three geological variables, their influence on batter stability once blasted and any undesirable outcomes that may result.

Table 3.4.1: Geotechnical factors influencing batter stability upon blasting

Geotechnical Variable	**Blast Influence**	**Undesirable Blast Outcome resulting in Additional Cost**
Through-going geological structure (faults and shears).	Propagation of explosive gasses and venting.	Overbreak (waste dilution). Underbreak and oversize.
Discontinuity / Bedding spacing.	Fragmentation. Powder Factor.	Oversize. Higher PF than necessary.
Groundwater conditions.	Hinders drilling (hole collapse) and charging. Dewatering required.	Hinders access to bench. Hole collapse (redrills). Additional explosive costs.

Fragmentation and **diggability** has *rarely* been a problem at Union Reefs. This is most certainly attributed to the flaggy fissile nature of the rock mass (well developed foliation with closely spaced jointing) and the relatively high powder factors used in the past.

The shales absorb water readily along the fissile slatey cleavage and upon exposure tend to disintegrate and slake into large thin sheets. This permeable shale transmits water readily. Slope stability analysis indicates that it is essential to drain benches (usually dewatering the wall with horizontal holes). During periods of heavy rainfall small localised areas (wedges formed from intersecting planes and joint sets) are noticed to fail. If however, after a short period of time (usually once the water table has become recharged from the initial heavy rainfalls), the areas behind the failure mechanisms (faults and shear zones) are not adequately drained or dewatered, then significant water pressure can build up. The main failures in the Union Reefs region have generally been associated with a rather minor rainfall event that provides just enough additional pressure and reduction in friction co-efficients to provide a trigger for a point of release.

Perhaps the most influential aspect of blasting is the site geology. It proved difficult to achieve consistently outstanding results when geology varies not only between pits or benches, but within individual blasts. Commonly blasts were fired crossing many different structural types including shales, greywackes and greywackes intermittently dispersed throughout shales and vice versa. Couple this with the highly fractured vertical nature of the shales and conversely the generally more massive greywackes and it became clearly evident that no one individual blast design will completely suit all aspects of the geology throughout individual blasts. No single blast design was developed that would sufficiently break the greywackes without releasing (venting) multitudes of gasses to the atmosphere (blowing out) within shale bands and further diluting ore zones.

3.5 Dewatering

Each of the geotechnical investigations found that depressurisation of the pit walls was essential. Water monitoring bores were installed around the perimeter of Crosscourse pit, enabling the water levels to be tested. Generally the levels rise shortly after the rains start, then take a while to drop again during the dry season.

Woodward-Clyde completed an investigation on dewatering procedures in Crosscourse in 1995/1996. They oversaw the drilling of test boreholes, which were tested for water quality and flow rates. Some were deemed suitable for dewatering purposes, all have been monitored since.

Below the water table, it was deemed necessary, by Ian McEnHill, to drill sub-horizontal drainage holes up to 150m into the toe of each bench at a 30m spacing. The holes were successful, with some being too long and acting as more of a drainage channel from major aquifers not part of the pit groundwater system. It was then recommended that they be reduced to 50m and 80m deep with each bench being individually designed to target major structures. The availability of drill rigs with the appropriate capabilities limited the frequency of drainage holes and eventually a purpose built rig was provided by Drill Torque to complete the task

The lack of effective surface drainage was also a potential hazard for the pit wall stability. Union North South was a good indication of the effects of water trapped at the surface. Surface runoff erodes the pit walls and adds to the water in the base of the pit. Drainage from the surface through tension cracks provides upward pressure, and increases pore pressure, adding to instability.

A trial of Bentofix matting was installed along the 1195m berm, in the NNW wall of Crosscourse. This seemed to be successful in preventing drainage into the “back” of the unstable slope, called the Glacier. Other preventative methods included removal of the clayey overburden from the edge of the pit, digging drains around the edge of the pit (as in Union South) and contouring the surface away from the pit edges. Union South, Middle Pit, was a good example of the implications that can arise after leaving the clayey material on the surface around the perimeter of the pit. The water was able to absorb into the ground and drain through a fault in the eastern wall. The combination of weight from above and lubrication and upward pressure from below created a slip on the East side.

3.6 Slope Monitoring

A number of methods were employed to monitor the stability of pit walls, including visual inspections, ground water monitoring and slide monitors. Each geotechnical activity has a Standard Operating Procedure, along the guidelines set by PSM *(Standard Operating Procedures, Failure Assessment)* and the Union Reefs Pit Slope Management Standard (Appendix K). Geotechnical SOPs are listed in Appendix L.

Every active mining area was visually inspected on a regular basis, daily if required. Any cracks noted in the berms were sprayed and recorded in the Slope Inspection Register, which was then entered into the "Slopes" database (Appendix M). Spraying cracks enables the next person in the area to see if the cracks have propagated further and allows easy identification of the problematic area. Appendix N contains all the pictures, such as figure 3.6.1 below, and the "Hotspot" maps/posters that were created following visual inspections.

Figure 3.6.1. Cracks painted along the eastern wall of Millars.

Regular mapping traverses were undertaken along pit walls where potential hazards were known to exist. Structural data was collected using standard Pells Sullivan and Meynink Pty. Ltd. (PSM) mapping codes and sheets. The data was then transferred onto 1:500 map sheets and incorporated into a pit-scale structural model. Lower angle shears and joint sets (<45°) were highlighted as potential slipping planes for planar-toppling failures.

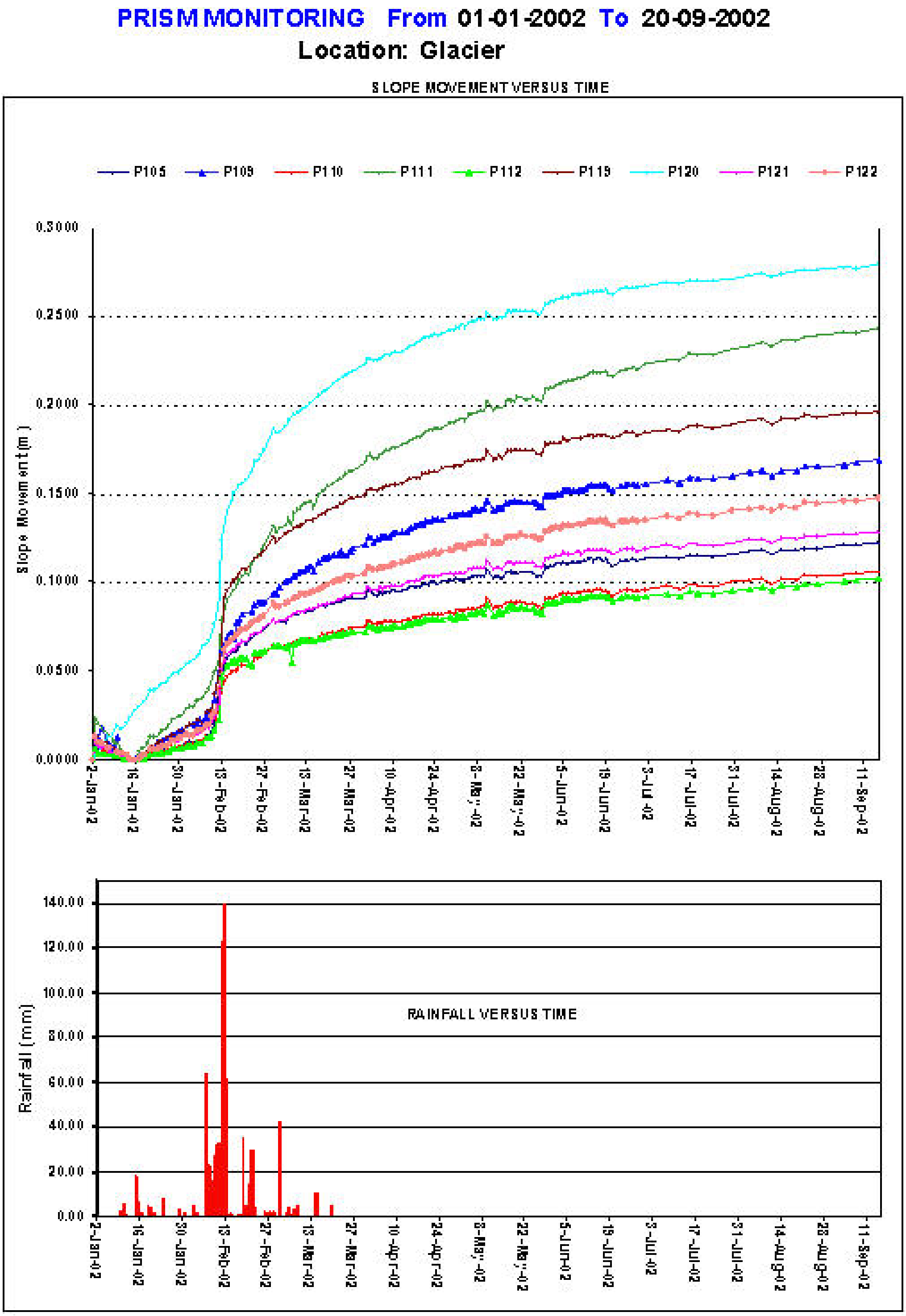
Major features such as faults and shears were occasionally transferred onto bench-plans, along with simplified lithological information for use in blast design planning through geotechnically sensitive areas, e.g. Western Cutback and the south wall of Crosscourse Pit.

Mapping data was also used for generating 3D fault projections in Vulcan and for the placement of prisms along berms.

Prisms were only installed where there was going to be a long exposure to the area i.e. not in the small, satellite pits. The survey prism pick up is entered into the “Prisms” database, Appendix O, which produces a graph of prism movement compared to rainfall data. On the next page is a pair of graphs produced from this database (figure 3.6.2), showing the correlation of prism movements to rainfall in the Glacier area. Prism locations in Crosscourse can be seen in Appendix F.

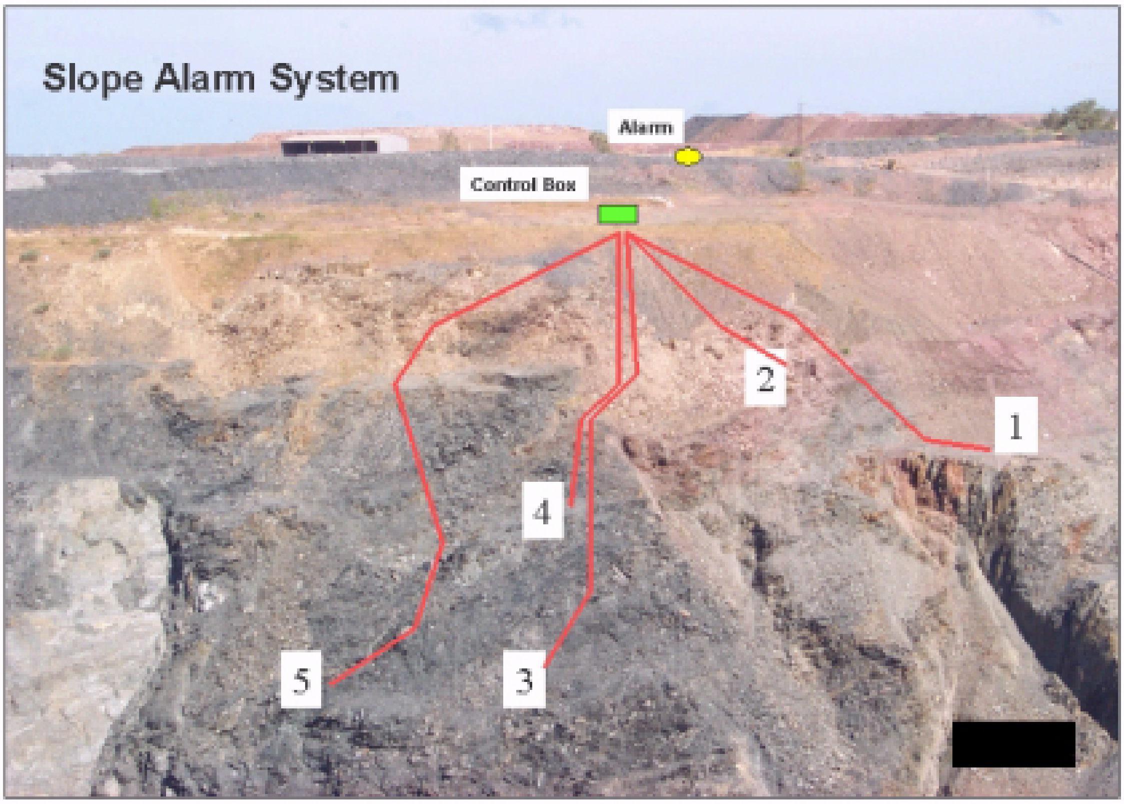
Slide monitors were installed across tension cracks to monitor their development from day to day. On a graph, it can be see when a crack is likely to become a failure, as the movement is shown as an exponential curve. There have been a number of different models of crack monitors, ranging from bars, to rulers, to 2 pieces of conduit with a tape measure glued to the inner pipe. These are seen in Appendix A. It was found that the conduit was more effective, provided the readings were taken at a constant time of day. A monitor was found to record up to 2mm of movement (both inwards and outwards) purely from changing heat temperatures.

Figure 3.6.2. Prism movements on the Glacier.



Below is a picture of the slope alarm system as installed over the glacier on the NW corner of Crosscourse (figure 3.6.3 from Appendix A). A series of lines (200lb braided fishing line) were extended down over the face of the Glacier and attached to a solar powered, extensometer alarm system. Any significant movements set off the alarm system and caused an evacuation of all personnel from the pit, until the area was inspected and cleared by a geotechnically competent person.

Figure 3.6.3. The slope alarm system, showing the positions of the individual lines.



As part of a research and development project carried out by ISSP, 3 geophone monitors were also installed around the Western Wall of Crosscourse. The Western Wall was chosen as that was thought most probable to fail. This failure was supposed to provide definite correlation points with seismic recordings that could be applied to develop failure models for other sites incorporating similar technological systems (Sunrise Dam for example). Unfortunately the West wall never failed and the project has since been terminated. Each solar powered monitor was connected to a borehole containing multiple geophones. Whenever seismic events occurred they were noted and timed using GPS satellites. Appendix P contains the project proposal and reports from ISSP.

3.7 Slope Failures

Throughout the life of the mine, there have been some large failures and some small, almost expected failures. It is almost tradition that tension cracks appear on the Western Wall and eventually topple. Union North South actually made it to the stage of toppling, following some impressive cracking. Big Tree and Millars each experienced almost identical slips on the east, which received almost identical remedial action. These were mainly due to the presence of the clayey material at the surface, also seen in Union South, Middle Pit.

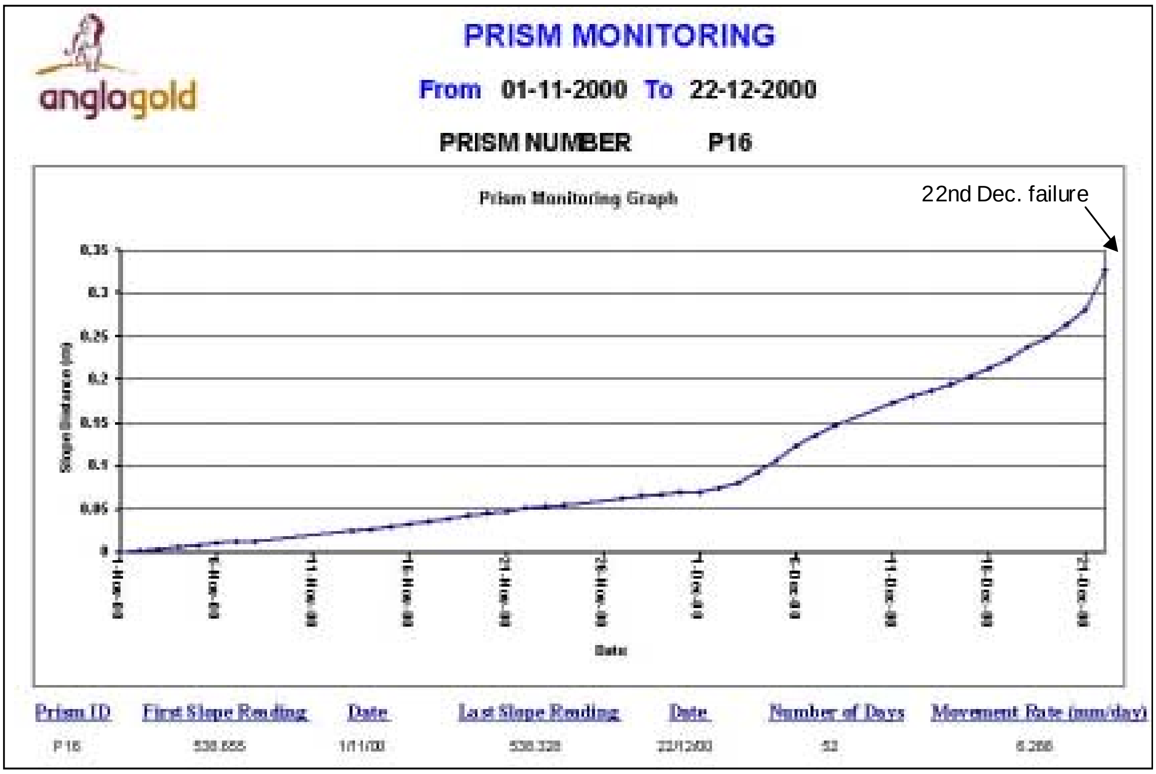
Conjugate fault intersections with pit walls commonly result in larger mass failures at this site. These failures are complex failure mechanisms involving a combination of toppling, rotation and planar-sliding. The failure of the northwest wall of Crosscourse Pit in December 2000 was a good example of this type of failure (Appendix Q also contains the sequence of events leading up to a major failure in Union North).

Toppling of weathered and transitional material is often initiated along acute fault-plane intersections with pit-walls. Fault/pit wall intersections forming angles less than 20 degrees are extremely unstable as a result of the very low shear strength values of the host rock. Fresh rock failures of this type are also observable along the Western Wall of Crosscourse Pit. The leading edge of the intersection progressively fails from the berm crest and initiates further failures along strike of the structure or bedding.

Infiltration of surface water into and along joints and bedding planes are common problems during the wet season. Water infiltration induces accelerated migration of the rock-mass. These types of failures often occur in zones in which rotation and toppling failure has previously occurred.

The most notable failure was in the NW corner of Crosscourse, the Glacier (Appendix Q). The area of influence covered the benches between and 1195 and 1017.5m RL. On the 22nd of December, 2000, about 170,000t of material between 1180 and 1100m RL failed after 2 days of >20mm rainfall, causing operations to come to a halt. This area has been under constant surveilance ever since. Below is a diagram showing the monitor movements in the lead up to the failure of the Glacier (figure 3.7.1).

Figure 3.7.1. Prism movements in the days prior to the failure of the Glacier.



Big Tree Pit experienced some instability along the eastern wall throughout its' life. Towards the end of mining, there were tension cracks developing along the western berms. Mining was stopped prior to completion, due to the instability of the walls. The main influencing factor was the saprolitic material of which the walls were comprised.

Crosscourse South experienced a major wall failure after brief, heavy rainfall. The ramp along the east wall was lost due to a large toppling failure associated with a major shear. The shear ran along the length of the ramp on the east wall. There was also a large wedge failure in the west. The final mining bench was abandoned.

Union North and Union North Southern Extension were also abandoned due to wall instability. The West wall of both pits toppled progressively, extending both into and along the wall. The failures were associated with the flat East dipping Union shear. Union North pit was abandoned before the final bench was mined.

Millars pit developed tension cracks along the entire length of the East wall and along the Western wall at the south as mining progressed. After mining was completed slumping and toppling of the walls occurred. This was due to the saprolitic material similar to Big Tree pit. A small amount of rainfall can affect clayey soil immensely, particularly when the basal support is being removed.

Ping Que South was close to failure at the time of final excavation. Both East and West walls were pulled back several times but continued to topple. A fault running almost parallel to the wall in the southwest also caused a problem, as well as the rock being weak as it was still in the oxide zone.

Dam A remained quite stable, with only a small fault at the west ramp, where the wall peeled away from a fault.

Lady Alice remained stable despite some well-developed and widespread tension cracking in the west wall. It experienced some slumping in the southwest and along the east wall, as well as tension cracks due to toppling in the west.

Prospecting Claim experienced a fairly large slip in the southern end, along the west wall, below the ramp. It appeared to have potential to continue to peel back towards the fault, but stabilized once mining of the southern pit was completed. There was also some well-developed tension cracks along the western wall, some of which required remedial work.

3.8 Geotechnical Appendices

These can be located under the directory of Reports \ Appendices \ Geotechnical Appendices on the CD that has been burnt and can be made available through head office.

Contents

A PowerPoint Presentation of Geotechnical activities in Crosscourse.

B Prospect Open Pit - Slope Design Geotechnical Report.

C Ian McEnhill site visit reports and Geotechnical updates.

D Lady Alice Open Pit - Geotechnical Report.

E West Wall Clean-up Option, Overall Slope Stability Analysis.

F Geomechanics classification system for rating of rock masses in mine design.

G Union North - Geotechnical Report

H Crosscourse Pit - Geotechnical Risk Assessment - Benches Below 1010mRL.

I Crosscourse Prisms and Glacier videos.

J Ground Support Data - Crosscourse.

K Geotechnical Pit Slope Management Standard.

L Standard Operating Procedures

M Slope Inspection Register and Slopes Database.

N Photo and Document folders for each pit.

O Prisms Database.

P Event sequence, Major Failure - Crosscourse NW and Union North, west wall.

4.0 Drill and Blast

4.1 Fundamental Blasting Concepts

This chapter assumes a basic understanding of some fundamental blasting concepts, rules of thumb, and their limited applicability. Aspects such as Burden and spacing (and their acceptable relationship ratio), Bench height, Effective Subdrill, Hole diameter, Batter rows, Explosive charging (in hole densities and concentrations), Stem Height, Powder factor, Detonation sequence, Overbreak control, fragmentation, Heave Energy, Velocity of Detonation, Physical Desensitisation, sympathetic detonation, Dynamic water and flyrock are included. A brief description of their respective **site** influences on blasting **may** be discussed.

As a rule of thumb, rules of thumb **DO NOT WORK** on this site. The dominant geological structure found at Union Reefs Gold Mine (refer chapters two and three) ensure great technical difficulty in producing consistently outstanding blast performance. This ground is VERY unforgiving of any minor errors in either design or application. Consequently, design parameters often require “tweaking” and application requires constant attention and supervision to ensure desired implementation of design is achieved.

4.2 Base Case Blast design

An example of a blast design for the lower levels of the main Crosscourse pit can be found under the Drill and Blast appendix burnt to CD. Many characteristics of this blast design may be assumed as the “Base Case” blast design that continued to be used even in the satellite pits up to two years later. Typically, bench heights were 10m, with a 3m Berm (in this case mid bench – which was not the standard). Production holes of 165mm diameter were then drilled on a pattern of 4.5m Burden x 5m Spacing. One metre of Sub-drill was applied to each hole. Due to the highly unstable nature of the western cutback, five batter rows (3x115mm, 1x140mm, 1x165mm) were employed to ensure sufficient stand-off distances of larger production holes from the toe of the batter (normally only use two batter rows – refer to section 4.2.5). Approximately 5.5m (or 135kg) of predominantly emulsion based PG2560 explosive was then charged into each production hole, with the remaining hole length of 5.5m stemmed to the surface. Timing varied upon individual blast requirements, however were generally 17ms/100ms (flat V) or 17ms/65ms (echelon) for production blasts and centre fired shots respectively (where the 17ms represents inter hole delays, and 65ms or 100ms represent inter row delays). Separate North, South, East, and West batter designs were required wherever the batter angles changed according to the pit design. Ramp blast design varied according to ramp widths and degree of weathering. In areas of highly weathered regolithic material, no subdrill was required for each hole on the ramp to achieve grade, whereas in fresh rock material up to 1.5m of subdrill would be required to achieve ramp grade.

4.2.1 Powder factor

Typical powder factors used at Union Reefs for different rock types can be found in table 4.2.1 below.

Table 4.2.1: Rock types and Powder Factors

Rock Type encountered (Weathering)	**Powder Factor (Kg / m^3)**	**Typical depth beneath surface (Metres)**
Highly Weathered Oxide	0.20 – 0.25	2 – 10m
Weathered / Oxide	0.40 – 0.45	10 – 30m
Transitional	0.50 – 0.55	30 – 50m
Fresh (CCS, UN)	0.65 – 0.70	50 – 200m
Fresh (Lower CC)	0.60 – 0.65	200 – 250m

In most places there existed approximately 2m of regolithic topsoil that may be removed without blasting. The 10m immediately below the “topsoil” may be “shaken up” with very low powder factors that still produce acceptable dig rates (i.e. 0.20kg/m^3). The next 20 – 30m of oxide material encountered below this were generally found to be quite difficult to excavate unless blasted to a minimum of

0.40kg/m^3. It is imperative when firing with such high powder factors in **weathered** material, that very large stand-off distances are used (at least 5-6m from the toe of the batter) as blast damage easily propagates back through to the final batters.

Depending on location, a transitional bandwidth may vary anywhere between 10m and 40m that still requires the use of quite high powder factors. By this stage, standoff distances are typically down to 3m from the toe of the batter.

Rarely have any excavations at Union Reefs Gold Mine demonstrated weathered material existing below a depth of 50m from the natural surface. Fresh material has been encountered as early as 25m below natural surface (such as in Dam A). By this stage, powder factors generally need to be at least 0.65 kg/m^3 to obtain acceptable dig rates. It is worth mentioning that at approximately the 1025mRL (200m depth) of the CC pit, the rock mass of material encountered appeared more “flaggy” and fissile in nature with well developed foliation and closely spaced jointing. This material no longer required such high powder factors as that ground immediately preceding it (i.e. powder factors were typically reduced to around 0.60 kg/m^3).

During the last two years of blasting at Union Reefs, the stem heights were radically increased in an attempt to reduce violent and excessive venting of blast energies to the surface. This essentially wastes blast energy that is not doing the work below where required. By increasing stem heights, typical powder factors dropped from 0.80kg/m^3 to 0.65kg/m^3.

By reducing the PF, the amount of throw (or movement) generated in each blast was also reduced which minimised backbreak and associated external waste dilution. Despite the top 3m of each 10m bench being quite blocky, the diggability and fragmentation of the remainder of the bench was not significantly decreased. Backbreak and associated external waste dilution may be reduced through *repeated* experimentation of stand-off distances for a rock mass of certain blockiness and by **timely** recognition of the controlling effects of through going fault and shear zones.

Powder factors were also adjusted according to the blockiness of the rock mass as dictated by the number of joint sets and joint spacings within it. This needs to be trialled in each new pit to obtain the minimum fragmentation and looseness acceptable from the lowest possible powder factor. Geotechnical conditions (rock quality) dictates the seismic velocity in a blast and therefore influences the amount of throw.

4.2.2 Mine Development

There are many factors related to the effectiveness of blasting, the essential of which are listed below (Gregory, 1993, p175),

1. Tenacity or cohesive strength of the rock (i.e. the rocks resistance to blasting).
2. Structure of the rock (massive, stratified, jointed or fissured).
3. Type, strength and nature of explosive used.
4. Proportion of burden distance to length of hole.
5. Detonation sequence (design and practicality).
6. Size, type and depth of shot.
7. Depth to which hole is charged (and stem height).
8. Loading density and degree of confinement.
9. Subsequent method of spoil removal (Excavator/Shovel/Front End Loader).
10. Degree of fragmentation required (also related to above).
11. Proximity or otherwise to important installations (services).
12. Ultimate shape of excavation.

4.2.3 Hole Diameter

Blast hole diameters are of paramount importance when considering a blast design. Smaller diameter blast holes better distribute explosives throughout the bench improving fragmentation and reducing toe. However, when mining low grade deposits, economies of scale generally necessitate increased bench heights to reduce the number of bench cycles. This subsequently requires an increased powder factor to sufficiently fragment the rock. The cost of increasing the powder factor is generally negated by reduced costs associated with labour and explosive accessories as there are less holes to prime, load, stem and tie in. Costs may be further diminished by incorporating larger diameter blast holes that yield higher BCM's per metre drilled and reduce redrill percentages. Larger diameter holes are also less susceptible to dynamic desensitisation (due to increased spacing) however, increased hole diameters can result in greater ground vibrations and perimeter damage as more explosive is detonating per delay.

Hole Diameters were progressively increased throughout the life of this mine to primarily withstand problems associated with collaring holes in wet shaly conditions but also due to the inefficient dynamics of using smaller diameter blast holes in increased bench heights.

Initially 89mm and 102mm blast hole diameters were drilled (for batter and production holes respectively) with Tamrock 500, Tamrock 1000 and Montabert drill rigs. These were later increased to 102mm and 115mm (respectively), predominantly using Gardner Denver 5000's and T51 drill rods. Trials were carried out using EL58 drill rods, 140mm drill bits and larger air compressors with MacMahons (refer to 140mm blasthole report by G.Davies). Upon contract re-negotiation, ROCHE appointed HUGHES drilling as their sub-contractor and again used Gardner Denver 5000's with EL59 drill rods that enabled 115mm, 127mm and 140mm diameter holes to be drilled. The GD's were struggling with the high volumes of water encountered at depth greater than 6m (particularly during the wet season) as holes would collapse during the rod change. Consequently, at ANGLOGOLD request, a larger Ingersoll Rand DM45 drill rig was commissioned that enabled single pass production drilling of 165mm diameter blast holes on 10 metre benches. The GD5000 was retained for support roles of drilling batter rows, infill holes, ramp and goodbye shots (predominantly with 115mm and 127mm diameter).

Primary reasons for increasing blasthole diameters where to minimise excessive ore dilution associated with a large number of small blasts on any given bench. To reduce the number of blasts per bench required, longer sleep times were trialled, which unfortunately increased the number of redrills (as holes had longer exposure to large amounts of ground water and thus collapsed). Larger diameter drill holes not only reduced redrills but also improved blast yields (BCM blasted per metre drilled) and allowed the increase in bench heights that further reduced the number of bench cycles per pit.

4.2.4 Burden / Spacing.

If the burden or spacing of a particular row is too high, then it may damage adjacent charges by excessive ground vibrations and result in decreased advance due to over confinement.

Theoretical rules of thumb assume the best Spacing to Burden ratio (S:B) is 1.16 to 1.0. This **theoretically** produces optimal fragmentation and yield as blast holes are situated on an equilateral triangle arrangement that exposes the largest amount of rock to blast energies provided by the least number of holes drilled. However, through numerous trials, a staggered pattern of equal burden and spacing proved optimum (i.e. S:B of 1:1). Trialling an S:B of less than one may prove further ideal (i.e. 0.9:1.0 or 4.5m Spacing and 5m Burden with 165mm diameter blast holes drilled to a depth of 10m as opposed to the “Base Case” scenario of 4.5m **Burden** by 5.0m **Spacing**).

4.2.5 Perimeter Holes / Batter Rows

Better perimeter results may be achieved by the use of lower charge concentrations. Trimex (a water resistant detonator sensitive emulsion) may also be used for perimeter holes as it provides a relatively low energy explosive with high VOD. Lower energy explosives establish a clear perimeter crack pattern without creating unacceptable levels of overbreak. However, generally their higher associated costs reduce their economic viability. Ultimately for effective perimeter control the correct combination of blasthole spacings, charge concentrations and initiation sequence must be established for each pit environment through careful design, trialling and monitoring.

Batter row standoff distances increased proportionally with the degree of weathering from 1 – 5m OFF the toe. In areas of extreme weathering (regolithic), or low structural integrity, batter rows have been successfully “left off” entirely. The **number** of batter rows depended upon the pit design batter angles, rock competency and any geotechnical/structural concerns (for example, evidence of any through going faults etc). However were generally as per the tables below.

Table 4.2.5(a) Number of batter rows fired for different batter angles

Batter Angle (degrees)	Number of batter rows
45 –54	Three
55 – 69	Two
70 – 75	One

Table 4.2.5(b) Batter row Stand-off distances for varying rock types

Degree of Weathering	Stand off Distances (m)
Regolithic Oxide	No batter rows
Oxide	3m – 5m
Transitional	2m – 3m
Fresh	1m – 2m

A typical batter row configuration may be seen under the drill and blast Appendix (burnt to CD) as part of the “Base case” blast design.

4.2.6 Detonation Sequence

During the life of this mine, detonation sequence and timing changed dramatically. Blasts were initially fired with a 42ms inter hole delay and 65ms inter row. To improve fragmentation and diggability this was reduced to a 25ms/65ms combination. To further reduce dilution a flatter “V” was introduced and a 25ms/100ms combination was trialled. Upon noticing significant improvement in ore monitor movement and to further reduce ground vibrations (for geotechnical concerns) this was further flattened to a 17ms inter hole delay and 100ms inter row. This timing sequence proved unequivocally supreme as it sped up the shattering effect between holes on one row (across structural lines of discontinuity – of major importance) whilst allowing more time for forward movement, further increasing throw along the strike of the ore (much more desirable than across ore strike). This timing configuration was then adopted as the “standard” and only varied where ramp shots or drop cuts were centre fired independently from the remaining bench .

4.2.7 Fragmentation

Fragmentation may be improved without increasing powder factors or decreasing Burden / Spacing or bench heights by simply decreasing the inter hole delays. This slight increase in speed increases the breakage and smashing interaction between holes on the same row without choking the shot up as it still has plenty of time to move forward. This has the added benefit of essentially reducing the angle of sideways movement by flattening the “V” and forcing more forward movement.

4.2.8 Heave Energy

Heave energy is often controlled by the quality of intact rock (i.e. the amount of cracks available for gas expansion). High strength rock may form few cracks for gas advance and thus limit the amount of heave energy produced. If the heave energy is reduced, fragmentation may be decreased and overall blast performance hampered due to insufficient rock breakage. Insufficient stemming (and poor stemming material) can also produce the same result as this allows explosive gasses to vent more easily to the surface without doing the work where it is most required (i.e. at the toe of the bench). However heave should be minimised in an attempt to reduce ore dilution.

A significant reduction in heave was noticed with the employment of improved blasting techniques (typically from as high as 5m of heave, down to a consistent, manageable 2m). By increasing stemming columns and using angular stemming material of around 20mm size, heave energy was adequately contained without significantly reducing fragmentation and diggability.

4.2.9 Velocity of Detonation

The velocity of detonation (VOD) is the rate at which the detonation wave travels from the point of initiation along the explosive. VOD varies with explosive types, and **increases** with charge diameter, explosive strength, and degree of confinement. A higher VOD creates greater shock energy and less heave energy. Generally the VOD is kept high to avoid misfires created when compression waves outrun the detonation front (stalling the initiation).

Explosives with a faster VOD (Emulsion based) are generally more suited to **Fresh** material than those with a slower VOD (Ammonium Nitrate based – i.e. ANFO), yet conversely the reverse is true for weathered material. To sufficiently fragment the **fresh** material in order to excavate efficiently, higher shock energy was required to produce a “shattering effect”. In **weathered** material however, explosives with slower VOD’s yielded better results as high percentages of the shock energy from emulsion based products were “absorbed” into the weathered material therefore decreasing its effectiveness. Higher levels of heave energy associated with ANFO, then proved more effective in separating the weathered material for purposes of excavation.

4.2.10 Fly rock / Venting of explosive gasses / Stemming

Flyrock and venting of explosive gasses generally result from an insufficient stemming column for the mass of explosives being used in that particular hole. Flyrock can also result from poorly timed delay sequences causing the hole to become overconfined and crater upwards instead of breaking towards the free face generated by the row of holes directly preceding it. Venting of explosive gasses is undesirable as this energy escapes to the atmosphere without doing any work actually fragmenting the rock.

The predominant vertical nature of thinly layered interbedded shales and greywackes encountered at Union Reefs Gold Mine implied that stemming columns required to sufficiently contain blasts lied outside general rules of thumb (at 33 drill bit diameters). Consequently, for a hole diameter of 165mm, stem heights of 5m and 5.5m in bench heights of 7.5m and 10m respectively were commonly used with much success.

Initially drill cuttings were being used to stem blast holes. This proved marginally effective for small diameter blast holes of 5m bench height, but once larger diameter blast holes were being used a more angular stemming material with better “locking in” attributes was required. Stemming material was screened on site at approximately 20mm (+14mm, – 30mm) using fresh waste material from Crosscourse and Crosscourse South pits. For the last 18 months, railway ballast was trucked from the quarry near Tennant Creek that was producing for the Alice to Darwin railway construction.

4.2.11 Physical Desensitisation

In closely spaced blastholes, or charges that are fired on the same delay, explosives may become desensitised by blast induced pressures from earlier fired charges. Physical desensitisation may occur as subsequent delayed blastholes are reached by either a compression wave, or gases (passing through joints from earlier fired charges) that compress or destroy the air and gas bubbles within and surrounding the charge. Physical desensitisation is not generally a problem with open pit blasting except where infill holes or batter rows are collared too closely to larger production holes.

Explosive column (drill hole) displacement may also be a problem whereby holes are spaced too closely together. The explosive column may become partially or completely separated (disjointed) from the detonator/booster system at the base of the hole when ground is displaced from earlier fired charges before the explosive has time to propagate itself fully up the column (or even before detonation of that hole has begun).

Several misfires encountered at Union Reefs may have been explained by a combination of these last two phenomenon. Sometimes infill holes are placed too closely along the boundary line between batter rows and production holes where it was deemed too great a burden distance for the batter rows to “pull” alone. Adequate communication with the blast crew generally negated this occurrence.

4.3 Drill rig Performances and Utilisations

The drill and blast database only dates back to the 1st of October 1997. At that time MacMahons brought in another drilling sub-contractor (PEARSON), to supplement the drilling capacity provided by AUSDRILL (as they were struggling to meet production requirements alone). Consequently, it is difficult to produce reports demonstrating life of mine aspects such as number of holes, total metres drilled, total BCM, yields per metre drilled, utilisations and availability. Tables 4.3.2(a-c) (at the end of this chapter) demonstrate aspects such as penetration rates, productivity and contribution of individual drill rigs for various degrees of weathering. It is important to note that this information is **representative only** from 1st of October **1997** onwards.

4.3.1 Statistics Summary

Since 1st Of October 1997, approximately 2,412,399 metres were drilled in 299,422 blast holes yielding 27,325,220 BCM using 17,062,654 kg of explosive for an average powder factor of 0.62kg/m^3. In addition, there were a further 312,470 metres drilled in 43,810 holes that were either not loaded or redrilled/blown out due to hole collapse (so were essentially not counted in the afore mentioned blasting).

Of the 299,422 holes blasted, 48,293 (15%) were of 5m Bench height, 203,340 (63%) were of 7.5m of Bench height, and 47,789 (12%) were of 10m bench height.

4.3.2 Drill Penetration Rates, Productivities and Contributions

On the following pages there are three tables representing drill rig performance over **the life of the database** for every single drill rig that operated at the URGM site. The first, table 4.3.2 (a) displays the number of drilled holes and metreage, working, shift and down time hours recorded. From this, penetration rates, utilisations, and contributions were calculated. The second, table 4.3.2 (b), attempts to average the penetration rates experienced by **types** of drill rigs for different hole diameters over **all** types of weathering. Table 4.3.2 (c) displays individual drill rig working hours, total metreage drilled, and penetration rates for the various degrees of weathering. These tables may be used when trying to determine a suitable drill fleet for another operation if it were to proceed.

Table 4.3.2 (a) Drill rig data

BLAST Drill Performance from 1st of October 1997 to 30th July 2003								
Rig	**Number of drilled**		**Time (Hrs)**			**Pen. Rate**	**Utilisation**	**Contribution**
No.	**Holes**	**Metres (m)**	**Shift**	**Downtime**	**Working**	**(m/Hr)**	**(%)**	**(%)**
Cubex	228	2,392.6	306.7	162.2	144.5	16.6	47.1%	0.09%
DM45	14,442	148,671.4	11,626.9	7,130.4	4,496.5	33.1	38.7%	5.46%
GD 1	31,167	227,890.7	21,175.3	13,629.5	7,545.8	30.2	35.6%	8.36%
GD 2	26,765	203,226.0	17,892.2	11,368.2	6,524.0	31.2	36.5%	7.46%
GD 3	12,190	81,123.9	6,034.3	2,717.7	3,316.6	24.5	55.0%	2.98%
GD 3100	11,216	90,591.3	7,276.4	3,490.7	3,785.7	23.9	52.0%	3.32%
GD 3101	49,212	403,546.4	17,722.2	6,761.0	10,961.2	36.8	61.9%	14.81%
GD 3106	21,134	163,647.5	9,864.5	4,252.7	5,611.8	29.2	56.9%	6.01%
GD 3107	42,467	332,514.4	18,096.6	7,335.7	10,760.9	30.9	59.5%	12.20%
GD 3182	19,175	150,154.3	10,307.7	4,805.2	5,502.5	27.3	53.4%	5.51%
GD 3186	45,264	371,886.9	17,073.5	6,579.6	10,493.9	35.4	61.5%	13.65%
Mont 3189	12,591	102,397.5	4,359.1	1,726.7	2,632.4	38.9	60.4%	3.76%
Mont 3190	33,809	263,827.7	11,295.7	4,265.0	7,030.7	37.5	62.2%	9.68%
Mont 3194	10,770	82,253.4	3,720.9	1,396.5	2,324.4	35.4	62.5%	3.02%
M3 (Pearson)	5,803	44,565.4	2,938.7	1,285.0	1,653.7	26.9	56.3%	1.64%
Pearson (Macmahon)	6,637	52,688.5	2,613.3	1,127.4	1,485.9	35.5	56.9%	1.93%
SK 45	362	3,491.5	399.6	124.7	274.9	12.7	68.8%	0.13%
Total	**343,232**	**2,724,869.4**	**162,703.6**	**78,158.2**	**84,545.4**	**32.2**	**52.0%**	**1.00**

Table 4.3.2 (b) Drill rig Average

Penetration rates per hole diameter

Average Penetration Rates per hole*		
Primary Rig Types	**Hole Diam (mm)**	**Pen Rate (m/Hr)**
GD3000 / Montaberts	89	34.74
GD3000 / Montaberts	102	34.01
GD3000 / GD5000	115	29.14
GD5000	127	28.66
GD5000	140	24.63
DM 45	165	30.48

* For all types of weathering combined

Table 4.3.2 (c) Penetration rates for Oxide \ Transitional \ Fresh

Individual Rig Penetration Rates by Weathering between 1/10/97 and 10/08/03

		Rig	Hours	Metres	Pen Rate
			Hr	M	m/Hr
Weathering	Fresh	Cubex 4	66.50	707.80	10.64
		SK45	274.90	3,491.50	12.70
		GD 3	2,642.12	57,848.40	21.89
		GD 2	3,736.04	82,767.45	22.15
		GD 1	4,662.34	105,504.15	22.63
		GD3100	3,624.70	83,988.20	23.17
		DM45	2,555.50	59,327.70	23.22
		GD3182	5,332.80	143,120.30	26.84
		Pearson (M3)	1,653.70	44,565.40	26.95
		Mont3190	3,882.80	109,058.90	28.09
		GD3107	9,364.40	264,443.90	28.24
		GD3106	5,407.60	154,756.43	28.62
		Mont3194	1,664.30	49,585.90	29.79
		Mont3189	1,789.03	53,854.60	30.10
		GD3186	8,218.80	248,113.60	30.19
		GD3101	8,849.23	282,646.30	31.94
		Pearson (Macmahon)	1,435.90	50,145.70	34.92
		Totals	**65,160.66**	**1,793,926.23**	
	Presplit	GD 1	40.75	769.70	18.89
		GD 3	57.00	1,130.20	19.83
		GD 2	80.95	2,172.90	26.84
		Totals	**178.70**	**4,072.80**	
	Weathered	Cubex 4	78.01	1,684.80	21.60
		GD 3	617.45	22,145.30	35.87
		GD3100	161.00	6,603.10	41.01
		GD3182	169.70	7,034.00	41.45
		GD 1	2,842.73	121,616.86	42.78
		GD3106	204.20	8,891.10	43.54
		GD 2	2,706.94	118,285.60	43.70
		DM45	1,941.01	89,343.70	46.03
		GD3107	1,396.50	68,070.50	48.74
		Mont3190	3,147.90	154,768.79	49.17
		Mont3194	660.10	32,667.50	49.49
		Pearson (Macmahon)	50.00	2,542.80	50.86
		GD3186	2,275.10	123,773.30	54.40
		GD3101	2,112.00	120,900.10	57.24
		Mont3189	843.40	48,542.90	57.56
		Totals	**19,206.04**	**926,870.35**	
	All Weathering (combined Ox. \ Trans.\ Fresh)	SK45	274.90	3,491.50	12.70
		Cubex 4	144.51	2,392.60	16.56
		GD3100	3,785.70	90,591.30	23.93
		GD 3	3,316.57	81,123.90	24.46
		Pearson (M3)	1,653.70	44,565.40	26.95
		GD3182	5,502.50	150,154.30	27.29
		GD3106	5,611.80	163,647.53	29.16
		GD 1	7,545.82	227,890.71	30.20
		GD3107	10,760.90	332,514.40	30.90
		GD 2	6,523.93	203,225.95	31.15
		DM45	4,496.51	148,671.40	33.06
		Mont3194	2,324.40	82,253.40	35.39
		GD3186	10,493.90	371,886.90	35.44
		Pearson (Macmahon)	1,485.90	52,688.50	35.46
		GD3101	10,961.23	403,546.40	36.82
		Mont3190	7,030.70	263,827.69	37.53
		Mont3189	2,632.43	102,397.50	38.90
		Totals	**84,545.40**	**2,724,869.38**	

4.4 Specialised Blasting Techniques

At Union Reefs there have been some specifically designed specialised blasting techniques employed with varying degrees of success. These include ramps, sumps, zipper lines and pre split.

4.4.1 Blasting Across Significant Geological Boundaries

When blasting across various geological structures (ore to waste, greywackes to shales or hard to soft) there is a tendency to alter the blast pattern. Preferentially this is to be avoided at all times unless able to fire different rock types in different blasts (also not recommended if at all possible as bigger shots fired less often are better). A far better result will be obtained by maintaining a singular blast pattern across varying boundaries tweaking aspects such as **subdrill** or **stem height** within them. For example, in softer shale areas subdrill may be reduced and still achieve an adequate floor level with minimal toe. Stem height may also be increased (again decreasing column charge) as less explosive energy is required to sufficiently break through to the surface (preventing blowouts).

4.4.2 Ramp Design

A site wide generic ramp design was generated that only required "tweaking" adjustments to be made in correlation with variations of rock type (weathered, transitional or fresh material). To compensate the rock type becoming harder, subdrill on the various ramp hole depths were increased accordingly to ensure ramp grade was still achieved. An example of each can be found under the Drill and Blast Appendix burnt to CD. Ideally, each drill hole down the ramp would have its depth delineated by survey (usually time constraints of the survey department prevent this). Ramp segments should be generated at no more than 1m depth intervals (preferably 0.5m depth intervals), whereby every hole in that segment is assigned the same depth. This ensures adequate control is placed over the entire ramp design in relation to depth, and further, the achievement of ramp grade.

The idea is to simply maintain a square pattern of equal **SPACING across** the ramp and only increase firstly, the drill hole diameter, then the **BURDEN,** as ramp depth increases, **down** the ramp. This allows easy access by all concerned (drill rigs, bomb truck, and stemming loader) and maintains a relatively constant powder factor even though ramp depth increases. By placing the drill holes on a single offset along ramp length entirety, then a more definite boundary contact between the ramp and batters is blasted and further excavated. Most significantly though, it also maintains a consistently low level of confinement throughout the entire ramp shot, thus reducing the probability that any toe will exist at full depth of the ramp (i.e. ramp grade will always be achieved).

4.4.3 Dewatering Sumps

The idea of firing sump shots in certain areas is limited primarily to the permeability of the rock type and the number of contacts that can be intercepted. No advantage is gained by firing sump shots in one single area of impermeable rock. Very little water will flow to that sump and essentially remain dry, whilst a 10 metre drill hole 5 metres away (from the sump) may be full of water. Consequently, the best sump that can be generated is a **full depth production** shot of minimum three rows (but preferably 5) that extends full width across the pit bench floor. This provides many benefits as it produces a sizable sump that holds a lot of water, and extends full depth of the bench to be drilled. It also intersects or exposes every single rock type and structure that will be encountered for the remainder of the bench. This allows the majority of water that is going to be encountered on any one bench, to flow directly into the sump. If a sizable sump in the middle of the pit is undesirable, then refer to the next section.

4.4.4 Zipper lines along Echelons

Once a small sump is fired (preferably with easy access to dewatering pipe lines (networks, established infrastructure)) then firing a zipper line of full depth production holes along an echelon from the existing sump as far away as possible (or practical), also works well. It is important to analyse the geological structure of each pit, and ensure that the echelon of blast holes chosen are going in the direction that intercepts/intersects (cuts across) as many structural contacts as possible. Otherwise, firing a zipper line along echelons running parallel to existing structure, will be of no more benefit than firing a small sump in one particular impermeable structure.

4.5 Quality Control Aspects

Quality Control is of paramount importance. Extreme amounts of toe were encountered early on that suggested the contractor was carrying out inadequate quality control. It was determined that MacMahons were continually under-drilling the **EFFECTIVE** sub-drill by not adequately allowing for substantial amounts of “fallback” arising from drill cuttings and holes collapsing. There were also insufficient adjustments being made whenever toe was encountered to compensate the extra drill depth required to get back “on design”. Consequently (after several benches), extremely uneven floors would result that did not even remotely resemble the design bench heights.

As such, a rather comprehensive quality control system was initiated. Quality control procedures were adopted from ORICA personnel who frequented the site. These were elaborated upon as required and once an operating standard was set, handed back over to the contractor. A full procedure pertaining to quality control and the Quality Control database can be found under the Drill and Blast Appendix burnt to CD.

Aspects such as survey control, pattern mark up procedures, bit diameters, drill depths, charge depth, column rise, stem heights, heave, throw, dilution, powder factor, fragmentation, diggability, and load times were all investigated and reported upon a weekly basis. Acceptable tolerances for each aspect were set and expected to be achieved with no exceptions. In addition, blast review meetings were held after EVERY blast with all representatives relating to drill and blast being present (i.e. Drillers, shot firers, bomb crew, ORICA, blasting and production engineers etc). The more people present the more issues could be resolved in a timely manner.

Significant, notable improvements were achieved primarily as a result of the additional quality control systems put in place.

5.0 Mining Operations

The last load of ore from the Union Reefs Gold Mine was mined at 1120am on Sunday the 27th of July 2003. Remedial work continued with the contractor until the end of the month pertaining to rehabilitation.

5.1 Optimisations

5.1.1 Introduction

It is not the intent of this section to detail all of the optimisation work performed for or at Union Reefs Gold Mine over the past 9-10 years. This section will endeavour to provide a snapshot of the earliest of the optimisation parameters that could be found at the time of writing and the last optimisation parameters used for each pit so that a comparison can be made to the final actual as-mined pit parameters shown in section 5.2. As many of the intermediate optimisation studies performed that can be located, will be listed and referenced in this section and in the footnotes at the end of this section.

5.1.2 Original Optimisation

The original open-pit geometry was determined from a detailed geotechnical assessment by Coffey Partners International Pty Ltd in February 1993 (1). A mining study was performed in September 1993 by Minenco (2) however, this report could not be located at the time of writing this section of the closure report. The Minenco report is referenced as the URGM Feasibility Study in Technical Report UR0232. The Coffey Partners (1) parameters were used in a Lerchs-Grossmann optimisation and open-pit design study in June 1994 (3) on the Crosscourse and Union North pits. These parameters were –

- 20m maximum bench heights
- 6m wide berms
- batter angles

- weathered zone
  - 50° on the west wall
  - 55° elsewhere
- fresh zone
  - 75° on the east and south-east walls
  - 65° on the north-east wall
  - 60° on the north wall
  - 65° on the west wall

- cable bolting of 30 to 50% of the fresh material in the west wall using 10-15 metre long cable anchors on a 3m vertical by 4m horizontal pattern.
- Haul roads designed assuming 50-85t trucks and were 18m wide for two-way traffic and 10m wide for one-way traffic. Allowances for drains and safety berms were included in these widths. Gradients were 1:10.
- SG data, weathered 2.52t/bcm, transitional 2.57t/bcm and fresh 2.74t/bcm.
- No cost data is reported in this initial mine plan.
- Final pit quantities were –

Table 5.1.1 Final pit quantities Crosscourse pit

ORE	WASTE	TOTAL
BCM	BCM	BCM
2,914,485	17,001,848	19,916,333

Table 5.1.2 Final pit quantities Union North pit

ORE	WASTE	TOTAL
BCM	BCM	BCM
421,029	1,534,830	1,955,859

- Crosscourse pit final RL was 980m and Union North pit final RL was 1130m.

5.1.3 Mine Expansion Study Optimisation

Major optimisation work was done in October 1996 as part of the Union Reefs Gold Mine, Mine Expansion Study – Mine Optimisation Study [(4)]. This study looked at a number of grinding options based around increasing throughput from the design 1.25Mtpa (operating at 1.7Mtpa on oxides after plant upgrades) up to 1.4, 1.6, 1.8 or 2.0Mtpa, processing fresh ore. Some of the optimisation input parameters for the 2.0Mtpa SAG Mill option are shown below –

Table 5.1.3 Mine Expansion Study – Mine Optimisation Study Parameters for the 2.0Mtpa SAG Mill Option, Crosscourse pit Mining Costs

RL (toe)	SG	ORE		WASTE	
		L&H	D&B	L&H	D&B
mAHD(+1000)	t/BCM	$/BCM	$/BCM	$/BCM	$/BCM
1220	2.5	1.84	0.46	1.30	0.46
1160	2.6	1.89	0.81	1.60	0.81
1060	2.7	2.71	1.40	2.62	1.40
960	2.7	3.46	1.40	3.35	1.40

Table 5.1.4 Mine Expansion Study – Mine Optimisation Study Parameters for the 2.0Mtpa SAG Mill Option, Union North pit Mining Costs

RL (toe)	SG	ORE		WASTE	
		L&H	D&B	L&H	D&B
mAHD(+1000)	t/BCM	$/BCM	$/BCM	$/BCM	$/BCM
1250	2.5	1.94	0.46	1.30	0.46
1190	2.6	2.10	0.81	1.46	0.81
1150	2.7	2.84	1.40	2.15	1.40
1110	2.7	3.20	1.40	2.49	1.40

Table 5.1.5 Mine Expansion Study – Mine Optimisation Study Parameters for the 2.0Mtpa SAG Mill Option, Crosscourse & Union North pit

Material	SG	Grade Control	Miscellaneous Mining	Processing incl. Maintenance	Administration	Total	Recovery
		$/t	$/t	$/t	$/t	$/t	%
Oxide	2.5	1.31	0.065	8.14	1.11	10.625	97
Transitional	2.6	1.31	0.065	8.86	1.11	11.345	97
Fresh	2.7	1.31	0.065	9.85	1.11	12.335	94

The geotechnical parameters are the results of work performed by Pells Sullivan Meynink Pty Ltd. These geotechnical reports are referenced in the Mine Expansion Study. In Crosscourse pit two domains were identified, Domain 1 being west of 5000E and Domain 2 being east of 5000E.

Table 5.1.6 Mine Expansion Study – Mine Optimisation Study Parameters for the 2.0Mtpa SAG Mill Option, Crosscourse pit Geotechnical

Wall Direction	Bench Height	Batter Angle	Berm Width	Number of Benches	Haul Road Width	Overall Slope Angle	Domain
Azimuth	m	deg	m		m	deg	
45	23	75	6	9	40	54.9	D1
80	23	65	10	9	40	43.4	D1
90	23	65	10	9	40	43.4	D1
100	23	65	10	9	40	43.4	D1
165	23	65	6	9	40	48.0	D1
210	23	70	6	11	30	54.0	D1
350	23	75	6	9	40	54.9	D1
0	23	75	6	9	30	56.9	D2
230	23	70	6	11	40	52.5	D2
270	23	65	10	9	40	43.4	D2
295	23	65	10	9	40	43.4	D2
305	23	70	6	9	40	51.4	D2

Table 5.1.7 Mine Expansion Study – Mine Optimisation Study Parameters for the 2.0Mtpa SAG Mill Option, Union North pit Geotechnical

Wall Direction	Bench Height	Batter Angle	Berm Width	Number of Benches	Haul Road Width	Overall Slope Angle
Azimuth	m	deg	m		m	deg
45	23	75	6	4	20	55.4
90	23	65	10	4	20	44.4
165	23	65	6	4	20	48.4
230	23	70	10	4	20	47.4
270	23	65	10	4	20	44.4
295	23	65	10	4	20	44.4

The gold price used to generate the following quantities was $550/oz.

Table 5.1.8 Final pit quantities Crosscourse pit

Pit	ORE		WASTE	TOTAL	Cash Cost
	t	g/t	t	t	$/oz
Crosscourse (Optimised)	7,403,854	2.10	45,529,896	52,933,750	341
Crosscourse (Mined up to July 1996)	3,475,180	1.70	15,108,939	18,584,119	
Crosscourse (Total)	10,879,034	1.97	60,638,835	71,517,869	

Table 5.1.9 Final pit quantities Union North pit

Pit	ORE		WASTE	TOTAL	Cash Cost
	t	g/t	t	t	$/oz
Union North (Optimised)	1,373,755	1.66	6,952,161	8,325,916	385
Union North (Mined up to July 1996)	0	0.00	0	0	
Union North (Total)	1,373,755	1.66	6,952,161	8,325,916	

Crosscourse pit final RL was 932.5m and Union North pit final RL was 1087.5m.

5.1.4 Intermediate Optimisations

Intermediate optimisation reports are listed below –

(5) Bertinshaw R. Mining & Resource Technology Pty Ltd. Optimisation of Union Reefs Deposit. July 1997. Technical Report UR0242.

(6) Bertinshaw R. Mining & Resource Technology Pty Ltd. Optimisation of Union Reefs Deposit. August 1997. Technical Report UR0248.

(7) Bertinshaw R. Mining & Resource Technology Pty Ltd. Creation of Optimisation Models for Union Reefs Deposit from November 1998 Resource Model. November 1998. Technical Report UR0268.

[8] Booth P. Union Reefs Gold Mine. Optimisations of November 1998 Resource Model Pit Designs for Crosscourse and Union North. January 1999. Technical Report UR0271.

[9] Kiely E. Mining & Resource Technology Pty Ltd. Creation of Optimisation Models for Union Reefs Deposit from November 1999 Resource Model. November 1999. Technical Report UR0281.

[10] Booth P. Union Reefs Gold Mine. Optimisations of February 2000 Resource Model for Resource Reporting Generation of Pit Designs for Reserve Reporting. April 2000. Technical Report UR0288.

[11] Bertinshaw R. Mining & Resource Technology Pty Ltd. Optimisation of Union Reefs Gold Mine (MRT0885). August 2000. Technical Report UR0???.

[12] Sinclair J. Mining & Resource Technology Pty Ltd. Optimisation of Union Reefs Gold Mine Revised Mining Costs (MRT0889). August 2000. Technical Report UR0???.

[13] Sinclair J. Mining & Resource Technology Pty Ltd. Optimisation of the Esmeralda Deposit Union Reefs Gold Mine (MRT0893). September 2000. Technical Report UR0???.

[14] Golder Associates Pty Ltd. Report on Dam A Pit Optimisation Union Reefs Gold Mine (01650103-a). November 2001. Technical Report UR0???.

[15] Davies G. Union Reefs Gold Mine. URGM Mine Design and Schedule 2001-2003. June 2001. Technical Report UR0???.

[16] Davies G. Union Reefs Gold Mine. URGM Mine Design and Schedule 2002-2004. June 2002. Technical Report UR0???.

5.1.5 August 2000 Optimisations

A major geotechnical review was undertaken at URGM by SRK Consulting [17] in August 2000 in response to the failure of the west wall in Crosscourse pit. The subsequent optimisation work done in August 2000 by MRT (referenced reports 11 & 12) used slope parameters that were provided by SRK and are included in Appendix 3 of report (11). A re-run of the optimisations by MRT was also performed in August to used revised mining rates generated by MineConsult Pty Ltd [18] who were preparing a shadow tender as part of the re-tendering process that was occurring with the mining contract. A summary of the optimisation parameters from report 12 are shown below. Report 12 and the individual results files for each pit including the satellite pits have been copied onto the CD –

Table 5.1.10 Optimisation of URGM Revised Mining Costs August 2000

		Waste ($/bcm)			Ore ($/bcm)		
Bench	**SG**	**L&H**	**D&B**	**Total**	**L&H**	**D&B**	**Total**
Crosscourse pit	**Nth of 6400N**						
1182.5 – 1160	2.54	1.69	1.40	3.09	1.54	1.40	2.94
1160 – 1137.5	2.66	1.78	1.40	3.18	1.72	1.40	3.12
1137.5 – 1115	2.66	1.93	1.40	3.33	1.85	1.40	3.25
1115 – 1092.5	2.66	2.07	1.40	3.47	1.96	1.40	3.36
1092.5 – 1070	2.74	2.18	1.40	3.58	2.05	1.40	3.45
1070 – 1047.5	2.74	2.31	1.40	3.71	2.23	1.40	3.63
1047.5 – 1025	2.74	2.49	1.40	3.89	2.39	1.40	3.79
1025 – 1002.5	2.74	3.78	1.40	5.18	3.71	1.40	5.11
1002.5 – 980	2.74	4.38	1.40	5.78	4.11	1.40	5.51
980 - 957.5	2.74	4.84	1.40	6.24	4.63	1.40	6.03
957.5 – 935	2.74	5.99	1.40	7.39	5.71	1.40	7.11
Crosscourse South pit	**Nth of 6000N**						
1205 – 1182.5	2.54	1.92	1.40	3.32	2.42	1.40	3.82
1182.5 – 1160	2.66	2.35	1.40	3.75	2.88	1.40	4.28
1160 – 1137.5	2.66	3.10	1.40	4.50	2.97	1.40	4.37
1137.5 – 1115	2.66	3.24	1.40	4.64	2.86	1.40	4.26
1115 – 1092.5	2.66	3.19	1.40	4.59	2.86	1.40	4.26
Union South / Millars pits	**Sth of 6000N**						
1220 – 1197.5	2.54	1.33	1.40	2.73	2.10	1.40	3.50
1197.5 – 1175	2.54	1.60	1.40	3.00	2.50	1.40	3.90
Lady Alice Hills pit	**Nth of 7350N**						
1242.5 – 1220	2.54	1.65	1.40	3.05	2.13	1.40	3.53
1220 – 1197.5	2.54	1.84	1.40	3.24	2.07	1.40	3.47
1197.5 – 1175	2.66	2.13	1.40	3.53	2.36	1.40	3.76
Union North pit	**Nth of 7850N**						
1205 – 1182.5	2.54	1.70	1.40	3.10	2.20	1.40	3.60
1182.5 – 1160	2.54	2.03	1.40	3.43	2.47	1.40	3.87
1160 – 1137.5	2.66	2.34	1.40	3.74	3.03	1.40	4.43
1137.5 – 1115	2.66	2.82	1.40	4.22	3.21	1.40	4.61
1115 – 1092.5	2.66	3.27	1.40	4.67	3.77	1.40	5.17
Alta / Orinoco pits	**Nth of 8600N**						
1180 – 1157.5	2.54	2.18	1.40	3.58	2.64	1.40	4.04
1157.5 – 1135	2.66	2.93	1.40	4.33	3.85	1.40	5.25

Table 5.1.11 Optimisation of URGM Overall Slope Angles August 2000

Azimuth	**Easting**	**Northing**	**RL**	**Overall Angle Base**
0-180	4302.5-5502.5	4905.5-6005.5	800-1270	43.3°
225-315				38.3°
0-180	4302.5-5502.5	4905.5-7105.5	800-1140	47.5°
225-315				45.1°
0-180	4302.5-5502.5	6005.5-7105.5	1140-1270	34.1°
225-315				36.0°
0-180	4302.5-5502.5	7105.5-9705.5	800-1270	37.9°
225-315				33.9°

Table 5.1.12 Optimisation of URGM Process Operating Costs August 2000

Description	**Ore Type**		
	Weathered	Transitional	Fresh
Ore Mining Diff.	0.16	0.10	-0.07
Grade Control	0.91	0.91	0.91
Mine Admin	1.31	1.31	1.31
Processing cost	8.01	8.01	8.01
Mine support costs	3.00	3.00	3.00
Rehabilitation	-	-	-
CostP ($/t)	13.39	13.33	13.16
Recovery (%)	94.0	92.5	92.5

Table 5.1.13 Final pit quantities

Pit	Gold Price	ORE		WASTE	TOTAL	Cash Cost
	$/oz	kt	g/t	kt	kt	$/oz
Crosscourse (Optimised)	500	3,386	1.76	7,603	10,989	353
CC (Mined up to August 2000)		11,538	1.65	66,598	78,136	
Crosscourse (Total)		14,924	1.67	74,201	89,125	
Union North (Optimised)	500	1,127	1.50	3,852	4,979	430
UN (Mined up to August 2000)		822	1.29	4,465	5,288	
Union North (Total)		1,949	1.41	8,317	10,267	
Lady Alice Hills	500	642	1.91	5,644	6,286	453
Alta/Orinoco	500	183	1.71	593	776	383
Crosscourse South	500	380	1.70	1,511	1,891	419
CCS (Mined up to August 2000)		113	1.77	776	889	
Crosscourse South (Total)		493	1.72	2,287	2,780	
Union South/Millars	500	147	1.55	454	601	387

Stability issues and further geotechnical evaluation of the Crosscourse pit west wall in the first half of 2001 resulted in the wall being flattened further from those recommended by SRK to an overall wall angle of 37 degrees above the 1100RL. All pits were re-designed to ten metre bench heights to take advantage of the single pass Ingersoll-Rand DM45 Blast Hole Drill that was mobilised to site in May 2001. The August 2000 $500/oz optimal pit shells were used as the basis of the re-designs.

Optimisation work from the first half of 2002 onwards was performed in conjunction with the Geological Superintendent as the new resource models for the satellite pits were created in MineMap. All pits were designed with ten metre bench heights and with a three metre wide berm every bench. Ramp widths were 12m and grades were 1:8. Batter angles were either 55 or 60 degrees depending on the depth of the pit with overall wall angles of 45 degrees.

5.1.6 Dam A Optimisation

Dam A pit was optimised in November 2001. The optimisation report [14] is included on the CD in PDF format. Below are the optimisation parameters and results.

Table 5.1.14 Optimisation of Dam A Mining Costs November 2001

		Waste ($/bcm)			**Ore ($/bcm)**		
Bench	**SG**	**L&H**	**D&B**	**Total**	**L&H**	**D&B**	**Total**
Surf – 1157.5	2.50	2.24	1.48	3.72	3.00	1.48	4.48
1157.5 – 1150.0	2.60	2.27	1.48	3.75	3.05	1.48	4.53
1150.0 – 1135.0	2.70	2.48	1.51	3.99	3.33	1.51	4.84
1135.0 – 1112.5	2.70	3.34	1.60	4.94	3.58	1.60	5.18
1112.5 – 1100.0	2.70	3.58	1.60	5.18	3.83	1.60	5.43

Table 5.1.15 Optimisation of Dam A Overall Slope Angles November 2001

Azimuth	**Overall Angle Base**
0	37°
90	37°
180	37°
225	34°
315	34°

Table 5.1.16 Optimisation of Dam A Process Operating Costs November 2001

Description	**Ore Type**		
	Weathered	Transitional	Fresh
Ore Mining Diff.	0.30	0.30	0.17
Grade Control	0.97	0.97	0.97
Mine Admin	1.28	1.28	1.28
Processing cost	8.11	8.11	8.11
Mine support costs	2.91	2.91	2.91
Rehabilitation	-	-	-
CostP ($/t)	13.57	13.57	13.44
Recovery (%)	93.1	93.1	93.1

Table 5.1.17 Final pit quantities Dam A

Pit	Gold Price	ORE		WASTE	TOTAL	Cash Cost
	$/oz	kt	g/t	kt	kt	$/oz
Dam A	500	48.2	2.13	344	392	401

5.1.7 Prospect Claim Optimisation

Prospect Claim was originally optimised in November 2002 however, work continued through until March 2003 due to additional drilling and updating of the models. The optimisation report (19) is included on the CD. Below are the optimisation parameters and results.

Table 5.1.18 Optimisation of Prospect Claim Waste Mining Costs November 2002

Prospect Claim		Waste ($/bcm)			
Bench Toe (rl)	**SG**	**L&H**	**D&B**	**Other**	**Total**
1207.5	2.5	2.471	1.22	1.37	5.061
1205	2.5	2.471	1.22	1.37	5.061
1202.5	2.5	2.471	0.76	1.37	4.601
1200	2.5	2.471	0.76	1.37	4.601
1197.5	2.5	2.471	0.76	1.37	4.601
1195	2.5	2.521	0.76	1.37	4.651
1192.5	2.5	2.521	0.76	1.37	4.651
1190	2.5	2.521	0.76	1.37	4.651
1187.5	2.5	2.662	0.76	1.37	4.792
1185	2.5	2.662	0.76	1.37	4.792
1182.5	2.5	2.662	0.77	1.37	4.802
1180	2.5	2.662	0.77	1.37	4.802
1177.5	2.5	2.662	0.77	1.37	4.802
1175	2.6	2.662	0.77	1.37	4.802
1172.5	2.6	2.801	1.4	1.37	5.571
1170	2.6	2.801	1.4	1.37	5.571
1167.5	2.6	2.801	1.4	1.37	5.571
1165	2.6	2.801	1.4	1.37	5.571
1162.5	2.6	2.801	1.41	1.37	5.581
1160	2.6	2.801	1.41	1.37	5.581
1157.5	2.6	2.801	1.41	1.37	5.581
1155	2.7	2.801	1.41	1.37	5.581
1152.5	2.7	2.801	1.57	1.37	5.741
1150	2.7	3.111	1.57	1.37	6.051
1147.5	2.7	3.111	1.57	1.37	6.051
1145	2.7	3.111	1.57	1.37	6.051
1142.5	2.7	3.111	1.62	1.37	6.101
1140	2.7	3.111	1.62	1.37	6.101
1137.5	2.7	3.111	1.62	1.37	6.101
1135	2.7	3.111	1.62	1.37	6.101
1132.5	2.7	3.111	1.77	1.37	6.251
1130	2.7	3.111	1.77	1.37	6.251

Table 5.1.19 Optimisation of Prospect Claim Ore Mining Costs November 2002

Prospect Claim		Ore ($/bcm)			
Bench Toe (rl)	**SG**	**L&H**	**D&B**	**Other**	**Total**
1207.5	2.5	2.427	1.22	1.37	5.017
1205	2.5	2.427	1.22	1.37	5.017
1202.5	2.5	2.427	0.76	1.37	4.557
1200	2.5	2.427	0.76	1.37	4.557
1197.5	2.5	2.427	0.76	1.37	4.557
1195	2.5	2.605	0.76	1.37	4.735
1192.5	2.5	2.605	0.76	1.37	4.735
1190	2.5	2.605	0.76	1.37	4.735
1187.5	2.5	2.754	0.76	1.37	4.884
1185	2.5	2.754	0.76	1.37	4.884
1182.5	2.5	2.754	0.77	1.37	4.894
1180	2.5	2.754	0.77	1.37	4.894
1177.5	2.5	2.754	0.77	1.37	4.894
1175	2.6	2.754	0.77	1.37	4.894
1172.5	2.6	2.974	1.4	1.37	5.744
1170	2.6	2.974	1.4	1.37	5.744
1167.5	2.6	2.974	1.4	1.37	5.744
1165	2.6	2.974	1.4	1.37	5.744
1162.5	2.6	2.974	1.41	1.37	5.754
1160	2.6	2.974	1.41	1.37	5.754
1157.5	2.6	2.974	1.41	1.37	5.754
1155	2.7	2.974	1.41	1.37	5.754
1152.5	2.7	2.974	1.57	1.37	5.914
1150	2.7	3.284	1.57	1.37	6.224
1147.5	2.7	3.284	1.57	1.37	6.224
1145	2.7	3.284	1.57	1.37	6.224
1142.5	2.7	3.284	1.62	1.37	6.274
1140	2.7	3.284	1.62	1.37	6.274
1137.5	2.7	3.284	1.62	1.37	6.274
1135	2.7	3.284	1.62	1.37	6.274
1132.5	2.7	3.284	1.77	1.37	6.424
1130	2.7	3.284	1.77	1.37	6.424

Table 5.1.20 Optimisation of Prospect Claim Processing Costs November 2002

Prospect Claim	Processing Cost ($/t)						
Bench Toe (rl)	Ore Waste Mining diff.	Other	GC	Mill	Maint.	Admin.	Total
1207.5	-0.02	0.09	2.25	5.85	1.69	1.33	11.19
1205	-0.02	0.09	2.25	5.85	1.69	1.33	11.19
1202.5	-0.02	0.09	2.25	5.85	1.69	1.33	11.19
1200	-0.02	0.09	2.25	5.85	1.69	1.33	11.19
1197.5	-0.02	0.09	2.25	5.85	1.69	1.33	11.19
1195	0.03	0.09	2.25	5.85	1.69	1.33	11.24
1192.5	0.03	0.09	2.25	5.85	1.69	1.33	11.24
1190	0.03	0.09	2.25	5.85	1.69	1.33	11.24
1187.5	0.04	0.09	2.25	5.85	1.69	1.33	11.25
1185	0.04	0.09	2.25	5.85	1.69	1.33	11.25
1182.5	0.04	0.09	2.25	5.85	1.69	1.33	11.25
1180	0.04	0.09	2.25	5.85	1.69	1.33	11.25
1177.5	0.04	0.09	2.25	5.85	1.69	1.33	11.25
1175	0.04	0.09	2.25	5.85	1.69	1.33	11.25
1172.5	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1170	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1167.5	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1165	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1162.5	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1160	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1157.5	0.07	0.09	2.25	5.85	1.69	1.33	11.28
1155	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1152.5	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1150	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1147.5	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1145	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1142.5	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1140	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1137.5	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1135	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1132.5	0.06	0.09	2.25	5.85	1.69	1.33	11.27
1130	0.06	0.09	2.25	5.85	1.69	1.33	11.27

Table 5.1.21 Optimisation of Prospect Claim Cost P and Recoveries November 2002

	Ore		
	Weathered	**Transitional**	**Fresh**
CostP ($/t)	11.23	11.27	11.27
Mill recovery (%)	93.7	93.7	93.7

Table 5.1.22 Optimisation of Prospect Claim Overall Wall Angles November 2002

Rock Type	**Azimuth**	**RL**	**Overall Angle Base**
OXID	0-360	>1177.5	41.0°
TRAN	0-360	1177.5-1157.5	45.0°
FRES	0-360	<1157.5	49.0°

Table 5.1.23 Final pit quantities Prospect Claim

Pit	Gold Price	ORE		WASTE	TOTAL	Cash Cost
	$/oz	kt	g/t	kt	kt	$/oz
Prospect Claim	560	255	2.59	2,565	2,819	419

5.1.8 Esmeralda Optimisation

Esmeralda optimisation work is not discussed in this section however, the report has been included on the CD.

5.1.9 Optimisation References

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[18] MineConsult Pty Ltd. Shadow Estimate and Tender Submissions for the Union Reefs Mining Contract. September 2000. Technical Report UR0???.

[19] Davies G. Union Reefs Gold Mine. Optimisation of Prospect Claim Deposit. November 2002. Technical Report UR0???.

5.2 Design Vs Actual Data

5.2.1 Actual Pit Specifications

Table5.2.1 at the end of this chapter lists the final “Actual” characteristics for each of the pits mined at the Union Reefs Gold Mine. Characteristics include total volume, pit depth, overall slope angle, and at what depth (mRL) changes in weathering, batter angles, bench heights, berm widths, ramp widths, and ramp grade occurred. Following that are summary tables and graphs for yearly volume mined by material, summary totals by pit, then totals by pit and year.

Some pits include variations as they were redesigned (partially or completely) after works had already begun. A copy of each of these pit designs may be viewed on disc as part of the mining appendix.

5.2.2 Mining Summary – Actuals by Pit

Also at the end of this chapter **Table** 5.2.2(a) presents a life of mine **summary** spreadsheet broken down by pit. This includes volumes mined of Ore (ROM and Leach), Waste (Mineralised and Waste 3), strip ratio, then respective tonnes and grade for each material (which are also totaled).

Following that **Figure** 5.2.2(a) displays a visual chart of the Ore/Waste component volumes mined by pit as a percentage of the total mined **for that pit** (i.e. each pit adds up to 100%)

After that another chart (Figure 5.2.2(b)) displays **<u>Total</u>** volume mined per year by material (i.e. ROM ore, Leach Ore, Mineralised waste and Waste).

Figure 5.2.2(c) exhibits total mining volumes by year and **<u>Pit</u>**. As can be seen, mining of all the significantly smaller satellite pits generally took place between 2001 and 2003, therefore (for clarity), an enlargement of this time frame has been placed in the top right hand corner of this chart.

A further visual breakdown attempt of this chart was made per year, per pit **and Material**, however, this chart, although **extremely** useful, proved excessively cluttered so was removed from the main body of this report. For the brave, Figure 5.2.2(d) may be viewed under the mining Appendix along with several other more detailed tables that are broken down by pit, material and **year** of mine life. These spreadsheets are combined into one excel file titled “Mining Actuals – Pit Specs – Actuals by Pit and Year – Tables and Graphs”.

5.2.3 Brief Annual Summary

When viewing all the afore mentioned mining tables and charts the following information can be observed on an annual basis.

For the first three years (up until the end of 1996), mining was exclusively from the Crosscourse pit. Approximately 70% of that was waste with the remaining 30% composed of ROM ore, Mineralised Waste and Leach ore.

In 1997 mining from Crosscourse South, Lady Alice, Prospecting Claim and Union North pits began but combined, only contributed approximately 20% of the total volume mined for that year.

Total mining volumes peaked in 1998, as did the **waste** component. Crosscourse mining was reduced to account for only 60% of that year's total mining volume. The high waste volumes recorded were largely attributed to the Joy Hill (Lady Alice) cutback and stripping of the Union North pit (23% and 17% of total mining volumes respectively). Stockpiling of Leach ore had now also essentially ceased.

Mining of the **Crosscourse** pit peaked in 1999 and accounted for almost 90% of total volumes mined for that year. Approximately 85% of that mined was waste, resultant of the excavations from Ping Ques, Joy Hill and Western cutbacks within the main Crosscourse pit. Stockpiling of Mineralised waste was now also completed (approximately 3.6mt at an average grade of 0.53g/t).

Again in 2000, mining from Crosscourse dominated the total mining volumes for the year, this time accounting for almost 95%. The remainder came from the Crosscourse South pit. ROM ore peaked to just over 3mt at an average grade of 1.43g/t, almost 100% of which came from the Crosscourse pit.

By 2001 the total volumes mined were shared almost equally amongst Lady Alice (27%), Union North (23%), Crosscourse (22%) and Crosscourse South (21%) pits. The remaining 7% came from stripping the Prospecting Claim pit.

By 2002, total mining begun to drop considerably with the completion of the third Western Cutback in late December 2001. Approximately 40% of the total ore mined came from the Crosscourse pit at a strip ratio of 0.37:1, however, the overall strip ratio for 2002 remained at 3.2:1 as stripping of the satellite ore bodies such as Big Tree, Dam A, Millars, Ping Ques South, Temple and Union South began. As can be seen in the enlargement of figure 5.2.2(c), similar contributions were made to the annual volume mined from each of these satellite pits.

Mining was completed by the end of July 2003 and was predominately from the Prospecting Claim and Union North South pits. These pits combined accounted for approximately 87% of total mining volumes for that year. The remainder comprised of remnant ore blocks being mined from the Crosscourse ramp, and completion of the Union South pit.

5.3 Equipment Used

Also at the end of this chapter is a summary table of all the **different types** of equipment ever used at the Union Reefs Gold Mine (Table 5.3.1). A more detailed list of **every** piece of equipment ever used at URGM can be found in the mining appendix burnt to CD. Both are generated from the equipment list found in the MINEMAN database (also burnt to CD), and as such may not be entirely complete.

5.4 Dewatering

A number of dewatering bores were drilled around the main Cross Course pit before production began, however these proved largely ineffective as the most productive bore only pumped approximately 1-2 Litres per second.

Whilst dewatering Crosscourse pit, a staged diesel pumping arrangement was employed down to the 1070mRL, whereby an electric pump station was established at the 1080mRL (2 x 110kW pumps). Diesel pumps were re-established each bench below that pumping back to the electrics against a vertical head up to 130m (at the base of the CC pit - the 952.5mRL).

The majority of the dewatering occurred from within pit. During the wet seasons when heavy rainfalls were experienced (December to February) from 2000 – 2003, up to 180litres per second was being pumped from the bottom of CC with the blue HH200 diesel pump.

The satellite pits were predominately dewatered using electric FLYGHT pumps. A generator was set up at the top of the pit, with electric cable trailing down over the pits edge to supply power.

At the very base of the Crosscourse pit (during the middle of the dry season), two FLYGHT pumps were successfully joined together in “series” that enabled a head of 130m to be pumped up to the electric staging pumps.

Table 5.2.1: Actual pit specifications

Pit	Voulme	Depth	Slope	Weathering *			Batter Angles			Bench Heights			Berm Widths			Ramp Widths			Ramp Grade		
	BCM	(m)	Angle	From	To	O/T/F	From	To	Angle	From	To	Height	From	To	Width	From	To	Width	From	To	Grade
			Deg.	mRL	MRL		mRL	MRL	Deg.	mRL	MRL	(m)	mRL	MRL	(m)	mRL	MRL	(m)	mRL	MRL	
Alta Orinoco	108,026	18		1178	1160	Oxide	1178	1160	60	1178	1160	10									
Big Tree	209,423	40	45 (E)	1193	1168	Oxide	1193	1153	60	1193	1153	10	1193	1153	3	1193	1153	12	1193	1153	1 in 8
			45(W)	1168	1153	Trans.															
Crosscourse	30,553,817	265	47 (E)	1218	1165	Oxide	1218	1160	50 (E)	1218	1002	7.5	1218	1002	5	1218	1017	20	1218	1017	1 in 10
			36(W)	1165	1145	Trans.	1160	1137	60 (E)	1002	950	10	1002	952	3	1017	952	12	1017	952	1 in 8
				1145	952	Fresh	1137	1115	70 (E)												
							1115	952	75 (E)												
Crosscourse South	1,602,892	120	47 (E)	1220	1170	Oxide	1220	1100	65	1220	1130	7.5	1220	1130	6	1220	1160	20	1220	1160	1 in 10
				1170	1160	Trans.				1130	1100	10	1130	1100	3	1160	1100	12	1160	1100	1 in 8
				1160	1100	Fresh															
Western Cutback		175	36(W)	1185	1165	Oxide	1185	1085	45(W)	1185	1010	7.5	1185	1070	7.5	1185	1010	12	1185	1010	1 in 8
				1165	1150	Trans.	1085	1070	50(W)				1070	1010	5						
				1150	1010	Fresh	1070	1010	55(W)												
Dam A	374,835	55	37 (E)	1175	1150	Oxide	1175	1120	50(W)	1175	1120	10	1175	1120	3	1175	1120	12	1175	1120	1 in 8
			33(W)	1150	1120	Fresh	1175	1120	60 (E)												
Lady Alice	4,166,652	100	37 (E)	1227	1177	Oxide	1227	1177	50	1227	1127	10	1227	1127	6	1227	1167	20	1227	1127	1 in 8
			37(W)	1177	1157	Trans.	1177	1167	55							1167	1127	12			
				1157	1127	Fresh	1167	1127	60												
Millars	302,593	40	38 (E)	1197	1157	Oxide	1197	1157	50	1197	1157	10	1197	1157	3	1197	1157	12	1197	1157	1 in 8
			46 (E)																		
Prospecting Claim	1,409,244	60	45 (E)	1207	1177	Oxide	1207	1177	50	1210	1150	10	1210	1150	3	1210	1180	20	1210	1150	1 in 8
			35(W)	1177	1167	Trans.	1177	1157	55							1180	1150	12			
				1167	1147	Fresh	1157	1147	60												
Pingque South	294,088	35	60 (E)	1210	1175	Oxide	1210	1175	60	1210	1175	10	1210	1175	3	1210	1175	12	1210	1175	1 in 8
Temple	48,669	20	60	1203	1183	Oxide	1208	1183	60	1208	1203	5	1208	1183	3	1208	1183	12	1208	1183	1 in 8
										1203	1183	10									
Union North	3,781,559	100	40 (E)	1210	1170	Oxide	1210	1130	50(W)	1210	1130	7.5	1210	1130	3	1210	1145	20	1210	1140	1 in 10
			33(W)	1170	1150	Trans.	1210	1130	60 (E)							1145	1130	12	1140	1130	1 in 8
				1150	1130	Fresh															
Union South	526,371	50	40 (E)	1195	1175	Oxide	1195	1175	55	1195	1145	10	1195	1145	3	1195	1145	12	1195	1145	1 in 8
			40(W)	1175	1155	Trans.	1175	1155	60												
				1155	1145	Fresh															

Table 5.2.2(a): Mining Summary - Actuals by pit

Mining Summary: "Actuals by Pit"														
Pit	Volume (BCM)					Strip Ratio	Tonnes (t) and Grade (g/t)							Total
	Ore		Waste		Total		ROM Ore		Leach Ore		Min. Waste		Waste	
	ROM	Leach	Mineral.	Waste			Tonnes	g/t	Tonnes	g/t	Tonnes	g/t	Tonnes	Tonnes
Alluvials	5,639				5,639	0.00	14,322	1.49						14,322
Alta Orinoco	16,306			91,720	108,026	5.62	41,091	0.98					231,134	272,225
Big Tree	21,605	5,050		182,768	209,423	6.86	55,255	1.52	12,922	0.79			465,726	533,903
Crosscourse	4,886,923	476,988	1,191,568	26,299,511	32,854,990	5.13	13,036,535	1.70	1,205,356	0.87	3,067,823	0.53	69,009,071	86,318,785
Crosscourse South	227,382	2,635	31,268	1,341,607	1,602,892	5.97	620,935	1.40	7,139	0.69	85,475	0.62	3,584,609	4,298,158
Dam A	50,538	1,560		322,737	374,835	6.19	131,593	1.57	3,931	0.73			829,311	964,835
Fine Ore Bin	8,058	9,512			17,570	0.00	20,285	1.66	23,970	0.83				44,255
Lady Alice	232,780	9,403	-	1,623,296	1,865,479	6.70	611,717	1.55	25,485	0.81	-	0	4,167,561	4,804,763
Millars	51,248	8,907		242,438	302,593	4.03	129,690	1.57	22,509	0.83			612,718	764,917
Prospecting Claim	155,237	9,722	7,635	1,236,650	1,409,244	7.54	400,146	1.65	24,499	0.64	19,241	0.58	3,134,296	3,578,182
Pingque South	50,632	3,988		239,468	294,088	4.38	127,592	1.36	10,049	0.84			603,460	741,101
Spillway				25,740	25,740	0.00							64,865	64,865
Temple	4,813	4,664		39,192	48,669	4.14	12,224	1.02	11,847	0.68			99,547	123,618
Union North	576,515	24,382	180,583	3,000,079	3,781,559	5.29	1,498,498	1.30	62,042	0.85	455,068	0.47	7,711,159	9,726,767
Union South	76,213	6,168		443,990	526,371	5.39	194,724	1.26	15,854	0.71			1,123,887	1,334,465
Waste Rehandle				154,989	154,989	0.00							392,894	392,894
						0.00								-
Total	**6,363,889**	**562,979**	**1,411,054**	**35,244,185**	**43,582,107**	**5.29**	**16,894,607**	**1.63**	**1,425,603**	**0.86**	**3,627,607**	**0.53**	**92,030,238**	**113,978,055**

Notes: **Strip Ratio is calculated whereby High Grade, ROM, Marginal and Leach imply "ORE", and Mineralised Waste and Waste imply "WASTE"**

Figure 5.2.2(a): Ore waste component percentages by pit (chart)

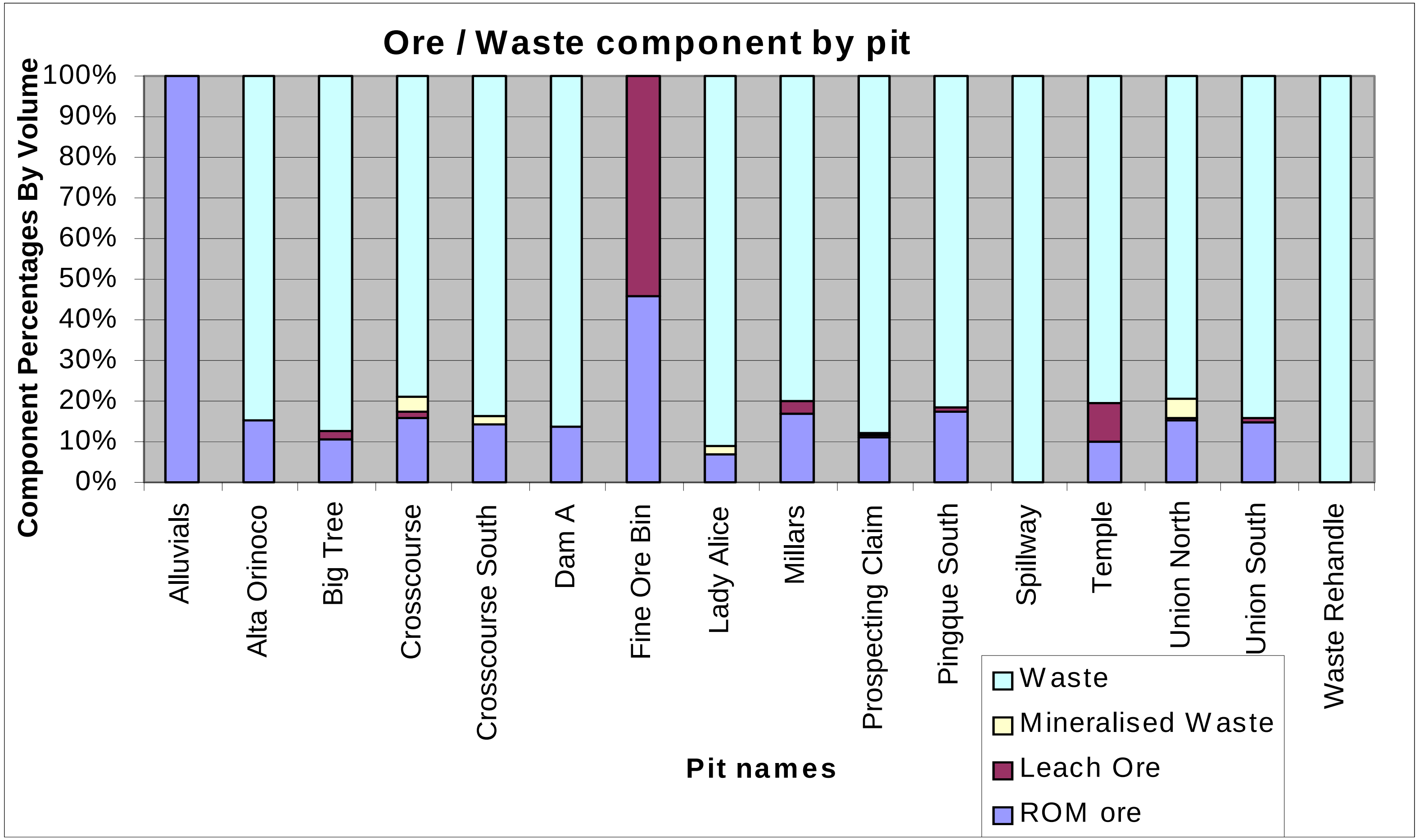


Figure 5.2.2(b): Volume of material mined per year and material (summary graph)

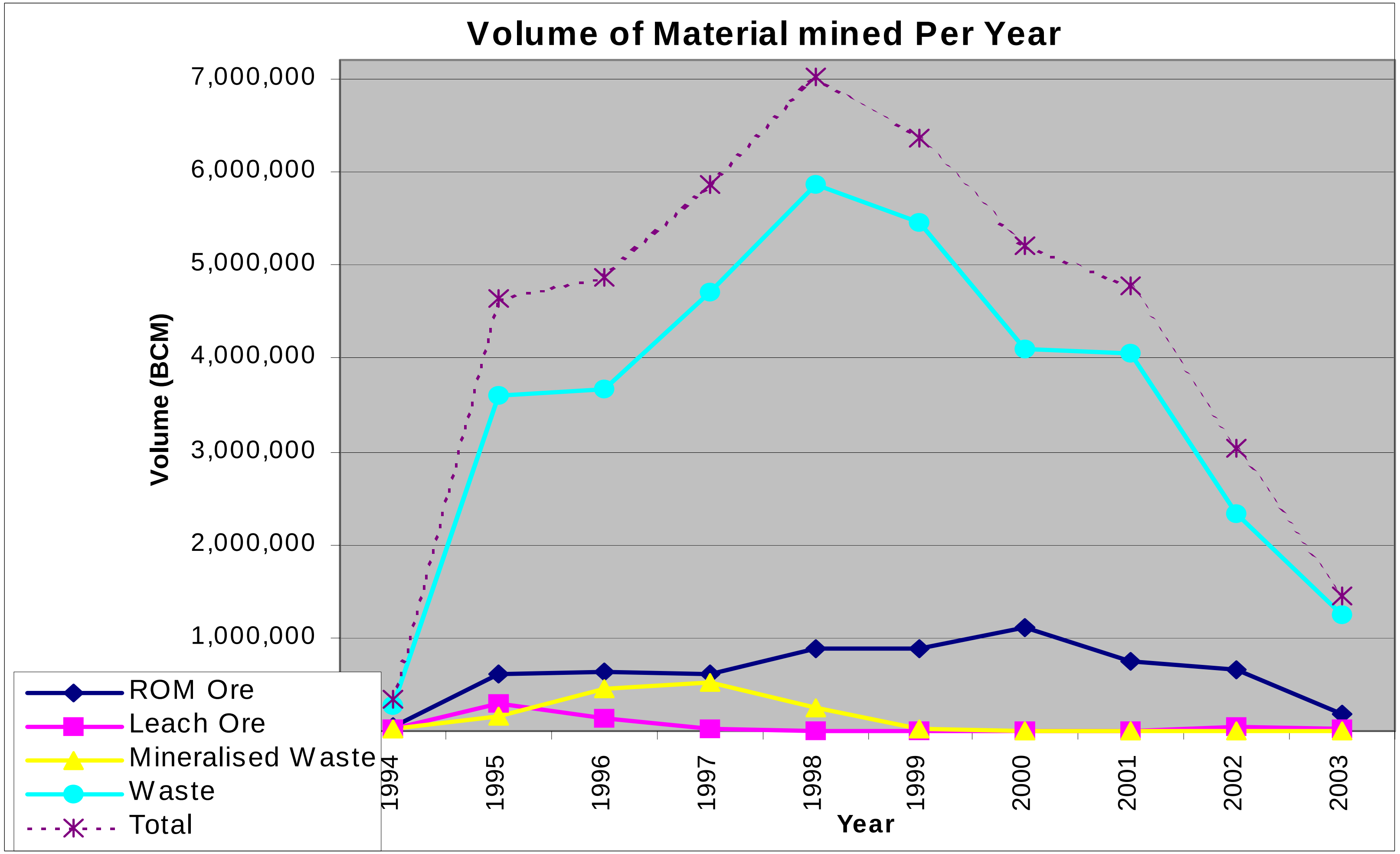


Figure 5.2.2(c): Mining Volumes by Pit and Year

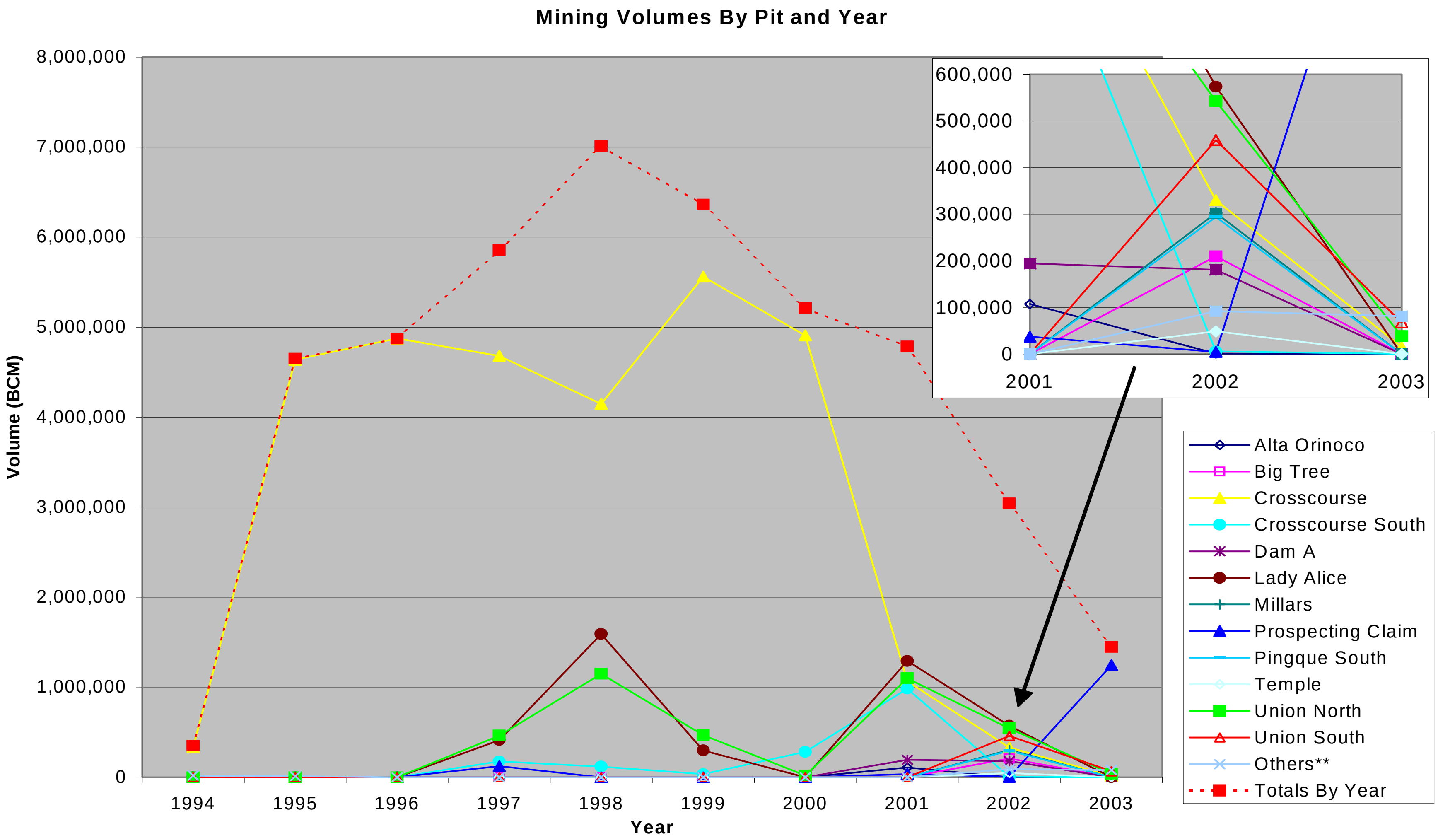


Table 5.3.1: Different types of Equipment used at URGM

	Equip. ID#	Equipment Group	Equipment Name	Contractors Code	Equipment Designation
Drills	B1	Drill	Tam 500 (11)	DR11	Blast Hole Rig 11
	B12	Drill	Schramm 2 - (RC)	SHRM2	Gadens Track Rig - Yellow
	B13	Drill	650-5 - (RC)	650-5	Gadens RC Rig
	B14	Drill	Drillcorp		Drillcorp
	B2	Drill	Tam 1100 (88)	DR88	Blast Hole Rig 88
	B4	Drill	Drilltech (116)		RC Drill Rig 116
	B5	Drill	UDR 650 (106)	DR106	Multipurpose Rig
	B7	Drill	Hidrill	DR110	Diamond Drill Rig
	B8	Drill	Warman 1000-1 (RC)	10001	Gadens Multi-purpose Drill Rig8
	B9	Drill	Gomex RCD2		Gomex RC rig RCD 2
	BA	Drill	Gomex Rota 50		Gomex grade control Rota 50
	BB	Drill	Warman 1000-5 (Diamond)	10005	Warman 1000-5
	BD1	Drill	Montabert 3189	3189	Blast Hole Rig 3189
	BS	Drill	RCD9 Yellow GC Rig	RCD9	Yellow GC Rig - RCD9
	BT	Drill	Drillex Rig 5		Grade control rig
	RO7	Drill	GD5000		GD5000
	RO8	Drill	SK45i		SK45i
	R32	Drill	DM45	7749	DM45 Tramming
Dozers	D1	Dozer	D8 /K		Out of Service
	D10	Dozer	D10N		Dozer
	D3	Dozer	Komatsu D155/A		KOMATSU D155/A
	D4	Dozer	Komatsu D85		Komatsu D 85
	D5	Dozer	Cat D9N	1115	Dozer D9
	D7	Dozer	Cat 824 Dozer	824	Dozer 824
	D8N	Dozer	D8N Dozer		D8N Dozer
	DO2	Dozer	Cat 690D Tiger	Tiger	Cat 690D Tiger
	DO3	Dozer	Cat D9L		Cat D9L Dozer Roche
	R18	Dozer	D8R	31099	Cat D8R Dozer
Excavators	E1	Exc	Liebherr 984	1237	Excavator 984
	E10	Exc	EX1100	22066	Hitachi EX1100 ROCHE
	E2	Exc	Cat 5130		Excavator 5130
	E3	Exc	Komatsu 650		Excavator
	E5	Exc	Hit EX1800		180t Excavator
	E6	Exc	Hit EX300		Union Extended Excavator
	EX4	Exc	EX 1800 7002	7002	EX 1800 7002
	EX6	Exc	994 Liebherr	H0660	Excavator - New rate Mar2000
	EX7	Exc	EX2500	22210	Hitachi EX2500 ROCHE
	EX8	Exc	PC1000	3501	Komatsu PC1000 ROCHE
	EX9	Exc	EX1200	22075	Hitachi EX1200 ROCHE
	R2	Exc220	Exc220 Rockbreaker	Ex220	Rockbreaker

	Equip. ID#	Equipment Group	Equipment Name	Contractors Code	Equipment Designation
Graders	G1	Grader	Cat 12G	1391	Grader 12G
	G2	Grader	Cat 16G	1387	Grader 16G 1387
	G4	Grader	Cat 14G		Grader 14G
	G6	Grader	Cat 16H		Cat 16H
	RO3	Grader	14H	22560	14H grader
Loaders	L1	Loader	Cat 966F	1417	Loader 966F
	L2	Loader	Cat 980F	1473	Loader 980F
	L3	Loader	Cat 988B	1483	Loader 988B
	L5	Loader	Komatsu WA 600		Loader Komatsu WA 600
	LD1	Loader	Cat 988B 1483	1483	Cat 988B 1483
	LD2	Loader	Cat 980F		Cat 980F
	LD5	Loader	Cat 990		Cat 990
	R31	Loader	950 loader		950
Labourers	R10	Person	Labourer		Roche Labourer
	R11	Person	Tradesman		Roche Tradesman
	R12	Person	Serviceman		Roche Serviceman
	R13	Person	Driller		Roche Driller
	R14	Person	Shotfirer		Roche Shotfirer
Trucks	R19	Truck	D21188	21188	Cat 777D
	R20	Truck	B21100	21100	Cat 777B
	RO1	Truck	777C		777C
	T6	Truck	Cat 2290	2290	85 Tonne Dump Truck 2290
	T8	Truck	Cat B 2294	2294	50 Tonne Dump Truck 2292
	TV	Truck	Road Train		Road Train
	R23	ServiceTruck	Service Truck		Fuel Service Truck
	M1	Services	Mercedes Truck	2457	Workshop service truck/fuel
	RO4	Watercart	773WC		773 Watercart
	W3	Water Cart	Cat 769		Watercart 769
	WC3	Mack	Water Cart Mack		Water Cart Mack
	WC4	Water Cart	Water Cart Scania		Water Cart Scania
	TLW	Water Cart	Water Cart		Tails Lift Water Cart
Cranes					
	C1	CRANE	8 TONNE CRANE	1738	8 TONNE CRANE
	WE1	Workshop	Yard Crane	1738	Yard Crane
S&J	R24	S&J	D7 Dozer		Cat D7 Dozer
	R26	S&J	Roller		Padfoot Roller
	R1	Compactor	Cat compactor roller	825C	Compactor hourly hire
	R27	S&J	Water Cart		Water Cart 8 wheel
	S1	Scraper	Cat 633D		Scraper
	TLG	Grader	14D		Tails Lift Grader

6.0 Costs

6.1 Drill and Blast Costs

Below is a summary Drill and Blast life of mine costs per BCM table (6.1.1) (from 1st of October 1997 onwards). This details all the costs associated with drilling, labour (priming, loading and stemming), bulk and initiating accessory explosives and includes total Design Vs Actual Powder Factors for the life of the mine. Table 6.1.2 (following) separates the costs for various rock material types of Fresh, Transitional, Oxide and Presplit.

Table 6.1.1: Drill and Blast costs per BCM

	Summary		Cost	
	Unit	**Amount**	**($)**	**($/BCM)**
Volume Blasted	BCM	27,318,802		
Drilling	Metres	2,724,869	$19,172,983.97	$0.70
Redrill	Metres	258,161		0.01
	(%)	9.5%		
Holes (Prime/Load/Stem)	No.	325,366	$2,524,305.80	$0.09
Bulk Explosive Mass	kg	17,061,214	$14,389,082.47	$0.53
Accessories	(IE and HE)		$3,841,131.08	$0.14
Powder Factor (kg/m3)	Actual	0.62		
	Design	0.55		
Total			**$39,927,503.32**	**$1.46**

Table 6.1.2: Drill and Blast costs ($/BCM) for various rock types

	Fresh	Transitional	Weathered	Presplit	Total
	Actual	**Actual**	**Actual**	**Actual**	**Actual**
BCM's	16,160,983	1,132,615	10,022,334	2,870	27,318,802
Drilling	$0.83	$0.56	$0.50	$ 12.59	$0.70
Loading	$0.11	$0.08	$0.07	$1.12	$0.09
Bulk Explosive	$0.61	$0.40	$0.41	$0.00	$0.53
Accessories	$0.17	$0.10	$0.10	$3.01	$0.14
Total	**$1.72**	**$1.14**	**$1.08**	**$16.72**	**$1.46**

6.2 Mining Costs

Below is a summary table (6.2.1) that displays the Total cost, then $/BCM, $/Tonne and $/Ounce for the departments of Exploration, Geology, Mining, Processing, Maintenance, and Administration.

Table 6.2.1 Total Cost by Department ($ per BCM, $/Tonne, and $/Ounce)

SUMMARIES	Cost By Dept. ($)		Totals / Averages
	Total ($)	Exploration	9,673,580
		Geology	19,547,811
		Mining	160,595,906
		Processing	108,759,497
		Maintennance	34,618,718
		Administration	23,427,377
		Total	**346,949,308**
	$ / BCM	Exploration	0.25
		Geology	0.50
		Mining	4.08
		Processing	2.77
		Maintennance	0.88
		Administration	0.60
		Total	**8.82**
	$ / Tonne	Exploration	0.53
		Geology	1.06
		Mining	8.75
		Processing	5.92
		Maintennance	1.89
		Administration	1.28
		Total	**18.90**
	$ / Ounce	Exploration	12.07
		Geology	24.39
		Mining	200.35
		Processing	135.68
		Maintennance	43.19
		Administration	29.23
		Total	**432.84**

At the end of this chapter, summary table (6.2.2) displays the Total Costs, Cost per BCM, Cost per tonne, and Cost per ounce (in AUD) for each department broken down per calendar Year of operation. A more detailed four page breakdown of costs may be found in file called "Life of mine Costs" under the Costs appendix burnt to CD. This includes costs associated with Exploration and Production Geology (drilling and assaying), Drill and blast (ore and waste), Mining (ore and waste) and for the last four years the indirect costs associated with mining. Observe the increase in information during the later years – early documentation for costs are scarce, contradictory and therefore unreliable.

Also at the end of this chapter table 6.2.3 demonstrates approximate costs per pit and per year. This is **representative** only as no pit by pit detailed cost breakdown exists. Essentially, costs per pit and year have been determined simply by apportioning total costs to the respective pits by the percentage that pit contributed to the total mining volumes for that year. Two more detailed cost breakdowns calculated in the same manner can be found in the file "Costs Per Pit Per Year", under the **cost** section of the appendices (also burnt to CD). The first, table 6.2.4, further breaks down costs by pit, year and **MATERIAL**, whilst the second, table 6.2.5 breaks down costs by pit, year and **DEPARTMENT**. Again, both are only representative, as apportioned relative to the contributions made by mining volumes for that year.

Table 6.2.2: Total Life of Mine Costs Summary per year ($ per BCM/tonne/ounce)

			1994	1995	1996	1997	1998	1999	2000	2001	2002	2003	Totals / Averages
SUMMARIES	Total ($)	Exploration			2,799,697	3,202,910	1,657,201	1,020,016	654,943	338,813			**9,673,580**
		Geology				3,337,566	2,804,880	2,490,136	2,757,996	2,945,757	3,668,670	1,542,806	**19,547,811**
		Mining			16,852,629	17,421,714	24,093,520	25,742,299	26,618,542	22,773,509	17,728,083	9,365,610	**160,595,906**
		Processing			10,501,749	10,296,101	14,546,921	15,738,696	16,402,811	15,716,751	16,166,742	9,389,727	**108,759,497**
		Maintennance			3,237,444	3,696,759	3,840,877	4,790,710	6,739,461	4,976,632	4,853,238	2,483,597	**34,618,718**
		Administration			2,245,381	2,058,149	2,837,634	3,138,655	3,749,455	3,564,188	3,637,414	2,196,501	**23,427,377**
		Total	**-**	**-**	**32,837,203**	**36,810,289**	**48,123,832**	**51,900,495**	**56,268,265**	**49,976,837**	**46,054,147**	**24,978,241**	**346,949,308**
	$ / BCM	Exploration			0.58	0.55	0.24	0.16	0.12	0.07			0.25
		Geology				0.57	0.40	0.40	0.52	0.62	1.06	0.85	0.50
		Mining			3.47	2.99	3.44	4.08	5.02	4.77	5.15	5.17	4.08
		Processing			2.17	1.76	2.08	2.50	3.09	3.30	4.69	5.18	2.77
		Maintennance			0.67	0.63	0.55	0.76	1.27	1.04	1.41	1.37	0.88
		Administration			0.46	0.35	0.40	0.50	0.71	0.75	1.06	1.21	0.60
		Total			6.77	6.31	6.87	8.23	10.61	10.48	13.37	13.78	8.82
	$ / Tonne	Exploration			1.63	1.88	0.69	0.37	0.23	0.13			0.53
		Geology				1.96	1.18	0.90	0.98	1.11	1.34	0.97	1.06
		Mining			9.84	10.23	10.10	9.28	9.43	8.61	6.50	5.90	8.75
		Processing			6.13	6.04	6.10	5.67	5.81	5.94	5.93	5.91	5.92
		Maintennance			1.89	2.17	1.61	1.73	2.39	1.88	1.78	1.56	1.89
		Administration			1.31	1.21	1.19	1.13	1.33	1.35	1.33	1.38	1.28
		Total			19.17	21.61	20.17	18.70	19.93	18.90	16.88	15.73	18.90
	$ / Ounce	Exploration			32.13	45.07	15.61	8.36	5.17	2.95			12.07
		Geology				46.96	26.42	20.41	21.76	25.63	30.96	28.06	24.39
		Mining			193.42	245.14	226.93	210.96	210.02	198.13	149.59	170.36	200.35
		Processing			120.53	144.88	137.02	128.98	129.42	136.74	136.42	170.80	135.68
		Maintennance			37.16	52.02	36.18	39.26	53.17	43.30	40.95	45.18	43.19
		Administration			25.77	28.96	26.73	25.72	29.58	31.01	30.69	39.96	29.23
		Total			376.88	517.96	453.27	425.33	443.96	434.80	388.61	454.37	432.84

Table 6.2.3: Approximate costs per pit per year

Pit		Year									
		1994	1995	1996	1997	1998	1999	2000	2001	2002	2003
Alta Orinoco	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,119,164	$ 13,536	$ -
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	2.24%	0.03%	0%
Big Tree	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 3,170,876	$ -
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	0%	7%	0%
Crosscourse	Total ($) For Pit	$ -	$ -	$ 32,837,203	$ 29,431,968	$ 28,481,782	$ 45,366,679	$ 53,014,903	$ 11,157,236	$ 4,992,567	$ 262,029
	% of Total / Year	0%	0%	100%	80%	59%	87%	94%	22%	11%	1.05%
Crosscourse South	Total ($) For Pit	$ -	$ -	$ -	$ 1,109,811	$ 811,798	$ 283,225	$ 3,042,749	$ 10,297,971	$ 80,505	$ 10,284
	% of Total / Year	0%	0%	0%	3%	1.69%	0.55%	5%	21%	0.17%	0.04%
Dam A	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 2,028,154	$ 2,735,799	$ -
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	4%	6%	0%
Lady Alice	Total ($) For Pit	$ -	$ -	$ -	$ 2,590,451	$ 10,931,232	$ 2,422,047	$ -	$ 13,496,832	$ 8,683,098	$ -
	% of Total / Year	0%	0%	0%	7%	23%	5%	0%	27%	19%	0%
Millars	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,581,565	$ -
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	0%	10%	0%
Prospecting Claim	Total ($) For Pit	$ -	$ -	$ -	$ 770,903	$ -	$ -	$ -	$ 389,121	$ 64,970	$ 21,483,258
	% of Total / Year	0%	0%	0%	2%	0%	0%	0%	0.78%	0.14%	86%
Pingque South	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 4,452,790	$ -
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	0%	10%	0%
Temple	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 736,898	$ -
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	0%	1.60%	0%
Union North	Total ($) For Pit	$ -	$ -	$ -	$ 2,907,157	$ 7,899,019	$ 3,828,544	$ 210,613	$ 11,488,359	$ 8,204,506	$ 659,111
	% of Total / Year	0%	0%	0%	8%	16%	7%	0%	23%	18%	3%
Union South	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 6,947,605	$ 1,164,862
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	0%	15%	5%
Others**	Total ($) For Pit	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ -	$ 1,389,430	$ 1,398,697
	% of Total / Year	0%	0%	0%	0%	0%	0%	0%	0%	3%	6%
Totals By Year	Total ($) For Pit	$ -	$ -	$ 32,837,203	$ 36,810,289	$ 48,123,832	$ 51,900,495	$ 56,268,265	$ 49,976,837	$ 46,054,147	$ 24,978,241
	% of Total / Year	OK	OK	OK	OK	OK	OK	OK	OK	OK	OK

7.0 Reserves/Resources/Reconciliations

This section outlines reconciliation figures from the commencement of mining in November 1994 to the cessation of mining in July 2003. All figures are expressed in dry tonnes.

Overall, Ore Mined was 108% of the tonnes, 87% of the grade and 94% of the ounces when compared to the Reserve. Milled tonnes represent 98% of tonnes mined, milled grade represents 102% of predicted mined grade and recovered ounces represent 99% of predicted mined ounces

7.1 As Mined Ore versus Reserve Block Model

Reconciliation data is a general guide to the performance of each pit compared to the reserve block model. Overall, reconciliation data is incomplete due to poor archiving of old data, moreover changes in reconciliation procedures. During 1997 and 1998, reconciliation's were only done on ore above the 0.9g/t cut-off even though ore above 0.55g/t was stockpiled and later milled. **The final "As-Mined" figure illustrated below is only the ore that has been reconciled against the same reserve parcel at a certain cut-off used at that time. Final "As-Mined" figures do not represent "Total Ore Mined" and hence "Total Ore Milled". "Total Ore Mined" is presented in the mining statistics section of this report and tabulated below in the mine to mill section.**

Reserve figures are a combination of several models that have been revised throughout the life of mine. These are a combination of revised ID and MIK reserve block models.

Table 7.1.1 illustrates Grade Control and As Mined performance against the reserve. Grade Control numbers are total tonnes and grade delineated by grade control before ore blocks are designed. In the majority of cases grade control models were produced by conditional simulation. As Mined figures are actual tonnes and grade mined from each pit.

From Table 7.1.1, Grade control delineated 106% of the tonnes, 91% of the grade and 96% of the ounces against the reserve. Actual ore mined (As Mined) totaled 108% of the tonnes, 87% of the grade and 94% of the ounces against the reserve. Comparisons between as mined figures and the grade control model illustrate increased tonnage's and decreased grades which combine to a slight decrease in ounces. This is due to the design of ore blocks to a mineable width and length in which encompasses minor dilution (dilution skin). In several of the smaller pits additional ore was mined ("visual" ore) supervised by geologists and pit technicians. The majority of extra ore mined was grab sampled, assayed and in some cases campaigned through the mill. In all cases, grades returned were well above the 0.6g/t cut-off. In short, "ore" which was not delineated by grade control was mined thus reducing the "As mined" predicted grade and increasing the tonnes mined. This characteristic is attributed to the high degree of grade variability within the orebody (nugget effect). These positive results are also illustrated in Mine to Mill Call factors outlined in the next section.

The majority of tonnes mined (81% of total reconciled tonnes) occur from Crosscourse Pit. Crosscourse Pit includes: E-lens, Western Lens, Western Cut Back, E-South, Crosscourse South, Ping Ques and Lady Alice Ore domains.

Table 7.1.1: Reserve Reconciliation

Pits		Tonnes	% Reserve	Grade	% Reserve	Ounces	% Reserve
Alta	Reserve	61,177		1.77		3,481	
	Gradecontrol	54,881	98%	0.92	87%	1,623	47%
	As Mined	43,033	70%	0.94	86%	1,301	37%
Big Tree	Reserve	64,832		2.14		4,461	
	Gradecontrol	67,273	104%	1.44	67%	3,115	70%
	As Mined	64,083	99%	1.43	67%	2,946	66%
Lady Alice	Reserve	564,810		1.74		31,597	
	Gradecontrol	618,083	109%	1.51	87%	30,006	95%
	As Mined	573,396	102%	1.53	88%	28,206	89%
Millars	Reserve	133,458		1.81		7,766	
	Gradecontrol	153,400	115%	1.60	88%	7,891	102%
	As Mined	159,299	119%	1.45	80%	7,426	96%
Ping Ques South	Reserve	108,878		1.57		5,496	
	Gradecontrol	132,858	122%	1.39	89%	5,937	108%
	As Mined	126,250	116%	1.37	87%	5,561	101%
Temple	Reserve	20,899		0.99		665	
	Gradecontrol	19,758	95%	0.89	90%	565	85%
	As Mined	22,369	107%	0.86	87%	618	93%
Union North	Reserve	1,343,479		1.48		63,798	
	Gradecontrol	1,374,776	97%	1.30	95%	57,654	92%
	As Mined	1,343,265	95%	1.29	83%	55,836	79%
Union North South	Reserve	193,204		1.60		9,939	
	Gradecontrol	219,598	114%	1.37	86%	9,673	97%
	As Mined	223,660	116%	1.32	83%	9,492	96%
Union South	Reserve	189,890		1.47		8,975	
	Gradecontrol	193,307	102%	1.25	85%	7,769	87%
	As Mined	191,217	101%	1.23	84%	7,562	84%
Crosscourse	Reserve	12,860,847		1.86		769,524	
	Gradecontrol	13,621,711	106%	1.70	91%	743,902	97%
	As Mined	13,941,947	108%	1.63	88%	732,379	95%
Prospect Claim	Reserve	356,821		2.13		24,398	
	Gradecontrol	345,035	97%	1.86	88%	20,646	85%
	As Mined	411,994	115%	1.60	75%	21,242	87%
Dam A	Reserve	117,864		1.55		5,874	
	Gradecontrol	124,886	106%	1.65	106%	6,625	113%
	As Mined	127,721	108%	1.57	101%	6,447	110%
Total	Reserve	16,016,159		1.82		935,972	
	Gradecontrol	16,925,566	106%	1.65	91%	895,407	96%
	As Mined	17,228,234	108%	1.59	87%	879,015	94%

7.2 Mine Verses Mill

Pit by Pit reconciliation's are not possible due to ore being stockpiled by grade rather than pit. Moreover, ore is blended by grade to achieve the budgeted milled grade. Mine to Mill reconciliation's are based on Total Mined ore (mined to the ROM) and Total Milled ore. Note that Tonnes Mined are based on volumes converted to tonnes and Tonnes Milled are calculated by weightometer readings from the mill and adjusted with the removal of moisture.

Ore mined at URGM totals 21,905,221 tonnes at grade of 1.40 g/t. This equates to 987,169 oz Au. Of this, 16,875,981t at 1.63 g/t was High Grade (0.90g/t cut-off), 1,401,633t at 0.86 g/t was Low Grade (0.60g/t cut-off) and 3,627,607t at 0.53 g/t was Mineralised Waste (0.4 – 0.5g/t cut- off (depending on pit)). Total tonnes mined by pit is illustrated in Table 7.2.2 All High and Low Grade material was delivered or re-handled to the ROM for milling. All Mineralised Waste was initially stockpiled and 2,433,021t at 0.49g/t was re-handled to the ROM for Milling. Thus, ore mined for milling (High Grade + Low Grade + Ore Re-handled) was 20,710,635t at 1.45g/t for 963,512oz Au.

Ore milled to the end of July 2003 totals 19,904,921 tonnes at a recovered grade of 1.48 g/t and 949,039 oz Au. Ore remaining to be milled (ROM stocks and COS) totals 320,439t at 0.82g/t (predicted grade). From this, it is assumed that at the end of processing, total ore milled equates to 20,225,360t at 1.47g/t and 957,523 oz Au. This presumption is based on a 100% mine to mill call factor for remaining ROM stocks and COS. Milled ounces equates to ounces before recovery adjustments. A total of 155,908t of Maud Creek ore at 3.38g/t was milled but not included in these figures.

From the above data, milled tonnes represent 98% of tonnes mined, milled grade represents 102% of predicted mined grade and recovered ounces represent 99% of predicted mined ounces. These results are outlined in Table 7.2.1. Mine to Mill Call Factors by month is illustrated in **Figure** 7.2.1.

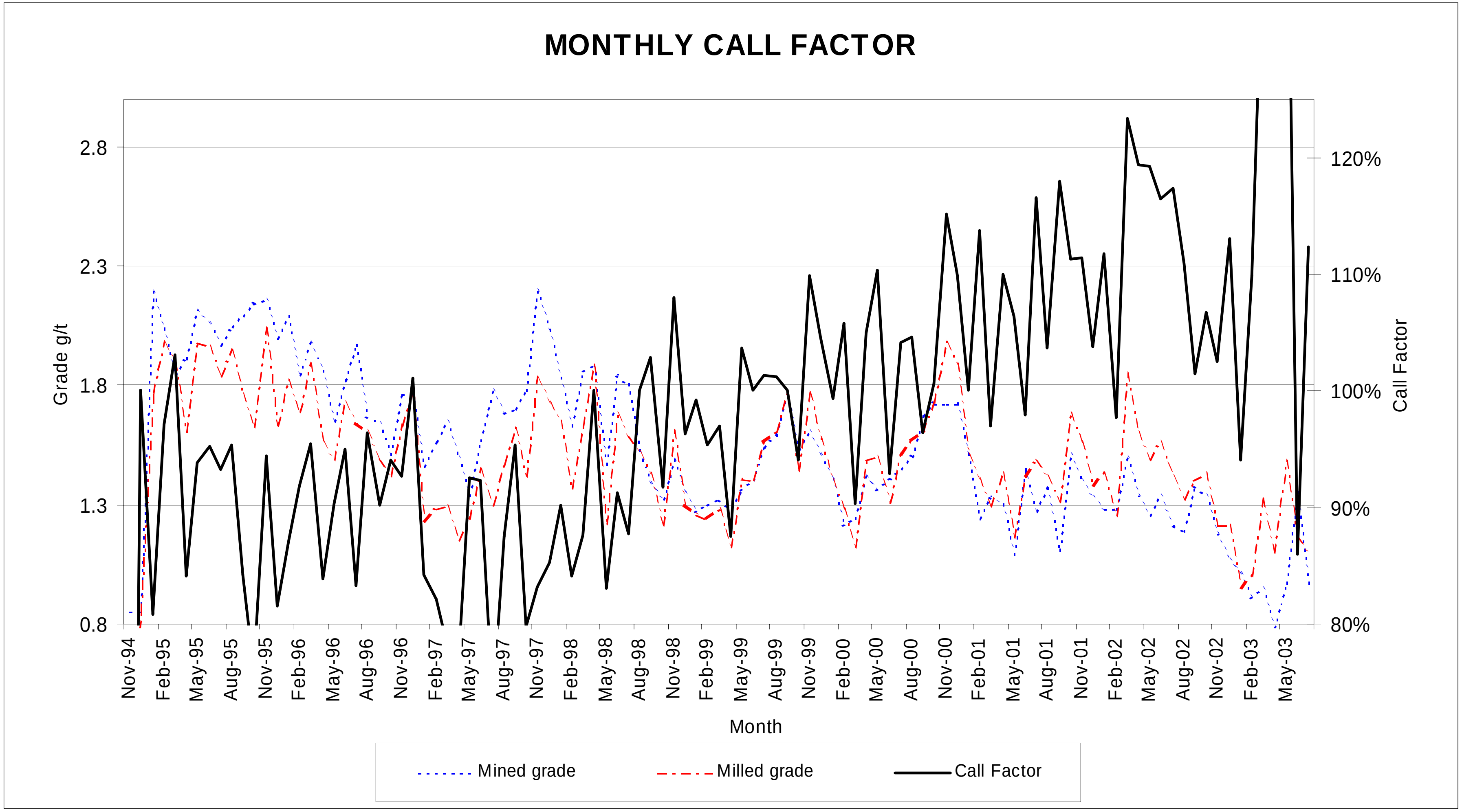
Table 7.2.1: Mine to Mill Reconciliation

Mine to Mill Reconciliation	**Tonnes**	**Grade**	**Ounces**
Ore Mined			
High Grade	16,875,981	1.63	886,812
Low Grade	1,401,633	0.86	38,672
Minerialised Waste Re-handled	2,433,021	0.49	38,028
Total Ore Mined to be Milled	20,710,635	1.45	963,512
Ore Milled			
Total Ore Milled as at end July 2003	19,904,921	1.48	949,039
ROM Stocks Remaining after July 2003	305,195	0.81	7,925
COS Remaining after July 2003	15,244	1.14	559
Total Ore Milled	20,225,360	1.47	957,523
Mill / Mine as Percentage	**98%**	**102%**	**99%**

Table 7.2.2: Ore Mined by Pit

AREA	ORE TYPE	TONNES	GRADE	OUNCES
Alluvials	HG	14,322	1.49	686
	LG	0	0.00	0
	MW	0	0.00	0
	Sub-Total	**14,322**	**1.49**	**686**
Alta	HG	41,091	0.98	1,295
	LG	0	0.00	0
	MW	0	0.00	0
	Sub-Total	**41,091**	**0.98**	**1,295**
Big Tree	HG	55,255	1.52	2,700
	LG	12,922	0.79	328
	MW	0	0.00	0
	Sub-Total	**68,177**	**1.38**	**3,028**
Crosscourse	HG	12,929,103	1.70	706,677
	LG	1,202,858	0.87	33,645
	MW	2,823,793	0.53	48,117
	Sub-Total	**16,955,754**	**1.45**	**788,440**
Crosscourse South	HG	620,935	1.40	27,949
	LG	7,139	0.69	158
	MW	85,475	0.62	1,704
	Sub-Total	**713,549**	**1.30**	**29,811**
Dam A	HG	131,593	1.57	6,642
	LG	3,931	0.73	92
	MW	0	0.00	0
	Sub-Total	**135,524**	**1.55**	**6,735**
Lady Alice	HG	720,808	1.58	36,616
	LG	27,983	0.84	756
	MW	244,030	0.59	4,629
	Sub-Total	**992,821**	**1.32**	**42,000**
Millars	HG	129,690	1.57	6,546
	LG	22,509	0.83	601
	MW	0	0.00	0
	Sub-Total	**152,199**	**1.46**	**7,147**
Prospecting Claim	HG	400,146	1.65	21,201
	LG	24,499	0.64	504
	MW	19,241	0.58	359
	Sub-Total	**443,886**	**1.55**	**22,064**
Ping Ques South	HG	127,592	1.36	5,579
	LG	10,049	0.84	271
	MW	0	0.00	0
	Sub-Total	**137,641**	**1.32**	**5,850**
Temple	HG	12,224	1.02	401
	LG	11,847	0.68	259
	MW	0	0.00	0
	Sub-Total	**24,071**	**0.85**	**660**
Union North	HG	1,498,498	1.30	62,631
	LG	62,042	0.85	1,695
	MW	455,068	0.47	6,876
	Sub-Total	**2,015,608**	**1.10**	**71,203**
Union South	HG	194,724	1.26	7,888
	LG	15,854	0.71	362
	MW	0	0.00	0
	Sub-Total	**210,578**	**1.22**	**8,250**
Total	HG	16,875,981	1.63	886,812
	LG	1,401,633	0.86	38,672
	MW	3,627,607	0.53	61,685
	Grand Total	**21,905,221**	**1.40**	**987,169**

Figure 7.2.1: Mine to Mill Call Factor by Month



7.3 Remaining Potential - By Pit

7.3.1 Prospect Claim

Prospect Claim pit has the best potential for additional open pit mining. An Identified Resource of 250,000t @ 2.73g/t Au currently reported with a strip ratio of 11:1. The resource is contained within a $700/oz gold optimized pit run on the ore block model PC_jun03.mdl. The optimisations were run on the model with the surface mined out with the July 2003 end of month survey pick-up.

Additional factors are; -

- Positive mill to grade control/ model reconciliation, good grades +2g/t
- Good continuity exists of the main zone in both strike (+400m) and dip.
- Secondary ore zones to the east more poddy in nature within a relatively continuous low-grade mineralized zone. The high-grade pods are more discrete and steeply plunging shoots.
- From mining the pit it is evident that the historical mining chased both the main and the more discrete zones to the east.
- In mining the good-bye cut up the ramp the main ore zone was a continuous quartz vein of up to 4m wide. The early miners targeted this zone and some of the old stopes are evident below the good-bye cut.

The negative aspects in deepening the pit would be:-

- High strip ratio
- West wall stability

7.3.2 Crosscourse

Any potential to deepen or extend the Crosscourse pit has been effectively sterilized by tailing into the pit. E Lens which plunges north into the wall and below the pit floor averaged 4g/t Au when mining was complete. Exploration drilling indicates that it continues down plunge. Some of the better intersections include; - URD125 11m @ 6.6g/tAu, URD66911 36m @ 4.1g/t, URD66902 11m @ 4.1g/t, URP66903 20m @ 3g/t, CE0629 16m @ 6g/t, CE0632 15m @ 3g/t. In addition to the tailings a major slip in the north west pit "The Glacier" compounds the problem of mining.

Other areas of remnant ore; -

- Northeast section of the pit which was rat-holed from surface. Ore basted to 10m from surface and only 5 meters mined as good-bye cuts. +12,000t @ 1.5g/t remains in 3rd (~50% remaining) & 4th flitch. Grade drilling indicates ore continues with depth and there is a good potential to block out additional easily assessable ore.
- Western Lens; is at the north end of the pit and cuts below the Main Ramp. From grade control drilling and exploration drilling show good vertical continuity. Strike extent of good grades ends just north of Crosscourse. Recent drilling was carried out to test the up dip extent of high-grade intersections in exploration work; URP69501 15m @ 4.7g/t, URP7001 19m @ 10.62 (includes 3m @ 60.7g/t) and down-dip of grade control drilling which identified the surface trend of the zone. Results confirm a strongly mineralized zone with results received of; CK0350 11M @ 2.92g/t, CK0351 11m @ 5.78g/t, CK0352 9M @ 4.0g/t, CK0357 10m @ 1.7g/t, CK0358 9m @ 24.4g/t and 5m @3.34g/t, CK0359 3m @ 4.2g/t, CK0360 7m @ 2.15g/t. May be able to extract portion of the remnant block of ore by open pit means but being in the pit wall and below "The Glacier" may incur higher levels of ore loss and dilution. This area may be more amenable to small scale underground mining.

7.3.3 Crosscourse South

With the main ramp failure and the west wall wedge failure and the high strip ratio to cut back there is no further open pit potential below Crosscourse South. Approximately 20,000t @ 1.5g/t of blasted ore on the final bench remains buried below the ramp failure buried. Bottom level of in-pit grade control drilling shows the pit stopped on 1.5 – 2.0g/t grades. Sparse exploration drilling information exists below the pit floor. South of the pit exploration drilling shows some broad zones of low grade (+1 – 1.5g/t) mineralisation. Some remnant ore in the high wall between Crosscourse and Crosscourse South pits but would be difficult to extract. 40,500t @ 1.4g/t blocked out from the ramp surface to the 1110RL.

7.3.4 Union South Middle and North pits

Mined as two pits below the 1185RL. Broad low grade mineralized shear zone with two higher grade cores zones, which were targeted by the pits. The higher gold grade zones within the mineralized shear have a high percentage of galena and to lessor extent sphalerite, with high associated silver values ranging from +5 to 30g/t.

Remaining Potential

The model was updated with all the grade drilling results and mined to the current surface. Pit optimizations carried out at $700/oz gold indicate marginal tonnage's to occur beneath both pits. Excluding the remnant ore in the pit walls which would incur high dilution/ ore loss when mining there is approximately 14,000t at 2.0g/t Au for 900 ounce gold is beneath the 1150RL of the Middle pit. Below the North pit 33,000t at 1.4g/t Au for 1,315oz is indicated. These would be high risk tonnes, high strip ratio for little gain. Additional drilling of the high-grade zones below both pits may increase ore tonnes and confidence level to warrant future mining.

7.3.5 Temple

Remaining Potential

Was a small pit i.e. a gouge that targeted a near surface mineralized zone that graded just over a gram. The southern end of the Union South mineralized shear. From mining it appears that a blow out of the ore zone is associated with two intersecting mineralized shears, the axis plunging steeply north. Drilling along strike and down dip indicate a narrow low-grade mineralized shear with associated quartz veining and stockworks. Temple is at the southern extent of the Union line of Lode. No additional potential exists, low grade and generally a narrow mineralized structure.

7.3.6 Lady Alice

The model was updated with the total grade drilling results and the LG was re-run at $700/oz gold. The pit was not extended or pulled down any further.

Extensive historic mining was carried out along the northern strike extent of Lady Alice along both limbs of the anticline. Mining followed along the zones of quartz veining and stockworks. With the deepest shafts/ stopes at the axis of the anticline which shows a thickening of the quartz veining and steeply plunging north. There appear to be a repetition of plunging higher grade shoots along the strike of the system. No grade control or close spaced drilling testing for near surface higher grade zones has been carried out. Review of the exploration drilling along the northern strike show that some significant intersections were hit at depth. There is some scope to build a small resource at surface by testing around / below the areas of higher quartz veining and stoping.

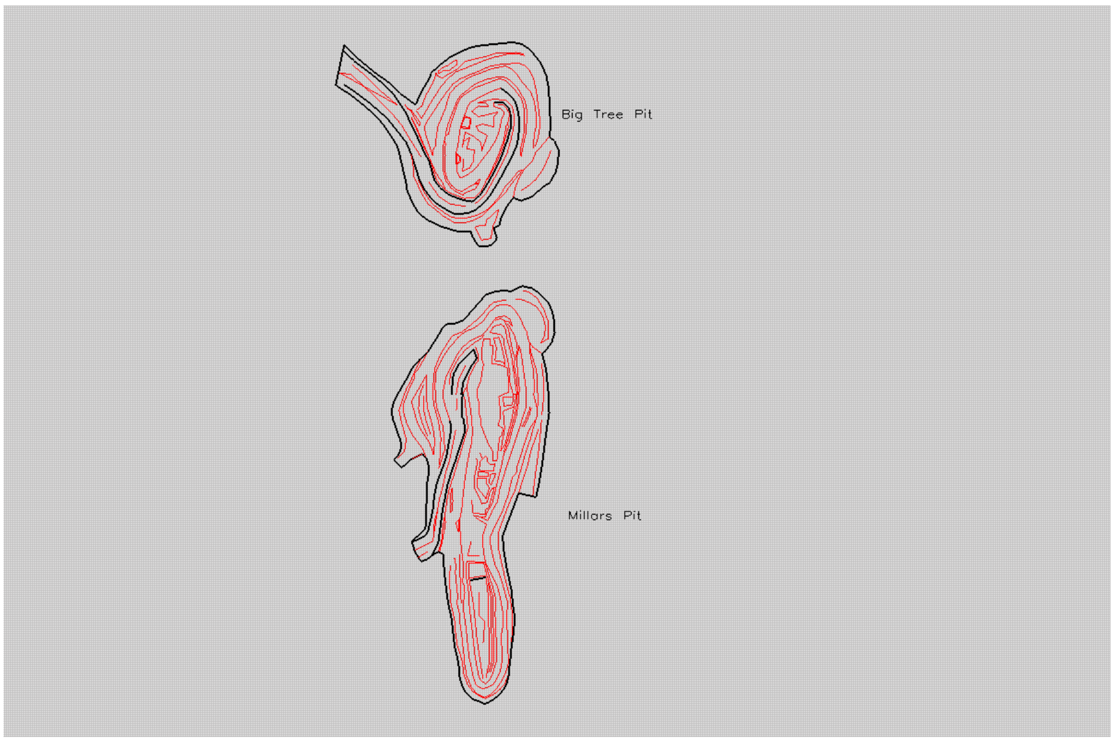
Remaining Potential

- No further economical open pit ore below the present pit floor
- Scope for limited resource along the northern strike extent
- Would require shallow grade control drilling to test tonnes and grade
- The high-grade ore lenses plunging below the pit floor may be developed as a small underground show.

7.3.7 Millars

Millars is a pit developed along the southern most extent of the Lady Alice Line of Lode (Ping Que South mineralized trend).

Figure 7.3.7: Plan of Millars (EOM December 2002) and Big Tree pit (EOM July 2003)



The pit was developed on two higher grade lenses leaving a saddle in the weaker mineralized zone? Old underground stopes were noted in this area which may account for the weak grades. A program of 10 RC exploration holes was drilled under the final Millars pit. Seven holes testing the northern zone, three in the southern zone. This drill program was designed to assess the potential to deepen Millars pit.

The northern extent was mined to final depth of 1159RL, ~ 37m below surface (east wall). The main ore zone is a steeply plunging pod 30m wide by +100m long strongly sheared and altered unit with major quartz veining and stockworks. The entire ore zone was intensely mined by early underground operations that selectively picked out the high-grade ore along quartz vein selvages. Evidence shows that underground mining occurred across the entire 30m width. The material left graded at +2g/t on which the pit was developed. The final floor was still in stope fill with grade of +2g/t. Historical information shows that stopeing continued another +25m below the pit floor and continued along strike through to and under Big Tree pit an additional 200m north. Seven RC drillholes were drilled beneath the pit where access was gained testing for the continuity of the main zone. Two holes drilled along the southern extent of the main zone (Section 5200N; AM0170 8m @ 1.09g/t, Section 5240N; AM0168 4m@ 0.57g/t) confirm down-dip continuity of mineralization though weaker gold values.

Five holes drilled between Sections 5320N to 5360N confirm the down-dip continuity of gold mineralization although indicating narrower zones and overall lower grades. Holes AM0155 8m @ 58g/t including 3m @ 152g/t, AM0154 16m @ 1.1g/t and AM0165 6M @ 3.24g/t. Hole AM0162 (10m @ 5.85g/t) drilled below AM0163 (3m @1.18g/t) on Section 5360N confirms a steep northern plunge of the main zone.

The southern pit extent was mined to a final depth of 1162.5RL, 20m below surface. The ore zone was strongly ferruginous with a lessor percentage of quartz veining and stockworks then the northern extent. The ore zones were narrower on average 3 to 5m with one pod blowing out to 10m mining width. Overall grade was 1.2 to 1.5g/t. Three holes drilled below the main southern lens confirmed similar type grades with depth; AM0171 7m @ 1.5g/t, AM0172 6m @ 1.08g/t, AM0173 9m @ 3.3g/t.

Remaining Potential

> The ore block model (MILJUL03.mdl) was updated based on the latest grade control and deep RC and final mined surface. Pit optimizations at $550 and $700/oz Au were carried using the latest costs and revised pit slope based on the pit mined. No significant tonnage of ore was optimized below the pit as mined. The most limiting factor was the pit slopes, which were flattened if the pit were to be deepened because of the highly unstable nature of the saprolitic oxide waste material.

7.3.8 Big Tree

Big Tree is 50m north along strike of Millars. Same block model as Millars

- The pit was targeted on two parallel ore zones ~15 to 20m apart.
- High grade lenses.
- Extensive workings identified by mining. Part of the historical Millars underground operation.
- Majority of the ore mined was stope fill and remnant quartz veins.
- Major slope stability problems occurred of both east (saprolitic) then the west wall due to structure.

As part of the Millars RC drilling program seven holes were drilled testing the down-dip continuity of mineralization below Big Tree. Three holes were drilled in the unmined bridge between Millars and Big Tree. Results indicated narrow weakly mineralized intercepts. Similar results were received on the four southern sections tested below Big Tree. Grade improvement in the drilling was noted along the northern strike of the mineralized zone. Results from south to north; AM0148 7m @ 1.01g/t, AM0147 3m @ 10g/t, 10m @ 1.27g/t & 8m @ 7.2g/t, AM0146 8m @ 10.2g/t

Remaining Potential

Based on a $550/oz optimized pit around 20,000t @ 2.5g/t for 1,500oz recovered gold may be achievable but at a high risk. Strip Ratio: 12:1. Would require further drilling to confirm.

The ore block model (MILJUL03.mdl) was updated based on the latest grade control and deep RC and final mined surface. Pit optimizations (LG) at $550 and $700/oz Au were carried using the latest costs and revised pit slope based on the pit mined. The LG targeted a single high-grade intersection to the southwest of the present pit. This area would require additional drill to confirm the extent of the high-grade ore. The LG also pushed the pit to the north chasing the high-grade ore identified in the last round of drilling. It would be expected that in mining high ore loss and dilution would be incurred as remnant ore blocks are chased down the pit wall. As with Millars the pit slopes was the most sensitive input in the optimization run. The east wall pit slope was run at 35º due to the highly unstable nature of the saprolitic oxide waste material.

7.3.9 Ping Que South (includes Ayers Rock)

No remaining potential exists. Pit is partially backfilled. Wedge failures occurred along both east and west walls while mining. The pit was mined to design. To deepen would have to flatten wall for safe mining. Deeper drilling does not indicate a likely hood of a significant resource at depth.

7.3.10 Union North

Union North pit was tailed into and capped, sterilizing whatever potential existed. The stability of the west wall, i.e. the Union Fault, was the key factor the pit not being mined to the full design depth and tailing into the pit. Optimizations carried out on the updated block model built including all grade drilling and exploration results confirmed that the pit could have been pushed down further. Union North was not properly assessed before the decision was made to tail into the pit.

The ore zone in Union North is a broad width low grade continuous mineralized zone averaging ~1.5g/t Au. Grades control drilling and deep exploration RC drilling show that this mineralized trend continuing with depth. Strike length +350m, average mining width +13m (10 – 17m); in the upper mining levels mining widths were 12 – 20m.

Remaining Potential

Due to the pit being tailed into there is no additional open pit potential at Union North.

7.3.11 Union North South Extension

Two styles of mineralization were identified by mining / mapping;

- associated with discreet steeply plunging shoots of variable size generally 5 – 15m making mining / grade control difficult
- continuous ore zones with strike lengths of +50m average width 6m with higher grade quartz shoots within the zone. Zone is altered, sericitized, and ferruginous with narrow quartz veining, quartz stockworks and thickened veins.

Overall mining was very selective and required increased level of geological supervision to minimize dilution. Closer spaced grade control would have made ore pod delineation more accurate again minimizing dilution.

The ore block model was updated incorporating the latest grade drilling results. Optimizations run at $700/oz gold indicated a small resource of 2000 oz.

Remaining Potential

> Due to its close proximity to Union North any resources would be sterilized due to in-pit tailings.

7.3.12 Alta

Mining was stopped due to poor reconciliation between grade control and mill. Grade control and exploration drilling indicates a strongly mineralized system averaging 0.8 to 0.9g/t Au with patchy/poddy higher grades zones. From a quick review of the sections it appears there may be two controlling structures of the mineralization. A dominant north – south trending structure dipping steeply east which crosscuts an earlier? Shallow east dipping / shallow north plunging structure. Requires review. Exploration drilling indicates continuity down dip and along the northern strike.

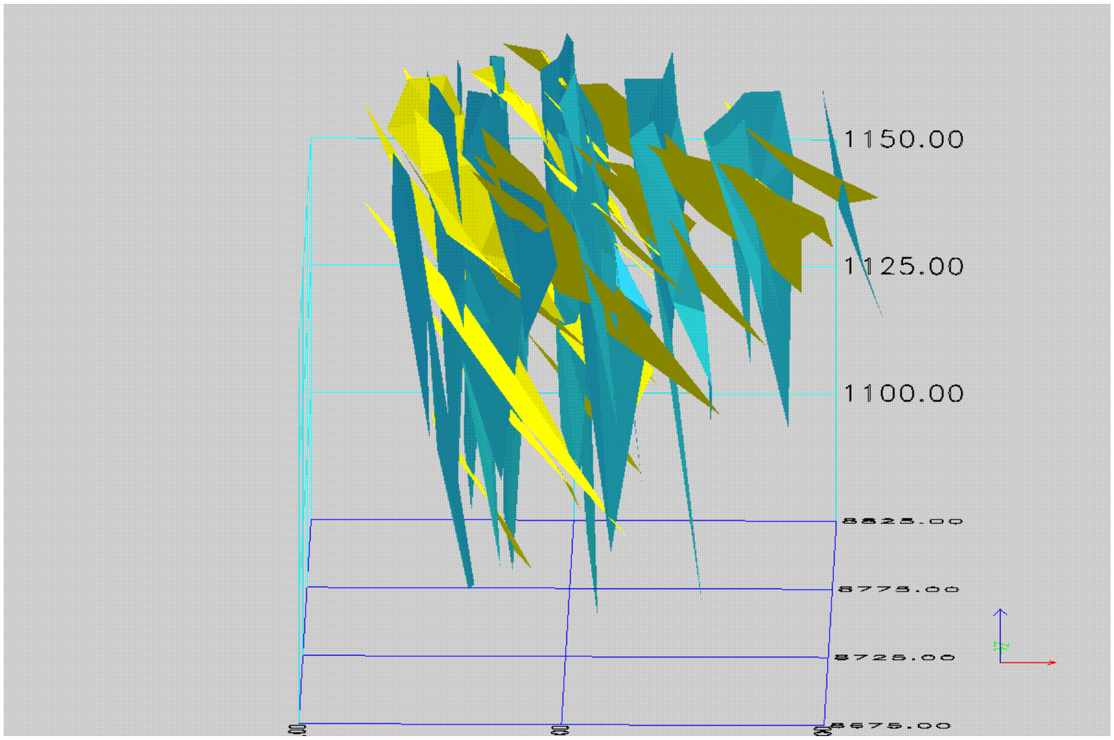


Figure 7.3.12: 3D view of the interpreted Alta mineralized structures.

View is looking down from the south. Blue wire frame steep east dipping north – south structures. Yellow/ ochre shows the flat east dipping structure Interpreted from the gold grade.

7.3.13 Dam A

Remaining Potential

Dam A model was updated during April 2003 with the latest grade drilling information included. Optimizations were run at $560 and $700 per ounce gold. A limited resource exists below the pit floor (good-bye cut). Remnants of ore remain in the pit walls but excessive ore loss and dilution incurred during mining would sterilize these pods. Review of the model shows that additional drilling would not greatly increase the recoverable tonnes.

Recommendations are the no further work is carried out.

7.4 Other Prospects

7.4.1 Wellington

The northern most significant mineralization delineated along the Union Line of Lode. Extensive old workings indicating narrow steeply dipping/ vertical zones. From drilling the shoots appear to be steeply plunging north with a short strike extent ~25m (2 sections). Two zones of three lenses each have been interpreted from drilling centering on 9500N / 5160E and 9650N – 9675N / 5160E. Best intercepts;- on section 9500N, WNP95503 4m @ 2.6g/t Au from 32m. On section 9650N, WNP96501 4m @ 22.5g/t from 20m, 4m @ 16.3g/t from 54m; section 9675N, WNP96752 5m @ 22.3g/t from 82m which lies down plunge of the 22.5g/t intercept on Sect 9650N.

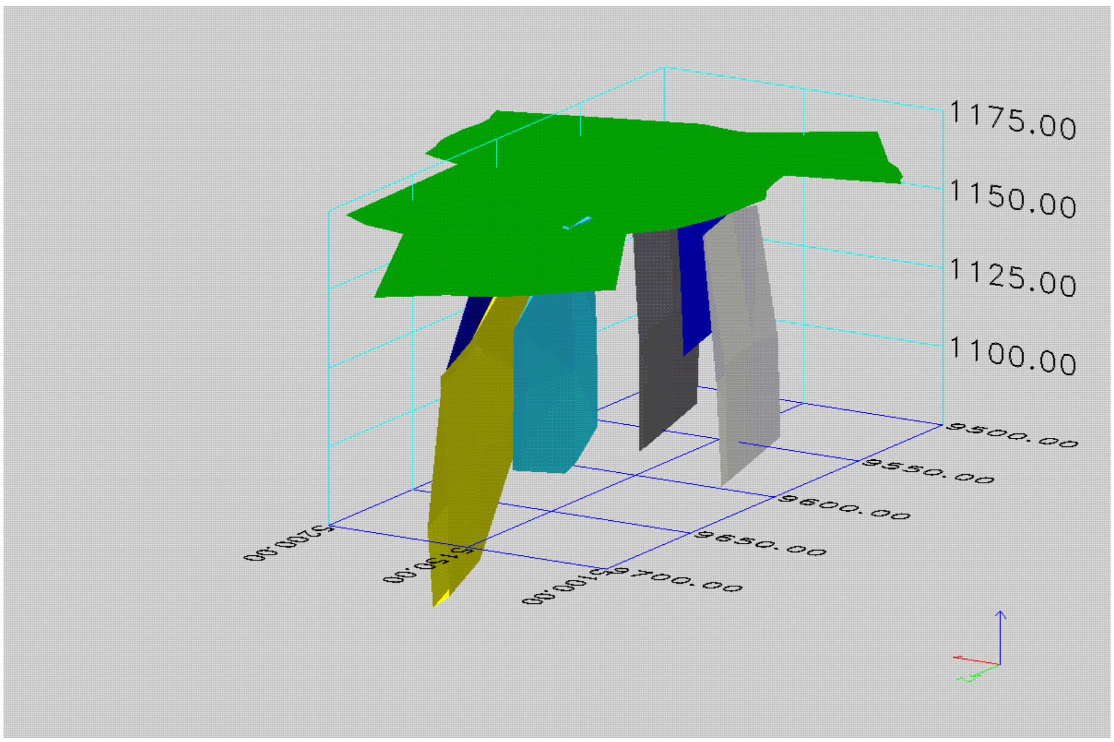


Figure7.4.1: Interpreted Wellington ore pods.

View is looking down from the northwest. The yellow pod contains the best grades.

Wellington may be more suited to a small underground show. For open pit mining the high strip ratio would increase the risk factor of achieving the ounces. Also increasing the risk is that Wellington has been extensive mining from surface. Additional drilling would assist in delineating the ore zones and may increase the resource at depth.

7.4.2 Orinoco

A low-grade prospect, contains no viable resource. Best zone lies between 9100N and 9150N. Lies between Alta to the south and Wellington to the north. Developed on a flexure along the Union line of Lode. 250m strike length from 9050N to 9300N. Dip rolls along strike from steep east dipping in the south changing to moderate west dipping along the northern strike.

An ore block model was built and optimized at $700/oz gold. The optimized pit shell contained 60,000t @ 1.25g/t Au for 2,400oz. The strip ratio was 4.44:1.

The strike extent between 8900N (north of Alta) and 9050N may warrant addition work i.e. detail surface grade control drilling. A small surface resource may be blocked out.

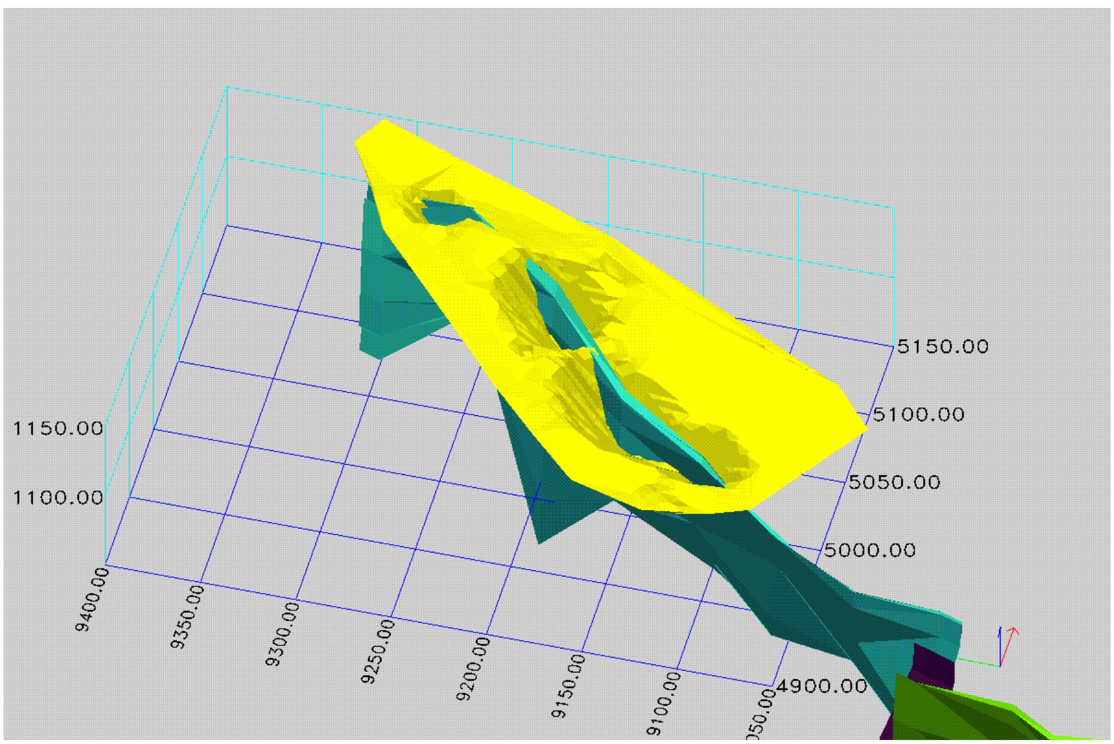


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Figure 7.4.2: Optimized $700/oz pit shell, showing wireframed ore zones at Orinoco

7.4.3 Esmeralda

Esmeralda is a small deposit located 7 km. south of the Union Reefs plant. It is a multiple lode system with exploration centering on two principal veins, one (Zone A) characterized by blue, brecciated quartz, the second (Zone B) by disseminated pyrite mineralization in a quartz-carbonate gangue. Cherts and argillites of the Mt. Bonnie Formation host mineralization. The lodes thicken and thin considerably and grade intersections range from 2-12m, downhole. It is expected that additional exploration in the will result in the discovery of additional lodes. Recent preliminary LG optimizations on the Inferred resource model indicate mineable potential of approximately 20,000 oz Au with 241,000t @ 2.10 g/t Au in Zone A and 90,000t @ 1.16 g/t Au for 4600 oz Au in Zone B. Additional in-fill drilling is required to upgrade from the current Inferred Resource to a Indicated/Measured category.

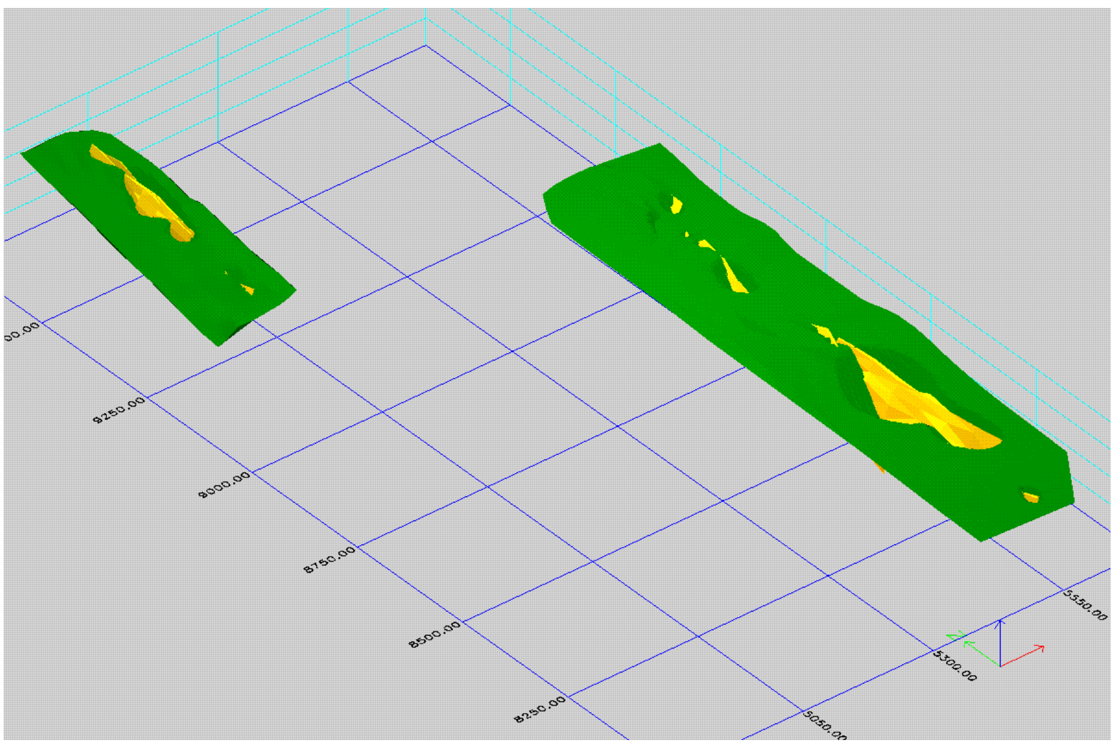


Figure 7.4.3: 3D view of the optimized pit shells at Esmeralda Zone’s A and B

7.4.4 Tobermoray Alluvial

A memorandum dated 8 September 1992 from M Syka to K Hellsten estimates 205,580 lcm @ 0.6 g/lcm of alluvial gold. The full report “Tobermoray Alluvials.pdf is included in the Geology Appendix.

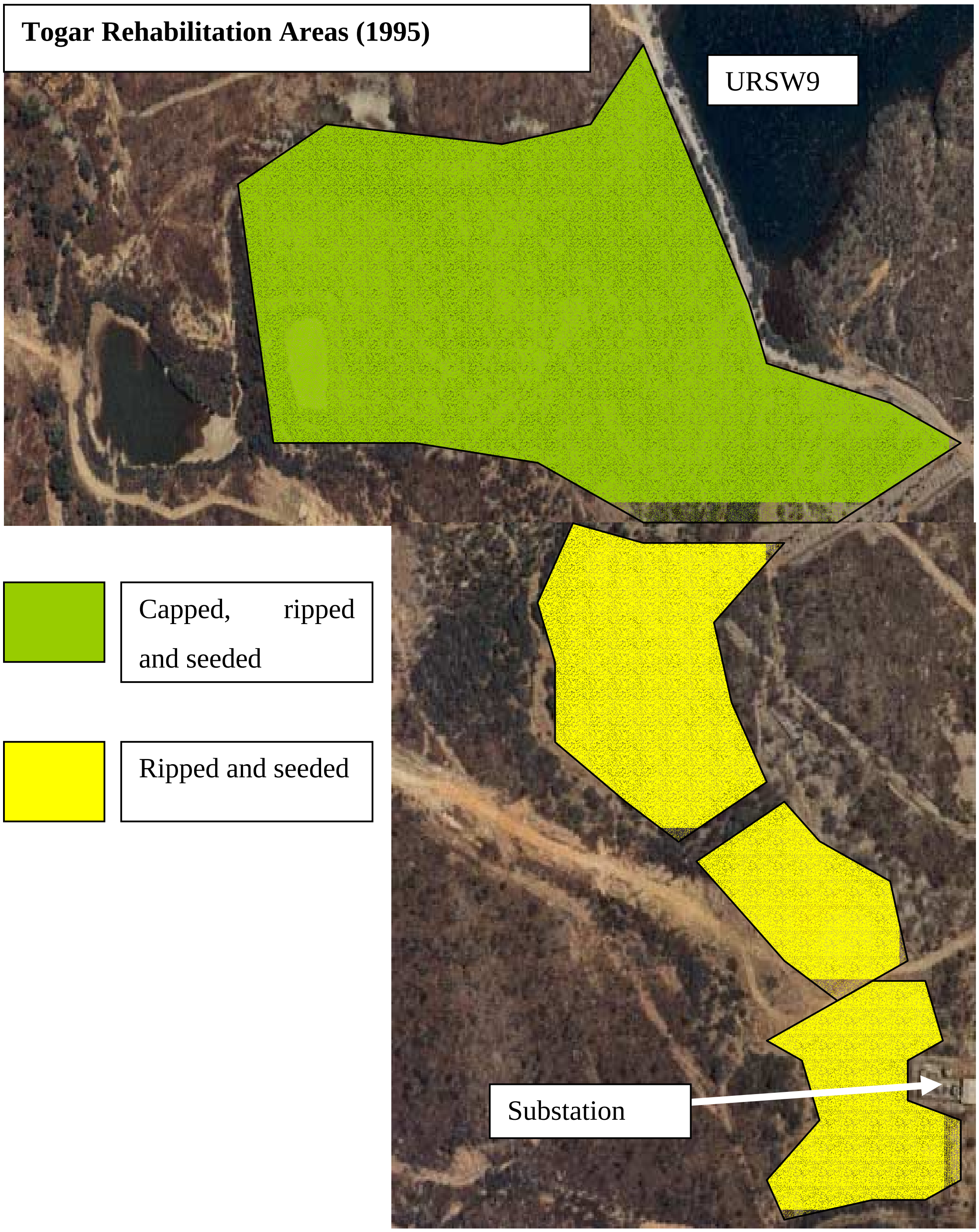
7.4.5 Low Grade in East Waste Dump

Approximately 250,000t of low-grade mineralization with an estimated grade of 0.75g/t Au have been buried in the East Waste Dump. The estimated grade is based on results of low-grade material treated in the last year of milling. At 93% recovery there is 5,600 ounces gold. The LGM is covered with 583,000lcm of waste.

7.4.6 Togar Tailings

During the late 1980’s to early 1992 an alluvial treatment plant was operating in the vicinity of the mill area. Togar was the operator. Throughput was in the order of 700LCM/shift or 1,400LCM/day but only worked days during the wet. (Information derived from Jim Messenger who worked for Togar from 90 to 92). Rough estimates would be that well in excess of 500,000t of tailings were discharge to the west of the mill area below Dam B. Alluvial material was sourced from Dam A area in the north to south of the tailings dam. Knelsons concentrators were the main method of gold recovery. 10 samples taken of the tailings in August 2003 indicate an average grade of 0.57g/t gold.

Figure7.4.6: The aerial photograph shows the extent of the Togar tailings.



7.5 Underground Potential

7.5.1 Prospect Claim

- 144,000T @ 11.7g/t Au modeled within 5 wireframe lenses.
- Along the entire strike length of Prospect Claim between 7090N to 7690N to a depth of 1060RL, 140m below the 1200RL surface.
- The cross-sectional ore pods were interpreted using a lower cut of 2.5g/t Au, minimum of 1m down-hole widths to establish continuity.
- True widths vary up to 4 to 4.5m.
- The x-sectional interps were wire-framed in 3-D joining up the related ore lenses.
- Sparse drill information below the 1120RL.

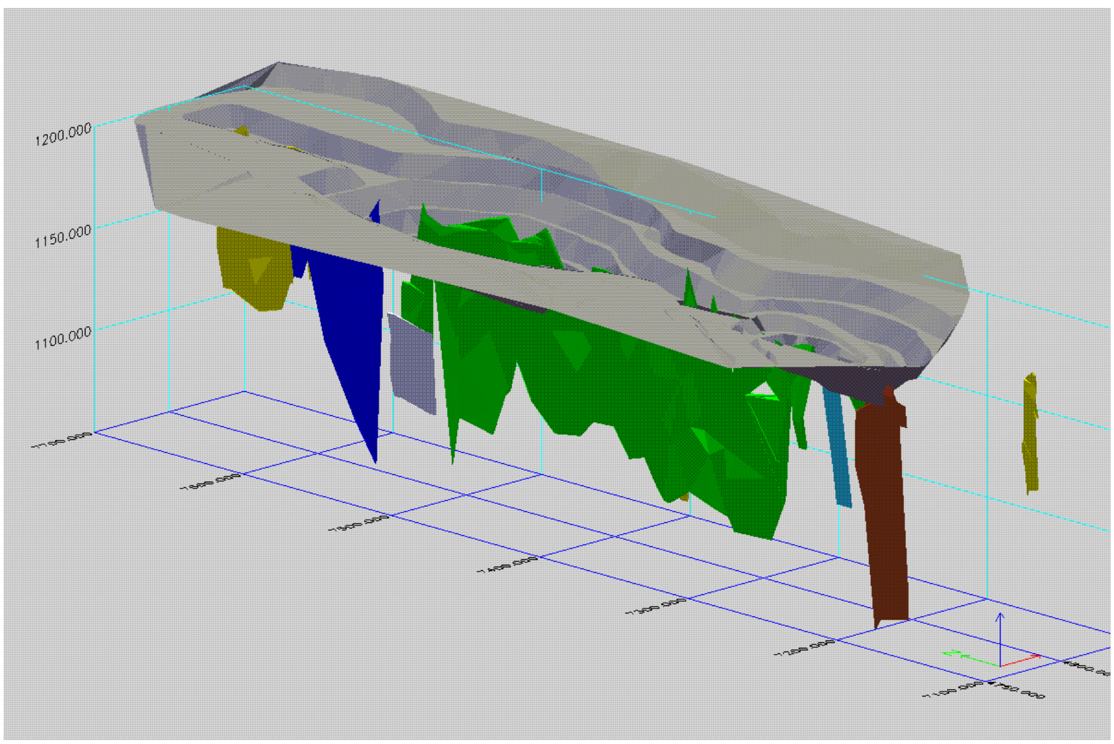


Figure 7.5.1: 3D view of Prospect Claim pit showing the wireframed U/G ore pods.

View looking down from the southwest

Conclusion and recommendations:

- Substantial resource exits based on a first pass modeling. More detail study is required to substantiate.
- Requires more detail drilling at depth
- Open down dip
- Narrow
- Easily accessible ore by underground. The near surface zones may be more suitable by open pit means. Strip Ratio would be high at 11:1.
- Very worth while target

7.5.2 Crosscourse

- Two significant zones identified; E-Lens dilation zone, which plunges north below the pit floor, and the Western Lens in the north pit wall running beneath the main ramp.
- +150kt @ 4.33g/t Au wire-framed with the E-Lens pod.
- +18kt @ 14.56g/t Au wire-framed with the Western Lens.
- Both require further drilling to confirm and fully test.
- The E Lens pod may prove uneconomical as it is below the level of tailings currently in the pit.
- Ping Que lode (Lady Alice Line of Lode); the low-grade mineralization is continuous along strike within which are discrete narrow high-grade steeply plunging shoots, multiple zones,
- Discrete high-grade shoots have been identified within other lenses but due to lack of drilling at depth it is difficult to ascertain the full extent of any resource. Refer to report by Mark Kent dated November 1998, “Gold Distribution and Lens Morphology at Union Reefs Gold Mine”. These lenses demonstrate continuity of low-grade mineralization but variable are very poddy at a high-grade (2.5g/t Au) cut-off.
- The southern extension of the E-Lens dilation zone centering 6550N – 6450N/ 4900E, 2 main lenses interpreted, based mainly on grade control data with some good exploration intercepts at depth; 6m @ 9g/t, 3m @ 17g/t, widths up to 5 meters. Are below pit / tails level.

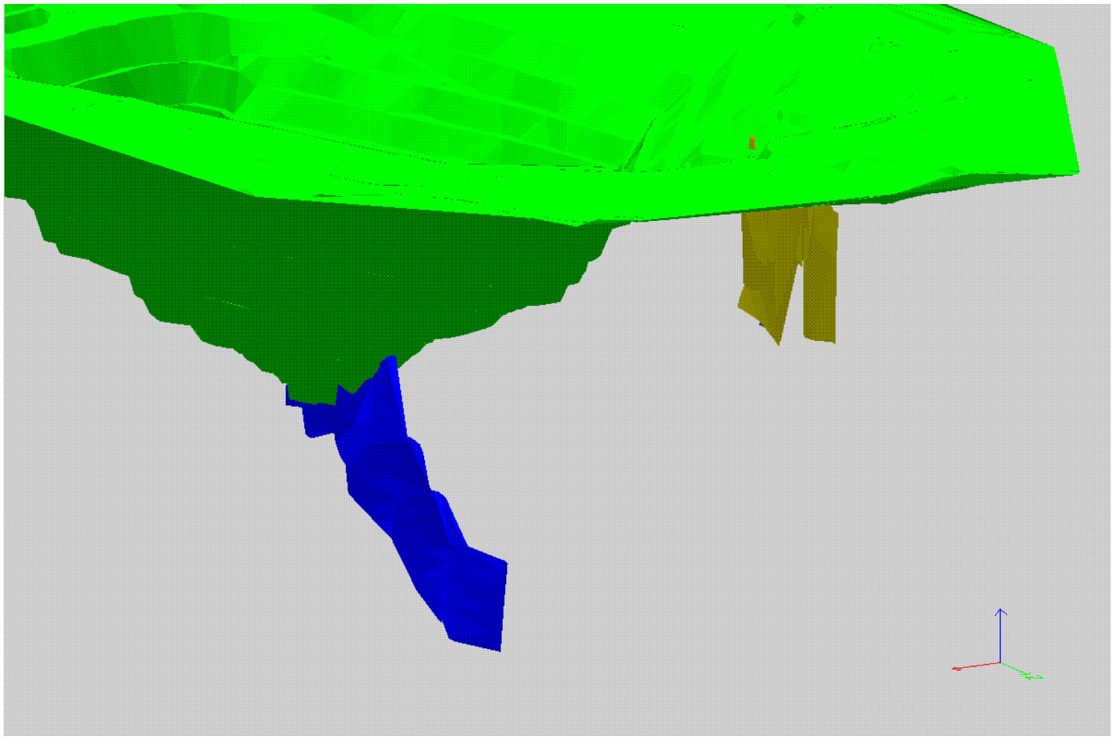


Figure7.5.2: 3D view showing E Lens (blue) and Western Lens (yellow).

Projecting below Crosscourse pit. View looking down from the northeast.

7.5.3 Crosscourse South

There is insufficient deep drilling below Crosscourse South to be able to assess underground potential. Best intersections below the pit include; 8m @ 6g/t Au, 3m @ 4.7g/t, 2m @ 7g/t. Any underground operations would also be affected by the in-pit tailings in Crosscourse further lowering prospectivity.

7.5.4 Millars & Big Tree

- Up to 40,000t at +16g/t Au in 13 pods along a 500m strike length from Big Tree to Millars. 60m high panel from the base of the pits
- Wireframes interpreted using a lower cut-off of 2.5g/t Au, minimum intercept length of 1m. No top cut applied.
- 6 lens interpreted at Big Tree, 7 at Millars. The high-grade zones are narrow average width being <2m (from <1m to 3m). Tonnes per vertical meter (t/vm) are low; Big Tree average 420t, Millars 370t. Pod grades vary from 2.7g/t to 40g/t.
- Developed along two parallel zones with the east zone more consistent.
- Some significant intersections include; 2m @220g/t, 5m @ 17.3g/t, 3m @ 16.5g/t, 2m @ 7.1g/t at Millars and 4m @ 44g/t, 4m @ 18.8g/t, 4m @ 13.1g/t, 5m 5.9g/t, 3m @ 7.8g/t, 2m @ 10.7g/t at Big Tree.

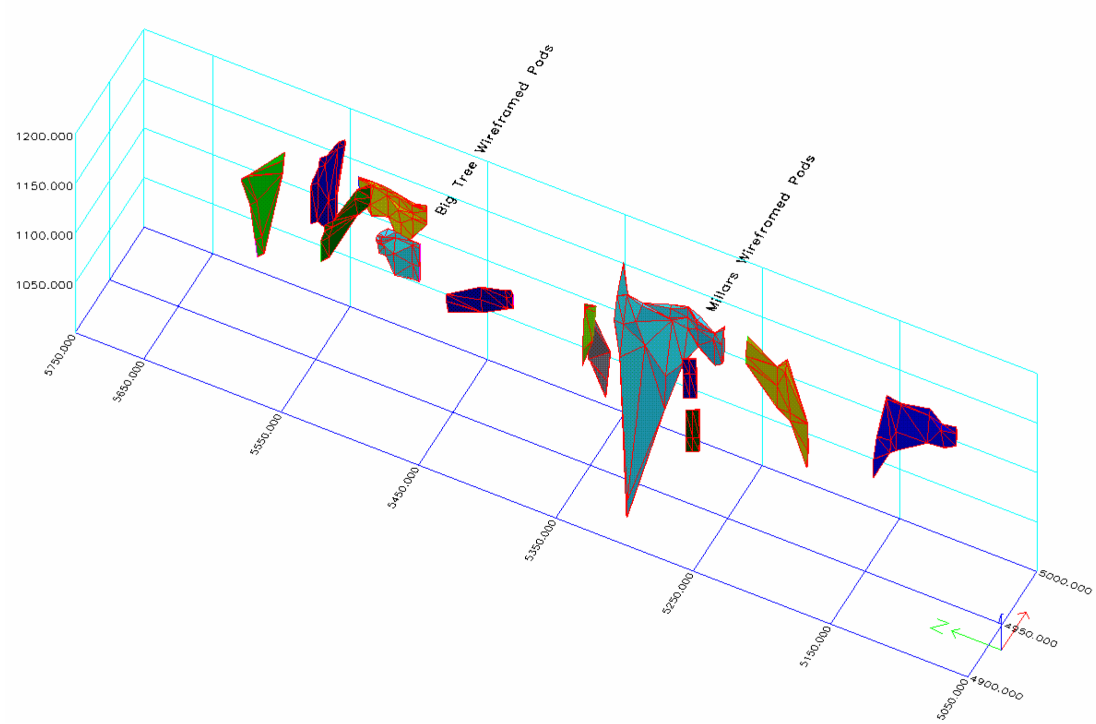


Figure 7.5.4(a): 3D view showing the wireframed 2.5g/t Au lens in the Millars/Big Tree pit. View looking down from the southwest.

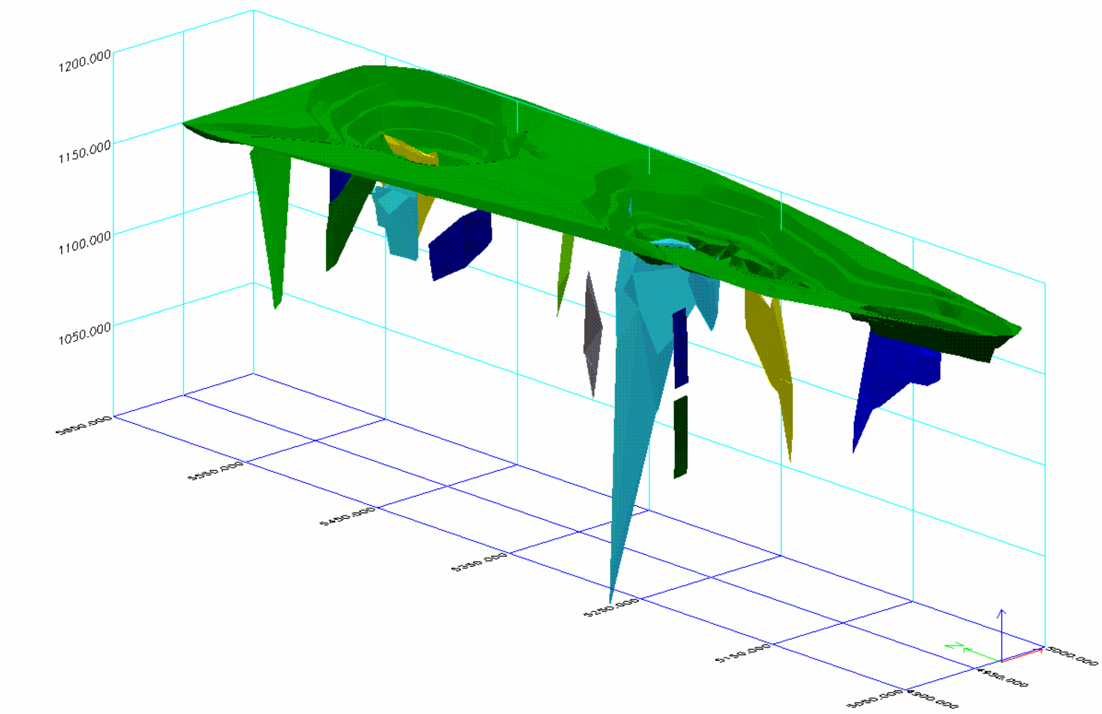


Figure 7.5.4(b): 3D view showing the wireframed lens in relation to the final surfaces of Big Tree and Millars pits. View looking down from the southwest

Conclusion and Recommendations;

- Series of small high-grade lens of limited strike extent but some are open down dip/ plunge.
- Narrow with low t/vm.
- Would require a highly selective mining method to minimize dilution.
- More drilling is required to confirm interpretations and tonnes. Access for drill sites would be difficult due to pits and extensive wall failures.
- Due to the limited tonnage no further work is recommended.

7.5.5 Union South

Main u/g resource below Union South Middle pit, 20,000t @ 12g/t (400t/vm).

- Based on grade drill and limited exploration drilling. The ore lens is the down dip extension of the high-grade ore zone mined in the Middle pit.
- Average width 3.5m.
- Characteristics are highly sheared and sericitized with quartz veining and stockworks, generally ferruginous and vuggy in the oxide horizon grading to increased fresh sulfides with depth, galena, pyrite, sphalerite also high silver content +10g/t in galena rich veins. Strongly disseminated throughout the altered shear zone with massive sulfide veins up to 0.5m in width.

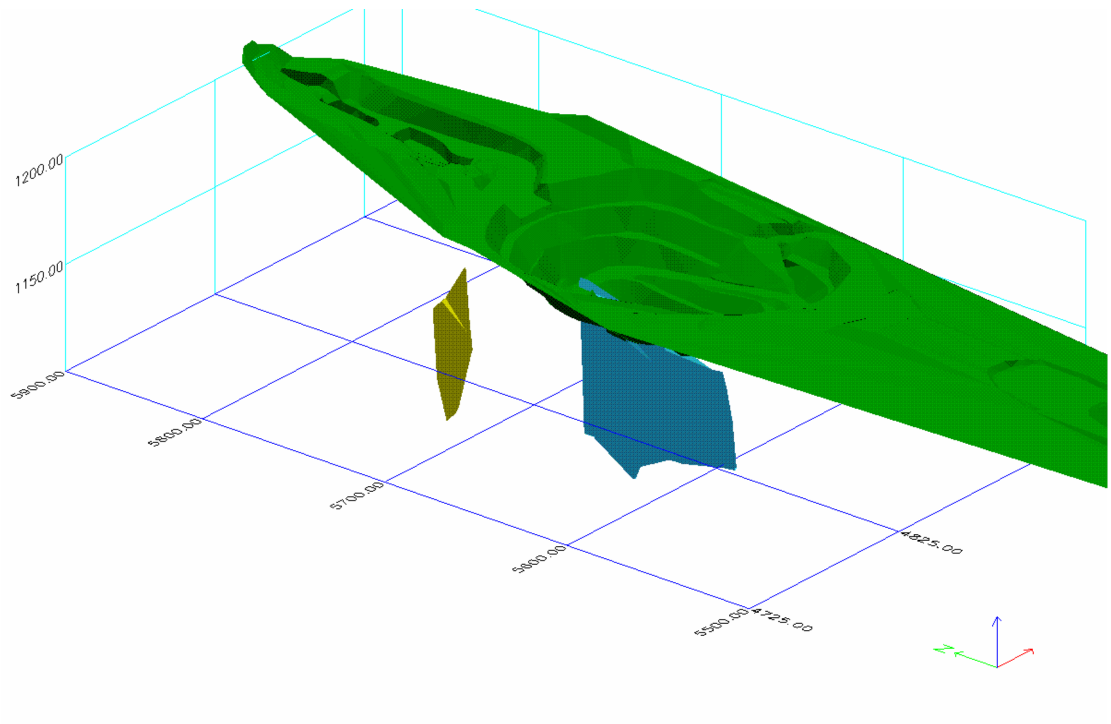


Figure7.5.5: 3D view of Union South showing the interpreted ore lens.

View looking down from the southwest.

A second zone occurs 50m below the North pit high-grade core zone mined out.

Conclusion and recommendations

- Although low t/vm the main pod below the Middle pit would be easily accessible.
- Would not provide the bulk tonnes to sustain an operation but may provide a grade sweetener.

7.5.6 Lady Alice

Review of a potential underground resource carried out indicated +40,000t @ 6.9g/t Au for 9,500oz gold is likely below the 1157.5RL of Lady Alice pit. This would include ore in the crown pillar. A total of seven wire-framed pods were interpreted from the total grade control and exploration RC drilling results at a lower cut-off of 2.5g/t Au.

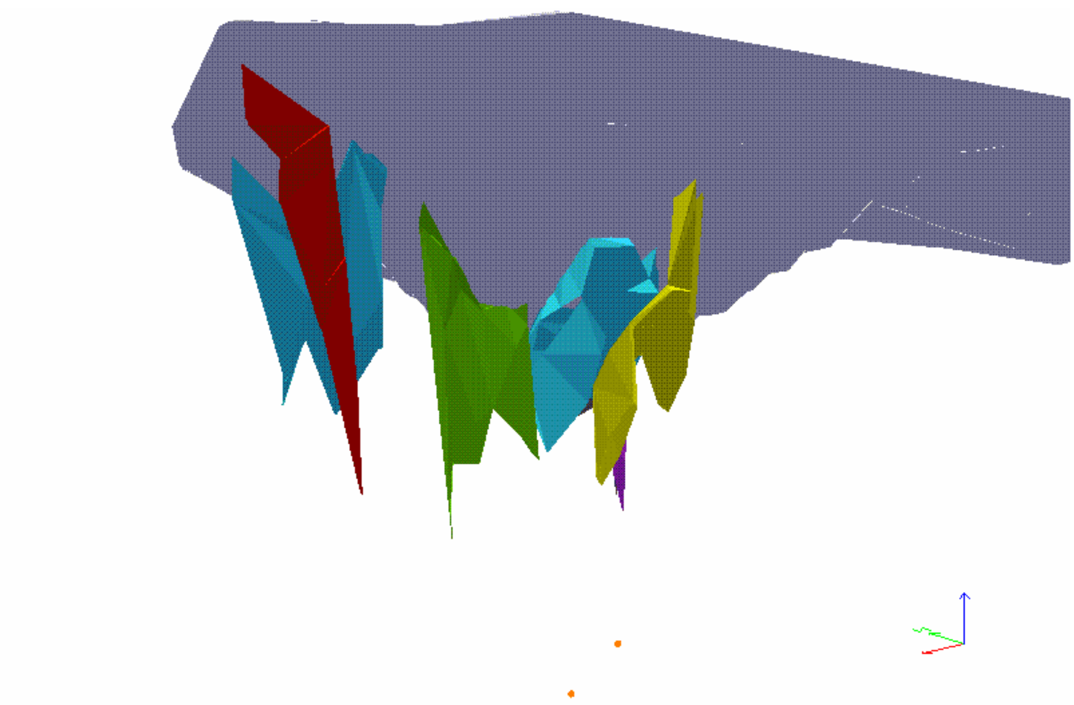


Figure 7.5.6(a): 3D view showing the interpreted ore lenses below Lady Alice pit. Looking up from the southeast.

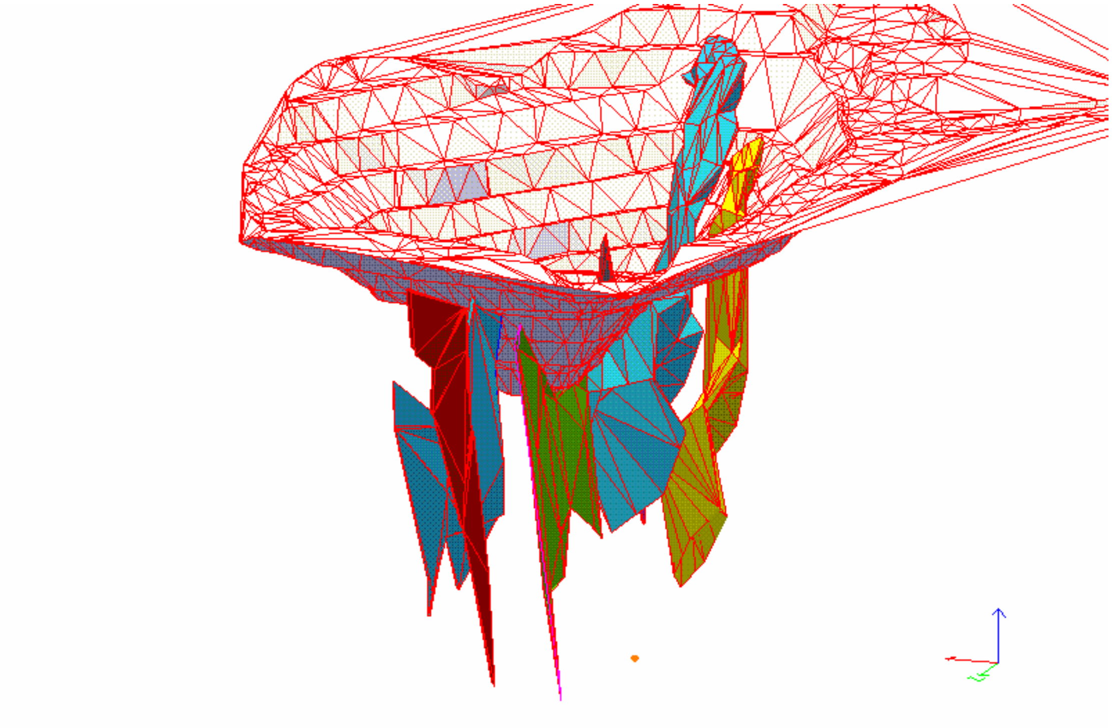


Figure 7.5.6(b):3D view showing the interpreted ore pods below Lady Alice pit.

Looking down from the Southwest.

7.6 Recommendation

- More detailed review of the block model, initial block modeling that was carried out was to assess likely potential
- Further drilling may be warranted to test the strike and down-dip/ plunge extents. The higher grade thickened quartz rich veins/ lenses plunge steeply north.
- Significant high-grade intersections were received in the exploration drilling along the northern strike extent of Lady Alice pit. These were not incorporated within this study, as there was insufficient drilling to wireframe shapes. Further drilling may develop additional resources or establish addition open pit tonnes.

7.6.1 Dam A

A geological resource of 2,200t at 6.59g/t Au for 470 ounces exists down-dip of the main ore zone of Dam A. A model was built using an interpreted lower cut-off of 2.5g/t Au. Two major parallel structures were identified. The eastern most was mined out. The western is located beneath the good-bye cut of the main pit.

No further work is warranted, as there is not enough upside to be able to generate the tonnes to develop a substantial underground operation. Water would also be a major problem.

7.6.2 Union North and Union North Southern Extension

Review of the drilling data below both areas does not indicate any likely stopeing blocks of high-grade ore. Isolated and random +2.5g/t gold intercepts occur along the strike extent of Union North. Due to tailings pumped into Union North any potential that may have existed for underground mining has been sterilized.

7.6.3 Wellington

A small underground operation targeting a small high-grade zone centering on 9650N – 9675N / 5160E. Intersections include;- 4m @ 22.5g/t, 4m @ 16.3g/t, 5m @ 22.3g/t gold.

REFERENCES

Kent M., November 1998, Gold Distribution and Lens Morphology at Union Reefs Mine. (Unpublished Company report).

Makar, B., Chen Chow, K., September 2003, Mineral Resource Estimation For The Union Reefs Gold Deposit. (Unpublished Company report).

APPENDICES

Geology – (Cut to CD under Appendices/Geology/"Folder")

- 2003 Resource Statement
- Digital Topographies
- Exploration Database
- Final Block Modeling MineMap Work Files
- Final Pit Surveys
- Grade Control Database
- Mine Lease Exploration
- QA QC
- Regolith Profiles
- Site Plans
- SOP's
- Tobermoray Alluvial

APPENDIX 2

AngloGold Announcement
(14 November 2003)

anglogold

Announcement

(Incorporated in the Republic of South Africa)
(Registration Number: 1944/017354/06)
ISIN Number:ZAE000014601

ASX Code: AGG
JSE Code: ANG
NYSE Ticker: AU

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14 November 2003

AngloGold reaches agreement to sell Union Reefs gold mine

AngloGold Australia Ltd ("AGG") advises it has today entered into an agreement with Greater Pacific Gold Ltd ("GPG") to sell its gold mining assets. These comprise the Union Reefs Gold Mine ("URGM") at Pine Creek and associated assets and tenements. The agreed staged purchase consideration for these assets is AUD$6.2 million.

The staged consideration comprises:

- AUD$0.2 Million non-refundable cash deposit payable immediately, and
- AUD$2.0 Million cash paid on or before 27 February 2004, and
- AUD$4.0 Million cash paid on or before 30 June 2004.

The sale is dependent upon GPG meeting the staged payments schedule and various other AGG related performance criteria. The transaction is conditional upon a satisfactory due diligence outcome, the attainment of all regulatory approvals (ASX and NT Mines Department), shareholder approval and securing requisite financing arrangements.

AngloGold Australia Limited

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CAUTIONARY STATEMENT CONCERNING FORWARD-LOOKING STATEMENTS

Certain statements in this announcement are forward-looking within the meaning of Section 27A of the Securities Act of 1933, as amended, and Section 21E of the Securities Exchange Act of 1934, as amended, including without limitation, those statements concerning (i) timing, fulfillment of conditions, tax treatment and completion of the Merger, (ii) the value of the transaction consideration, (iii) expectations regarding production and cost savings at the combined group's operations and its operating and financial performance and (iv) synergies and other benefits anticipated from the Merger. Although AngloGold and Ashanti believe that the expectations reflected in such forward-looking statements are reasonable, no assurance can be given that such expectations will prove to have been correct.

For a discussion of important terms of the Merger and important factors and risks involved in the companies' businesses, which could cause the combined group's actual operating and financial results to differ materially from such forward-looking statements, refer to AngloGold's and Ashanti's filings with the US Securities and Exchange Commission (the "SEC"), including AngloGold's annual report on Form 20-F for the year ended 31 December 2002, filed with the SEC on 7 April 2003 and Ashanti's annual report on Form 20-F for the year ended 31 December 2002, filed with the SEC on 17 June 2003 and any other documents in respect of the Merger that are furnished to the SEC by AngloGold or Ashanti under cover of Form 6-K.