COPPER CLAYS SCOPING STUDY

COPPER MINES OF TASMANIA PTY LTD

AMC Consultants Pty Ltd

ABN 58 008 129 164

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AMC 107122

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EXECUTIVE SUMMARY

AMC was requested by Copper Mines of Tasmania Pty Ltd (CMT) to prepare a scoping study for the King Lyell Copper Clays. The study aimed to identify whether the potential of the deposit justifies spending the money required to gather further data to proceed with a Feasibility Study process.

While the project still has a very high level of uncertainty, AMC believes there is a justification to proceed with the work required to take the project to the next level of evaluation, namely, a Pre-feasibility study.

The Copper Clays project has the potential to provide value to CMT. The analysis suggests a pre-tax NPV at an 8% discount rate ($NPV_{8\%}$) of around A$10M to A$20M for the base case copper price of US$6,700 /t, should prices remain at the present level of US$8,000 /t the project will generate a pre-tax NPV of A$20M to A$30M. This is based on a pit containing approximately 900,000t of ore at a grade of 1.9% Cu and a mass based strip ratio of 6:1.

Presently there is a very high level of uncertainty as there is limited geological, geotechnical and metallurgical understanding of the deposit. This would need to be addressed before the project could commence. As the geographical extent of the project is limited, the total cost of addressing these issues should not be high.

The Pre-feasibility study should address the various mining and processing options, selecting the best for final analysis and costing in a Feasibility Study that would then form a basis for the investment decision. The cost of the final Feasibility Study would depend on the results from the Pre-feasibility study and the level of surety required by CMT prior to proceeding with the project.

As part of the Feasibility Study, plans and protocols would need to be developed to manage the risk presented by the water in the Iron Blow open pit.

Taking the project to feasibility decision point is likely to take two years with a cost of approximately A$4M. This will need to be confirmed as the project progresses, particularly the cost of the feasibility study which is dependant on the outcome of the pre-feasibility study.

The first major milestone is the completion of drilling and geological interpretation and modelling. The scoping study should be revisited at that stage to confirm there is value in moving to pre-feasibility.

Risk Analysis

The major risks which will require to be addressed before a decision can be made to proceed with the project are:

- **Geological Model** – The geological model is based on poor quality data. Further drilling and analysis is required to bring it to a JORC Indicated or measured standard.

- **Iron Blow Water** – The water presents a danger of sudden inrush. A plan is required to remove the water and ensure the pit remains dry for the period of mining. This may require significant remedial action.
- **Historic Waste Dumps and Clay Slopes** – Both of these present a significant danger of collapse. The extent of the risk needs to be further evaluated through geotechnical analysis, design and operating protocols developed to minimise the danger.
- **Metallurgical** – There is presently limited metallurgical understanding of the mineralised zones. There is a risk that the material will need crushing to liberate the native copper or the copper may be in a form which cannot be separated by gravity. A well structured metallurgical test program will be required to address this issue.

There are a number of other waste dumping, environmental and heritage risks which need to be addressed prior to mining. While these are important they do not have the similar potential to lead to failure of the project as the issues listed above.

Geological Review

The geological review of the King Lyell Copper Clays deposit included a preliminary check of the existing geological database; preparation of a geological block model and identifying any additional geological work required to conduct a pre-feasibility study.

AMC produced two models, a conservative model and an optimistic Upside model of the copper clays. The Upside model has been used as a means of identifying further potential of the deposit, hence it has been used for the scoping study to decide if additional expense would justify gathering data to confirm this potential.

The Upside Model for the CMT Copper Clays King Lyell deposit contains 1.7 Mt to 1.9 Mt at 1.4% to 1.6% copper at a cut off grade of 0.7% copper.

These results are not classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2004 edition (JORC Code) due to insufficient confidence in the data on which they are based.

There are a number of factors that affect confidence in the data and hence the estimate:

- The drill hole dataset is limited to thirteen holes in the central portion of the model, the mineralised zone was extrapolated up and down dip of the known ore zone.
- No information was available on QA/QC aspects of the sample and assay data sets, and a reproducibility test failed on the 1970/71 assay data (Wills 1995).
- There is no density data available in the current drill hole database.
- Inaccurate topographic surface.
- Lack of recovery and void data.

To bring the estimation to a standard where it can be classified under the JORC code, an infill drilling programme is required. The drilling would be located mainly in the

western extent of the area, requiring a minimum of sixteen cased diamond drill holes on a 50m X 50m grid enabling the validation of the geological interpretation. All mineralisation should be sampled using coring to ensure good recoveries and enable geotechnical evaluations.

Consistent geological logging of future holes will alleviate discrepancies found in the current database. This will enable a good understanding of the geology and mineralisation allowing the construction of geological wireframes that will provide greater confidence in defining the mineralisation.

It is recommended that bulk density tests be carried out on samples from any future drilling. This is particularly important as there may be considerable variability in bulk density depending on the copper grade. There may also be a large variability of bulk density between the ore zone and waste.

In summary, to upgrade the mineral resource and to validate the existing work any future drilling should include but not be restricted to:

- Geological and geotechnical logging.
- Collection of samples for assay sample submission.
- Collection of samples for and calculation of bulk density measurements.
- The inclusion of checks, blanks and standards to provide industry standard QA/QC data.
- Measurement of recovery of core with particular emphasis on the mineralised zone.
- Establishment of the location of existing voids for previous workings in relation to existing and new drilling.
- Validation of new data against existing data to increase confidence in the drill hole database.

Geotechnical

The geotechnical assessment of the King Lyell Copper Clays deposit included a review of data available, assessment of the geotechnical materials for design parameters based on the data available, and recommendations for further investigation and analysis.

The conclusions from the assessment were:

- The six lithologies in the project area can be grouped into three sets of distinguishable geotechnical engineering materials. Each of these materials is expected to differ in behaviour when exposed.
- The inter-ramp slope angles that should be used in preliminary pit design are:
  - 25º to 30º for Material 1, the clays.
  - 35º to 40º for Material 2, the Pioneer Sandstone, the Gordon Limestone and the Gordon Shale.

- 48º for Material 3, the Owen Conglomerate.

- Based on the interpreted projection of the lithology expected to daylight in the pit, the pit may be zoned into three, with the NE and upper NW walls at an inter-ramp angle of 40º to 48º, the mid section of the E wall at an inter-ramp angle of 35º to 40º and the SE, SW and lower NW walls at an inter-ramp angle of 25º to 30º.
- The Number five Tunnel, the 323m adit and the Batchelor shaft will be intersected during excavation. Without prior dewatering, disturbing the number five tunnel would cause water inflow into the pit and potential pit slope instability. The impacts of intersecting the other two old workings are difficult to predict because little is known about their current condition.
- The general working equipment suitable for use in this material would be a bulldozer to rip where required and an excavator for loading the material.
- Old Iron Blow dumps may have to be removed entirely due to the cohesionless nature of the loose aggregate, which would otherwise leave the material difficult to stabilise and hazardous for operations.
- In light of the limited geotechnical data available, a thorough geotechnical drilling and laboratory testing programme will be required to advance this project to the standard required for pre-feasibility and feasibility studies. Benefits realised will be in the increased accuracy of optimum design parameters and reduce expenditure during the future studies.
- It is recommended that five to six orientated, HQ triple tube geotechnical holes are drilled and logged geotechnically and structurally. A testing programme should include shear and triaxial strength testing, UCS and tensile strength testing, particle size distribution analysis, Atterberg Limits testing and mineralogical identification. Geo-hydrological testing must be included in the programme.

Metallurgical Assessment

This analysis is based on limited metallurgical data, an inspection of the site, examination of a limited number of samples and discussions with CMT personnel. It has necessarily included an assessment of the potential process response of Copper Clays ore. This is yet to be confirmed by specific metallurgical test work. Hence all the conclusions regarding process routes, recoveries, concentrate grade and costs need confirmation by test work and further analysis.

The analysis identified gravity separation as the process method with the best potential.

Operating and capital costs were generated based on a process route with:

- Grizzly and Jaw crusher.
- Surge bin.
- Trommel scrubber.
- Screens and cyclones.
- In-line pressure jigs.

It is estimated this process would recover 65% of the contained copper. The estimated capital and operating cost are shown in Table 1.

Table 1 Processing Operating and Capital Cost

		Annual Throughput (ktpa)	
Cost Item		**250**	**500**
Capital Cost			
All new equipment	(MA$)	8.00	13.00
Refurbished second-hand equipment	(MA$)	6.00	9.50
Operating Cost	(A$/t)	10.00	7.50

Mining Options and Cost

The proposed mining fleet consists of:

- One 80t Excavator.
- Four 40t articulated six wheel drive trucks.
- One 40t tracked dozer.
- One 12t Motor grader.

Six wheel drive articulated trucks will allow operations to continue in the wet conditions regularly seen at Queenstown, which would make the site clays un-trafficable for rigid body trucks.

Due to the potential difficulties operating on narrow benches above 2,330 mRl, in the area of the existing waste dumps, an additional dozer has been assumed and mining rates reduced by 20% above 2,330 mRl.

The estimated operating cost is:

- Mining.
  - Above 2,330 mRl – A$5.00 /t of rock.
  - Below 2,330 mRl – A$3.50 /t of rock.
- Grade control – A$1.00 /t ore.

Capital cost is estimated at A$1,750k. This includes de-watering the Iron Blow and treating the water prior to release into the environment. Other items included are shown in Table 2.

Table 2 Mining Capital Cost

Item	Cost (k$)
Site Access Road	25
Vehicle Washdown	25
Contractor Mobilisation	50
Workshop	50
Hardstand	50
Office and Crib Room	50
Item	**Cost (k$)**
Dewatering	100
Power Reticulation	50
IT and Coms	20
Security and Safety	50
Water Reticulation	50
Sediment Traps	25
Fuel Storage	50
Contingency (30%)	179
TOTAL	**774**

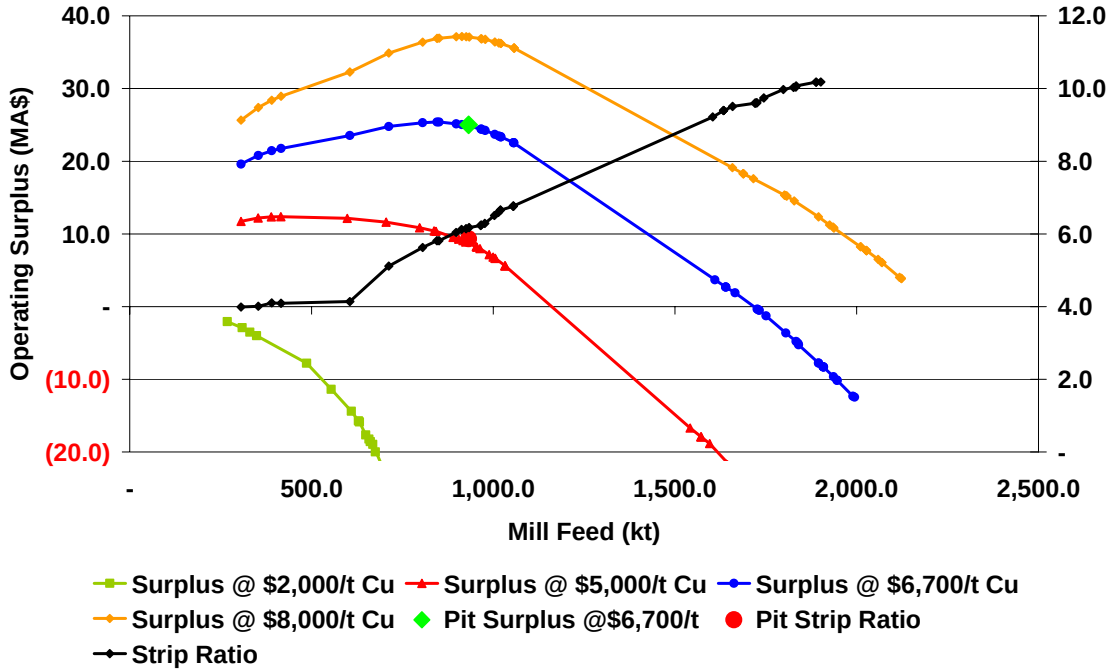
LG Optimisation

Lersch-Grossman Optimisation (LG) is a process used to identify the optimal limit of an open pit. The process considers the potential revenue generated from a block of material, the cost of mining the block material and the cost of mining the blocks above for access. The blocks which must be mined to access an ore block are selected based on overall slope angle including batters, berms and ramps. If the result of the net revenue of the cost is positive, the increment, including the ore block and those which must be mined to access it, are added to the shell. The process considers deeper and deeper material until the increment does not add value. This is considered the optimal limit under the financial scenario being tested. The process is run iteratively with increasing commodity prices to generate a suite of shells increasing in size which can be evaluated under a range of financial scenarios. This analysis provides an understanding of the potential return from a shell and the financial risks associated with selecting a particular shell as the basis of design work if the inputs are different to those forecast.

LG analysis identified potential for a pit with approximately 900,000t to 1,000,000t of mill feed at a grade of 1.9% Cu with a strip ratio of around 6.0 t:t. This would provide between 10,000t and 11,000t of recovered Cu. A pit this size is close to optimal for copper prices of between US$5,000 /t and US$8,000 /t.

Figure 1 shows how the operating surplus for commodity prices above US$5,000 /t grows up to shells containing between 900 kt and 1,000 kt of mill feed. Above this the surplus starts to drop, then falls dramatically when the shells jump in size from 1,100 kt of mill feed to more than 1,500 kt. For the very low sensitivity case none of the shells generates a positive discounted surplus.

Figure 1 LG Optimisation and Pit Total Mill Feed vs. Operating Cash Surplus and Strip Ratio



Mine Design and Scheduling

At a cut off grade (COG) of 0.5% copper the conceptual design contains 6.4 million tonnes of rock, of which 950,000t is mill feed at a grade of 1.8% Cu containing 16,600t of copper and providing 10,800t of recovered copper. This gives a strip ratio of 5.9:1.

The conceptual pit design has contents which very closely replicate the LG optimisation results. This validates the optimisation as a good estimate of the potential mill feed quantity, head grade and strip ratio, when operational and access constraints are applied.

The schedule highlights the need for a twelve month pre-stripping period followed by approximately four years of treatment giving a project life of five years.

The contents of the pit are shown in Table 3 and the schedule in Table 4.

Table 3 Conceptual Design Contents

Level	Rock	Mill Feed			
mRl	kt	kt	% Cu	t Cu	t Cu Rec
2,390	0	-	-	-	-
2,385	14	-	-	-	-
2,380	59	-	-	-	-
2,375	125	-	-	-	-
2,370	93	-	-	-	-
2,365	176	-	-	-	-
2,360	132	-	-	-	-
2,355	249	-	-	-	-
2,350	143	0	0.8%	2	2
2,345	239	5	1.3%	62	40
2,340	238	10	1.3%	132	86
2,335	403	18	1.2%	221	144
2,330	219	13	1.1%	150	98
2,325	436	20	1.1%	224	146
2,320	230	20	1.4%	283	184
2,315	559	55	1.6%	870	565
2,310	208	43	1.8%	779	507
2,305	753	103	1.9%	2,001	1,301
2,300	203	54	1.5%	836	544
2,295	645	101	2.0%	2,007	1,305
2,290	140	57	2.7%	1,512	983
2,285	434	90	1.8%	1,597	1,038
2,280	84	40	2.2%	866	563
2,275	272	94	1.4%	1,275	829
2,270	96	58	1.6%	936	609
2,265	122	66	1.4%	915	595
2,260	49	36	1.5%	552	359
2,255	65	52	2.3%	1,187	771
2,250	16	13	1.7%	221	144
Total	**6,403**	**948**	**1.8%**	**16,631**	**10,810**

Table 4 Schedule

Period		TOTAL	Y1H1	Y1H2	Y2H1	Y2H2	Y3H1	Y3H2	Y4H1	Y4H2	Y5H1	Y5H2	Y6H1
Physicals													
Mining													
Total Movements	kt	6,402.8	750.0	1,500.0	1,500.0	750.0	750.0	750.0	402.8	-	-	-	-
Ore Mined													
Tonnes	kt	947.7	-	53.5	161.9	129.3	148.4	210.3	244.3	-	-	-	-
Grade	%	1.8%	-	1.2%	1.7%	1.8%	2.2%	1.7%	1.7%	-	-	-	-
Metal	t	16,631	-	650	2,674	2,299	3,295	3,645	4,069	-	-	-	-
ROM Stocks													
Tonnes	kt		-	-	36.9	41.2	64.6	149.9	269.2	144.2	19.2	-	-
Grade	%		-	-	1.7%	1.8%	2.2%	1.7%	1.7%	1.7%	1.7%	-	-
Metal	t		-	-	610	733	1,435	2,598	4,500	2,411	321	-	-
Mill Feed													
Tonnes	kt	947.7	-	53.5	125.0	125.0	125.0	125.0	125.0	125.0	125.0	19.2	-
Grade	%	1.8%	-	1.2%	1.7%	1.7%	2.1%	2.0%	1.7%	1.7%	1.7%	1.7%	-
Metal	t	16,631	-	650	2,065	2,176	2,593	2,481	2,167	2,090	2,090	321	-
Mill Production													
Concentrate	t	24,023	-	939	2,982	3,143	3,745	3,584	3,130	3,018	3,018	464	-
Grade	%	45.0%	-	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	-
Metal	t	10,810	-	423	1,342	1,414	1,685	1,613	1,408	1,358	1,358	209	-

Financial Analysis

The project produces a positive $NPV_{8\%}$ for copper prices greater than US$5,000 /t but has a negative $NPV_{8\%}$ for the low sensitivity case of US$2,000 /t. The base case has an operating margin of around 50% of revenue. While this is a good margin the short project life, three and a half years of full plant production, keeps the base case $NPV_{8\%}$ low at ~A$18M. This does offer an opportunity should more mill feed be identified to extend the mine life and increase the $NPV_{8\%}$.

The cashflow schedule is shown in Table 5.

Table 5 Cashflow Schedule

Period		TOTAL	Y1H1	Y1H2	Y2H1	Y2H2	Y3H1	Y3H2	Y4H1	Y4H2	Y5H1	Y5H2	Y6H1
Revenue													
Gross Revenue (On Payable Metal)													
US$8,000/t Cu	MUS$	83.5	-	3.3	10.4	10.9	13.0	12.5	10.9	10.5	10.5	1.6	-
US$6,700/t Cu	MUS$	70.0	-	2.7	8.7	9.2	10.9	10.4	9.1	8.8	8.8	1.4	-
US$5,000/t Cu	MUS$	52.2	-	2.0	6.5	6.8	8.1	7.8	6.8	6.6	6.6	1.0	-
US$2,000/t Cu	MUS$	20.9	-	0.8	2.6	2.7	3.3	3.1	2.7	2.6	2.6	0.4	-
Realisation Costs													
Treatment Cost	MUS$	1.1	-	0.0	0.1	0.1	0.2	0.2	0.1	0.1	0.1	0.0	-
Refining Charge	MUS$	1.1	-	0.0	0.1	0.1	0.2	0.2	0.1	0.1	0.1	0.0	-
Transport	MUS$	3.6	-	0.1	0.4	0.5	0.6	0.5	0.5	0.5	0.5	0.1	-
TOTAL	**MUS$**	**5.8**	**-**	**0.2**	**0.7**	**0.8**	**0.9**	**0.9**	**0.7**	**0.7**	**0.7**	**0.1**	**-**
Exchange Rate			0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86
Net Revenue													
US$8,000/t Cu	MA$	90.4	-	3.5	11.2	11.8	14.1	13.5	11.8	11.4	11.4	1.7	-
US$6,700/t Cu	MA$	74.7	-	2.9	9.3	9.8	11.6	11.1	9.7	9.4	9.4	1.4	-
US$5,000/t Cu	MA$	54.0	-	2.1	6.7	7.1	8.4	8.1	7.0	6.8	6.8	1.0	-
US$2,000/t Cu	MA$	17.6	-	0.7	2.2	2.3	2.7	2.6	2.3	2.2	2.2	0.3	-
Costs													
Royalty (5.00% of Net Revenue)													
US$8,000/t Cu	MA$		-	0.2	0.6	0.6	0.7	0.7	0.6	0.6	0.6	0.1	-
US$6,700/t Cu	MA$		-	0.1	0.5	0.5	0.6	0.6	0.5	0.5	0.5	0.1	-
US$5,000/t Cu	MA$		-	0.1	0.3	0.4	0.4	0.4	0.4	0.3	0.3	0.1	-
US$2,000/t Cu	MA$		-	0.0	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-
Operating													
Mining	MA$	25.5	3.8	7.3	5.3	2.6	2.6	2.6	1.4	-	-	-	-
Grade Control	MA$	0.9	-	0.1	0.2	0.1	0.1	0.2	0.2	-	-	-	-
Treatment	MA$	9.5	-	0.5	1.3	1.3	1.3	1.3	1.3	1.3	1.3	0.2	-
Admin	MA$	0.9	-	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-
TOTAL	**MA$**	**36.9**	**3.8**	**7.9**	**6.8**	**4.1**	**4.1**	**4.2**	**3.0**	**1.4**	**1.4**	**0.2**	**-**
Capital													
Mining	MA$	1.8	1.8										
Plant	MA$	6.0	2.0	2.0	2.0								
TOTAL	**MA$**	**7.8**	**3.8**	**2.0**	**2.0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Re-habilitation													
Mining	MA$	0.8											0.8
Plant	MA$	(1.5)											(1.5)
TOTAL	**MA$**	**(0.8)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**(0.8)**
Cashflow													
Undiscounted													
US$8,000/t Cu	MA$		(7.5)	(6.5)	1.9	7.1	9.2	8.6	8.2	9.4	9.4	1.4	0.8
US$6,700/t Cu	MA$		(7.5)	(7.1)	0.0	5.1	6.9	6.4	6.2	7.5	7.5	1.2	0.8
US$5,000/t Cu	MA$		(7.5)	(7.9)	(2.4)	2.6	3.9	3.4	3.7	5.1	5.1	0.8	0.8
US$2,000/t Cu	MA$		(7.5)	(9.3)	(6.7)	(1.9)	(1.5)	(1.7)	(0.9)	0.7	0.7	0.1	0.8
Discounted													
US$8,000/t Cu	MA$		(7.2)	(6.1)	1.7	6.1	7.6	6.8	6.2	6.9	6.7	1.0	0.5
US$6,700/t Cu	MA$		(7.2)	(6.6)	0.0	4.4	5.7	5.1	4.7	5.5	5.3	0.8	0.5
US$5,000/t Cu	MA$		(7.2)	(7.3)	(2.2)	2.2	3.2	2.7	2.8	3.7	3.6	0.5	0.5
US$2,000/t Cu	MA$		(7.2)	(8.6)	(6.0)	(1.7)	(1.3)	(1.4)	(0.7)	0.5	0.5	0.1	0.5
$NPV_{8\%}$													
US$8,000/t Cu	MA$		30.3										
US$6,700/t Cu	MA$		18.3										
US$5,000/t Cu	MA$		2.6										
US$2,000/t Cu	MA$		(25.1)										

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The signing of this statement confirms this report has been prepared and checked in accordance with the AMC Peer Review Process. AMC's Peer Review Policy can be viewed at www.amcconsultants.com.au.

Project Manager

James Stoddart		1 April 2008
	Signed	Date

Peer Reviewer

Peter McCarthy		1 April 2008
	Signed	Date

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1 copy to AMC Melbourne office

1 INTRODUCTION

AMC Consultants Pty Ltd (AMC) was requested by Copper Mines of Tasmania Pty Ltd (CMT) to prepare a scoping study for the King Lyell Copper Clays. The study aimed to identify whether the potential of the deposit justifies spending the money required to gather further data to proceed with the feasibility study process.

An initial high-level assessment by AMC prior to the scoping study identified sufficient potential to proceed with the scoping study. The report from this review is in Appendix A.

1.1 Location and History

The Copper Clays deposits are located on the West coast of Tasmania, to the east of the pass over looking Queenstown close to the small community of Gormanston. The deposits are adjacent to the Lyell Highway and approximately 5 km from Queenstown.

The Copper Clays deposits have been known since the late 1800's and were mined between 1892 and 1910 producing 2,750t of copper. This mining was by both alluvial hydraulic and underground mining.

The most recent study of the Copper Clays took place in 1995 and was reviewed in 1998. No development commenced on the basis of this work though some further resource definition drilling occurred.

The 1995 study has an extensive discussion on the history and geological understanding of the Copper Clays. The 1995 report and the 1998 review are included as Appendix B.

1.2 Copper Clays Deposits

There are three copper clay deposits identified in the area; King Lyell, Lyell Consols and Lyell Blocks. Prior to commencement of the scoping study, CMT reviewed the copper clays and decided the study should be based on the King Lyell deposit alone.

The King Lyell deposit is the shallowest with better grade then either the Lyell Consols or the Lyell Blocks. The early underground mining ceased due to difficult ground conditions, as would be expected mining in clays. The generally low grade and small extent of the deposits, coupled with the likely very high cost of mining safely in the anticipated conditions led to underground mining not being considered as a potential option. Hence, if the scoping study does not identify enough potential to further develop the King Lyell deposit as an open pit, it can be assumed that there would be no potential to develop the Lyell Consols or the Lyell Blocks.

1.3 Weather

As the Copper Clays project is mining mainly clays high rainfall will be a significant issue.

Figure 1.1 shows the average monthly rainfall for Queenstown and Figure 1.2 shows the average number of days with greater than 1 mm of rainfall.

Figure 1.1 Average Monthly Rainfall for Queenstown

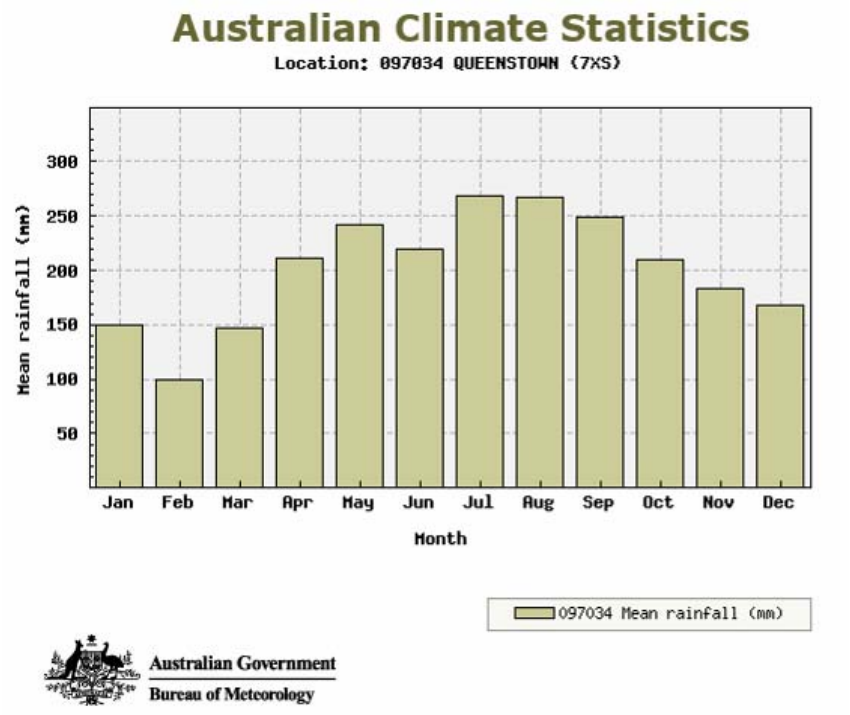
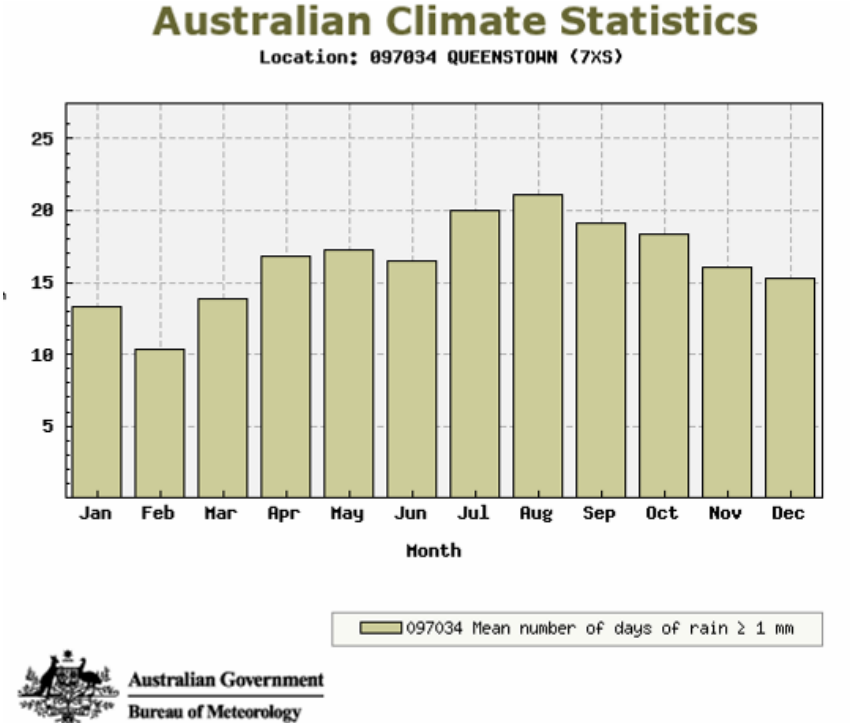


Figure 1.2 Average Number of Days per Month with Greater than 1 mm of Rain for Queenstown



Queenstown has an average of approximately 2,500 mm of rain per annum falling on a little less than 200 days of the year. While the majority of the rain falls between April and October, even the driest month, February has close to 100 mm of rain. The number of days with rain shows a similar characteristic with the most rain days being between April and October, and February having the least number of rain days.

Combining this data suggests the average rain day has between 9.5 mm and 14 mm/day. This implies that the rainfall occurs throughout the year, with the potential for significant rainfall events at any time of the year.

1.4 Site Conditions

The photograph in Figure 1.3 shows the valley floor under which the King Lyell copper clays are situated.

Figure 1.3 King Lyell Copper Clays Site Photo

The shallowest part of the resource is located at the head of the valley; it then plunges toward the east down the valley. The open pit will potentially mine into the hills at the head of the valley, removing the waste dumps and mining toward drives running off the Iron Blow open pit (marked).

The Iron Blow open pit is flooded, as shown in the photograph in Figure 1.4.

Figure 1.4 Iron Blow Open Pit Photo

This presents a very significant risk of a sudden in rush of water to a copper clay mining operation.

2 GEOLOGICAL REVIEW

2.1 Introduction

The AMC geological review of the King Lyell Copper Clays deposit included a preliminary check of the existing geological database; preparation of a geological block model and identifying any additional geological work required to conduct a pre-feasibility study.

AMC produced two models, a conservative model and an optimistic Upside Model of the copper clays. The Upside model has been used as a means of identifying further potential of the deposit, hence it has been used for the scoping study to decide if additional expense would justify gathering data to confirm this potential.

The Upside Model for the CMT Copper Clays King Lyell deposit contains 1.7 Mt to 1.9 Mt at 1.4% to 1.6% copper at a cut off grade of 0.7% copper.

These results are not classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2004 edition (JORC Code) due to insufficient confidence in the data on which they are based.

There are a number of factors that affect confidence in the data and hence the estimate:

- The drill hole dataset is limited to thirteen holes in the central portion of the model, the mineralised zone was extrapolated up and down dip of the known ore zone.
- No information was available on QA/QC aspects of the sample and assay data sets, and a reproducibility test failed on the 1970/71assay data (Wills 1995).
- There is no density data available in the current drill hole database.
- Inaccurate topographic surface.
- Lack of recovery and void data.

To bring the estimation to a standard where it can be classified under the JORC code an infill drilling programme is required. The drilling would be located mainly in the western extent of the area, requiring a minimum of 16 cased diamond drill holes on a 50m X 50m grid enabling the validation of the geological interpretation. All mineralisation should be sampled using coring to ensure good recoveries and enable geotechnical evaluations.

Consistent geological logging of future holes will alleviate discrepancies found in the current database. This will enable a good understanding of the geology and mineralisation allowing the construction of geological wireframes that will provide greater confidence in defining the mineralisation.

It is recommended that bulk density tests be carried out on samples from any future drilling. This is particularly important, as there may be considerable variability in bulk density depending on the copper grade. There may also be a large variability of bulk density between the ore zone and waste.

In summary, to upgrade the mineral resource and to validate the existing work any future drilling should include but not be restricted to:

- Geological and geotechnical logging.
- Collection of samples for assay sample submission.
- Collection of samples for and calculation of bulk density measurements.
- The inclusion of checks blanks and standards to provide industry standard QA/QC data.
- Measurement of recovery of core with particular emphasis on the mineralised zone.
- Establishment of the location of existing voids for previous workings in relation to existing and new drilling.
- Validation of new data against existing data to increase confidence in the drill hole database.

2.2 Geology

The sedimentary sequences of the King Lyell area were deposited in a shallow continental shelf environment, during the early Ordovician through to the middle Devonian. The sequences, for the most part, sit unconformably atop of the Cambrian Mt Read Volcanics. They are made up of the Owen Conglomerate Group, the Pioneer Beds and the Gordon Limestone. Post depositional deformation at the time of the Tabberabbera Orogeny resulted in folding and faulting of the sequences. Several phases of mineralisation occurred in the region, with the Copper Clays being one of the more recent. It is uncertain the exact timing of the mineralisation, but it has been suggested it coincides with Quaternary glaciations as fluvioglacial deposits from the Linda Glacial stage cover the mineralised clays (Wills, 1996).

The Copper Clays are located in the Western Linda Graben, east of the Great Lyell Fault, south of the North Lyell Fault and north of the Owen Spur Fault. The deposits are found at altitudes between 200m and 600m in synclinal valleys under water courses, concurrent with WNW trending Tabberabbera faults.

There are many theories on the genesis of the Copper Clays ore. The paper by Wills (1996) outlines the theories and prevailing arguments. He surmises (direct quotation) “The Copper Clay deposits have geometries which are sub-parallel to the present land surface. They are thought to have formed by native copper precipitation after neutralisation of copper bearing acid groundwater by reaction with Gordon Limestone”.

2.3 Data Provided

The following data was provided by CMT:

- Digital tables in Access Database format of drill hole collars, down hole surveys and assays, in AMG, IMPMINE, METMINE and working grid.

- Digital solids in Surpac format of the copper mineralisation, lithological units and faults located on the working grid.
- Digital topographic surface in Surpac format, located on the working grid.
- Report 'King Lyell Copper Clays – 1995 Open Cut Potential' by K. Wills, 1995.
- Report 'King Lyell Copper Clays – 1998 Evaluation' by T. Weston, 1998.
- Core photographs of the 2005 drilling programme (05KLD series).

2.4 Work Undertaken

The following work was undertaken in Datamine based on the working grid:

- Load drill hole data into Datamine and apply preliminary checks.
- Load wireframes into Datamine and perform a preliminary check on the geological interpretation.
- Construct three-dimensional wireframe of copper mineralisation using an approximate 0.3% copper cut off, ensuring drill holes intersections are included.
- Subset drill data on mineralisation wireframe.
- Conduct statistical analyses of copper grade.
- Composite drill hole assay files.
- Establish estimation parameters and select estimation method.
- Construct block model and estimate copper grade.
- Cut model to topography.
- Evaluate model contents.
- Make recommendations for additional work required to enable a Pre-feasibility study to be carried out and upgrade the model contents to a reportable mineral resource in accordance with the JORC[1] Code.

2.5 Data Validation

The digital drill hole data of seventeen holes was loaded into Datamine for limited validation. No obvious import errors were detected.

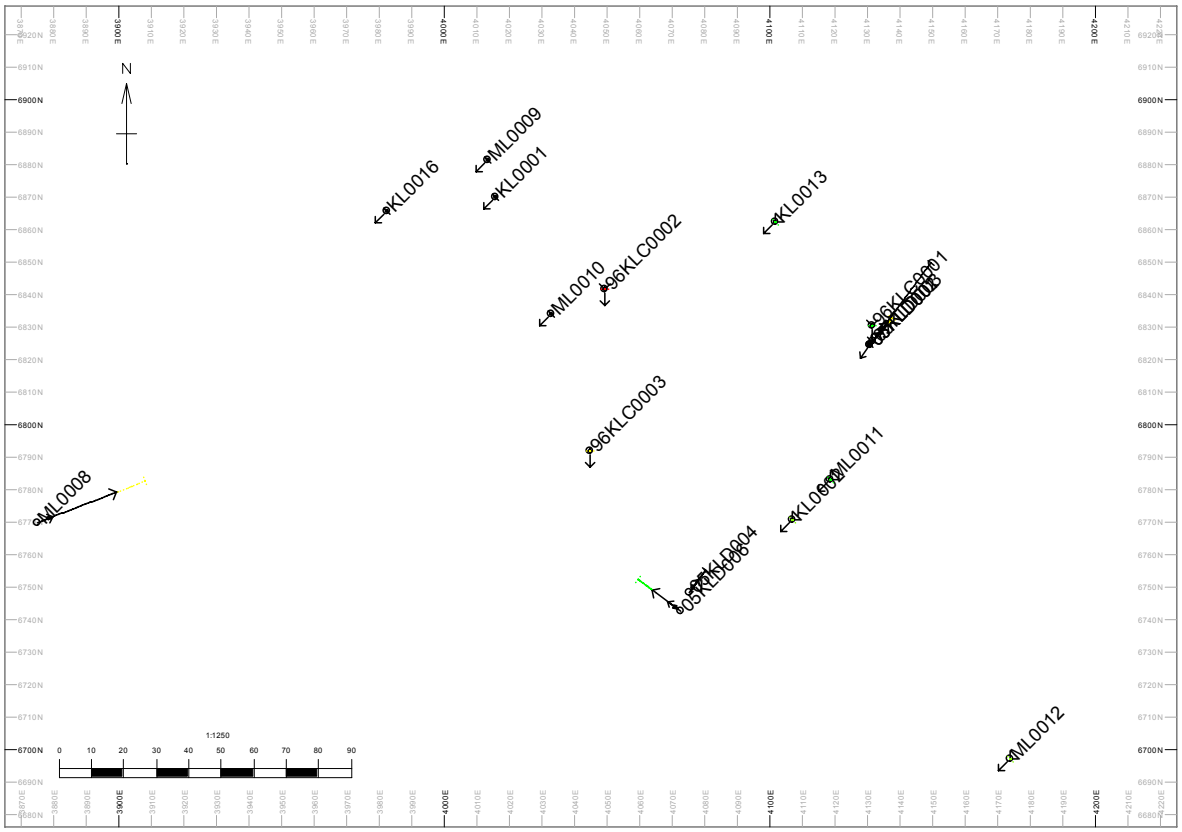
The validation process checked for the following inconsistencies:

- Collar co-ordinates outside of range.
- Consistent FROM and TO values in all files.
- Duplicate records, duplicate holes.

Appendix C contains a list of all holes in the King Lyell drill database.

[1] the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2004 edition

Figure 2.1 Location of Drill Holes King Lyell Deposit



2.6 Geological Interpretation

The geological interpretation provided by CMT was reasonably consistent with the geological model described by Wills (1995). AMC visually inspected the four lithological wireframes provided (mineralised clay, sandstone, limestone and conglomerate) with the drill holes. It was evident the lithology's were not accurately represented, especially the copper clay mineralised zone.

The primary focus of AMC's geological review was to produce a block model containing the copper mineralised zone and it was apparent the mineralised wireframe provided did not include important drill hole data and therefore needed to be corrected.

In general, some lithological wireframe boundaries conflicted with the logged intervals. Inconsistencies in the logged lithology may reflect the different sets of historical drilling (drill hole data dates from 1902 through to 2005). Further interrogation of the sandstone, limestone and conglomerate wireframes was not deemed necessary for this study. AMC however recommends work be done to complete the digital geological model, as this will increase confidence in future mineral resource estimates.

AMC constructed two wireframes of the copper mineralisation at an approximate 0.3% cut off, which were provided to CMT for approval. The Upside wireframe was

modelled using the relevant drill hole data, whilst extending up and down dip of the ore body (beyond drill hole information) to represent exploration potential. The Conservative wireframe was also modelled to contain all relevant data, but did not extend greatly beyond the existing drilling. Appendix D contains plots of the three wireframe with drill hole intercepts.

2.7 Statistical Evaluation of Drill Hole Data

A subset of the 13 drill holes located within the Upside copper mineralised zone wireframe was used for statistical evaluation. Appendix E contains a list of all holes used in the mineral resource estimation. Table 2.1 contains the raw drill hole data statistics for the copper mineralised zone.

Table 2.1 Raw Drill Hole Data Statistics for the Copper Mineralised Zone

	No. Samples	Max	Min	Mean	Median	Std Dev	Variance	Co-eff Variation
Length (m)	146	14	0.04	1.51	1.52	1.14	1.3	0.76
Copper (%)	141	15.5	0.01	1.22	0.57	2.13	4.54	1.75

A log probability plot and log histogram plot for copper percent assays are shown in Figures 2.2 and 2.3. These figures show one population of data and can be grouped for the estimation. It also suggests that grade distribution is relatively continuous.

Figure 2.2 Log Probability Plot of Raw Cu %

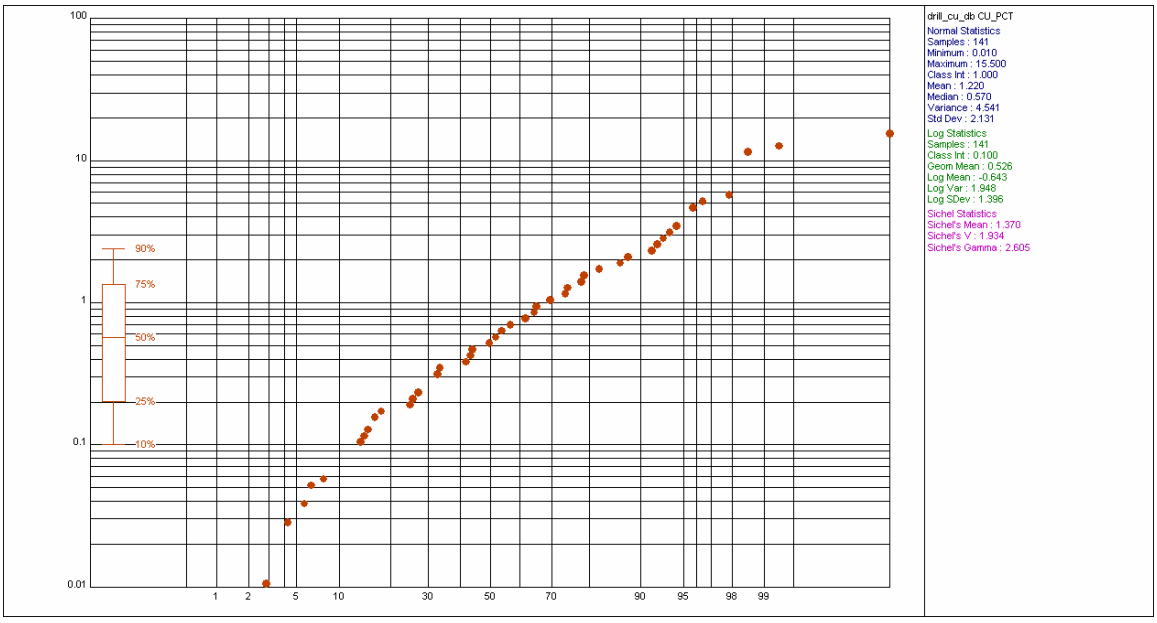
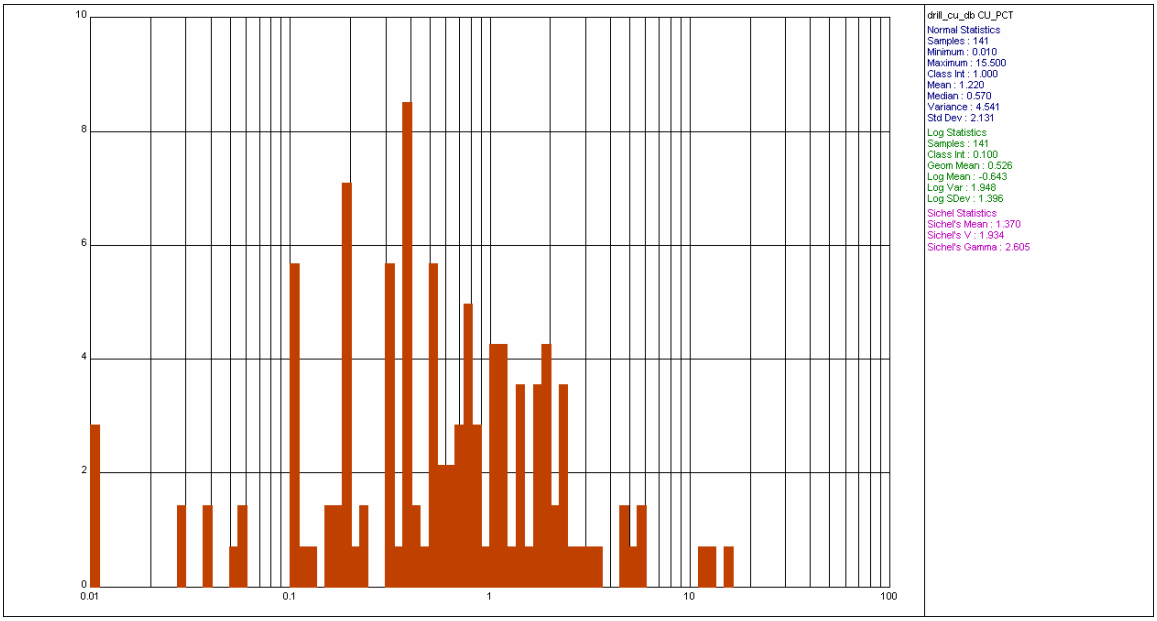


Figure 2.3 Log Histogram Plot of Raw Cu %



2.7.1 Compositing

To assist in providing the best block grade estimates requiring samples to be of a consistent sample support the drill hole assay samples were composited to one metre down-hole composites. The statistics for the composites are shown in Table 2.2. Figures 2.4 and 2.5 show the log probability and log histogram plot for the composited copper assays. This demonstrates the compositing has maintained the integrity of the database.

Table 2.2 Composited 1m Drill Hole Statistics for the Copper Mineralised Zone

	No. Samples	Max	Min	Mean	Median	Std Dev	Variance	Co-eff Variation
Length (m)	220	1	0.54	0.99	1	0.06	0.004	0.06
Copper (%)	206	14.16	0.01	1.22	0.63	1.8	3.23	1.48

Figure 2.4 Log Probability Plot of 1m Composited Cu %

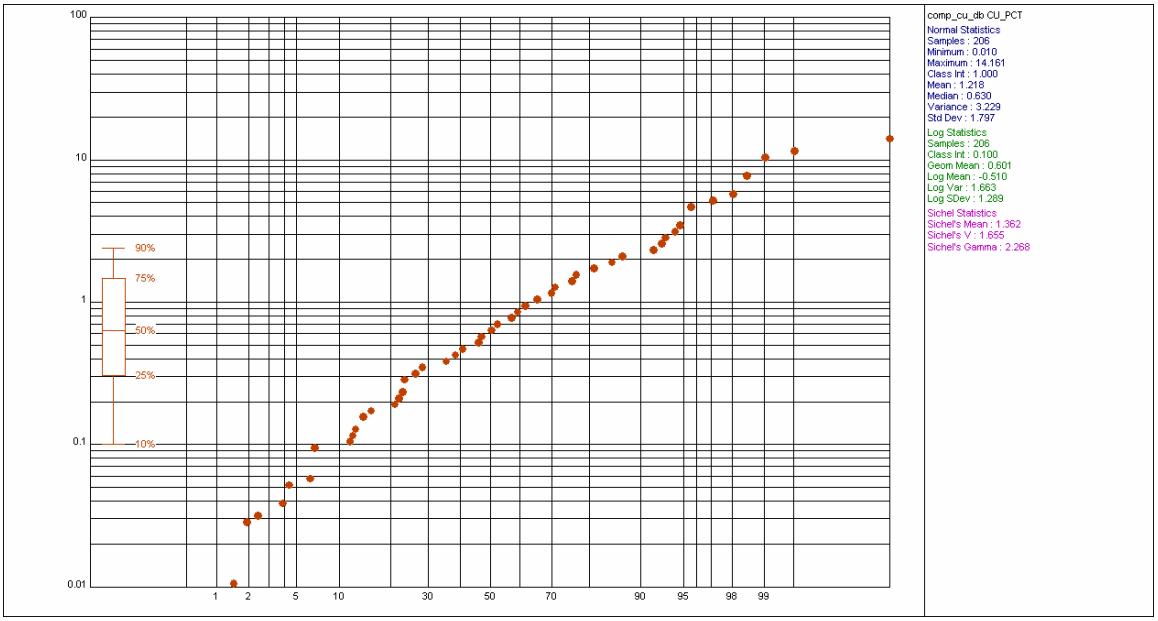
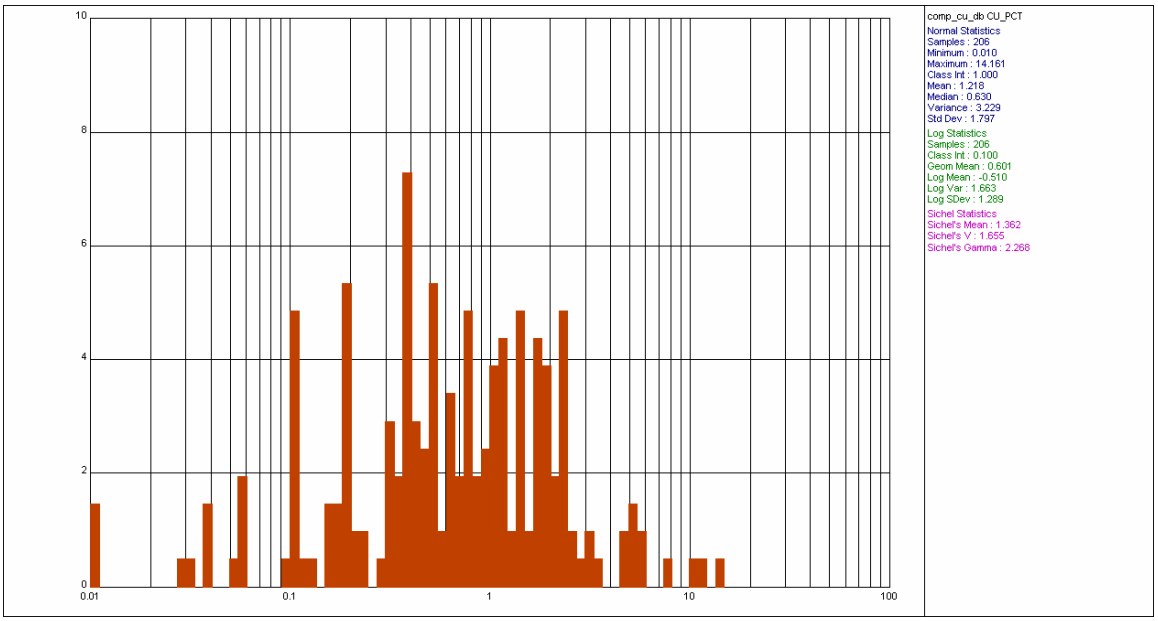


Figure 2.5 Log Histogram Plot of 1m Composited Cu %



2.8 Mineral Resource Estimation

2.8.1 Block Model Parameters

AMC prepared an Upside and a Conservative model using two copper mineralised zone wireframes at an approximate 0.3% copper cut off prepared by AMC and honouring the original geological interpretation provided by CMT.

The block models were constructed with the parameters shown in Table 2.4.

Table 2.3 Block Model Parameters

Model Parameters	Easting	Northing	RL
Origin	3,800	6,600	2,100
Number of cells	15	12	30
Cell size (m)	30	30	10
Minimum sub cell size (m)	7.5	7.5	0.01

The small sub cell size in the RL direction was used to allow the topographic surface to be accurately defined.

Model volumes were calculated and checked against the volume within the mineralised zone wireframe outline for model validation.

2.8.2 Resource Estimation

The inverse distance squared method was used for grade estimation.

The search directions were oriented along strike and down dip. The rotation occurred around the cardinal axis. The order of rotation is Z axis, Y axis and then X axis.

Table 2.5 lists the search and estimation parameters used.

Table 2.4 Search and Estimation Parameters

Parameters	Size
Search Radius	
X	50m
Y	75m
Z	10m
Rotation of Search Ellipse	
X axis	10°
Y axis	-30°
Z axis	60°
Search Range Multiply Factor	
1st Pass	1
2nd Pass	2
3rd Pass	4
Number of samples	
1st Pass (Min/Max)	4 / 30
2nd Pass (Min/Max)	4 / 30
3rd Pass (Min/Max)	2 / 20

2.8.3 Bulk Density

Bulk density measurements were not included in the drill hole data set provided. AMC used a default density of 2.2 t/m^3 based on clay, copper and waste rock densities from standard values contained in the Field Geologists' Manual of the AusIMM (Monograph 9).

The bulk density used in previous estimates was discussed in the report by K. Wills, 1995. Wills suggests a specific gravity (SG) of 2.1 t/m^3 for resource estimation purposes. Wills also suggests the SG of clay is 2.6 t/m^3 while applying the principle that voids making up 20% of the area, reducing the SG to 2.1 t/m^3. This figure is reasonable; however, even if the voids are present, the initial density of 2.6 t/m^3 for clay is high. The Field Geologist's Manual suggests the density of clay is 1.9 t/m^3. Wills also states that work by Batchelor in 1902, measured the SG of King Lyell clay at 2.08 t/m^3 yet used 2.4 t/m^3 in tonnage estimations. This density may have resulted in an over estimation of tonnes in Batchelor's estimate.

2.9 Results

All of the available drill hole data was used. AMC also created a model using the original CMT wireframe of the mineralised zone to compare the results. Table 2.3 shows the differences between the global (no cut off applied) tonnages and grade. It shows that the initial CMT interpretation was excluding data. Appendix F has further comparisons of the three models.

Table 2.5 Global Tonnes within Copper Mineralised Zones

Copper Mineralisation	Tonnes	Cu%
AMC Upside Model	2.9 Mt	1.07
AMC Conservative Model	2.5 Mt	0.90
CMT Initial Model	2.4 Mt	0.64

2.9.1 Upside Model Contents

Due to limitations of the scope of work and limited data, the grade and tonnage estimation focused on defining Upside Model tonnages and grade delineated by the drilling.

The grade and tonnage estimates in this report have **not** been classified in accordance with the Australasian Code for Reporting of Exploration Results, Mineral Resources and Ore Reserves, 2004 edition (the "JORC Code").

The contents for the Upside Model copper mineralised zone model at a 0.7% copper cut off is shown in Table 2.6. See Appendix F for the global figures from the Upside model generated. See Appendix G for plots of the copper model.

Table 2.6 CMT Copper Clays King Lyell Upside Model

Mineralisation Type	Cut Off Grade (%)	Mt	Cu (%)
Copper	0.7	1.7 to 1.9	1.4 to 1.6

Note These figures are for company internal use only and must not be released publicly except under section 18 of the JORC code which requires using a range of figures and must include all qualifying statements.

The primary reasons for not classifying the results as a Mineral Resource under the JORC Code are:

- The drill hole dataset is limited to thirteen holes in the central portion of the model. The mineralised zone was extrapolated up and down dip of the known ore zone.
- No information was available on QA/QC aspects of the sample and assay data sets.
- There is no density data available in the current drill hole database.
- Inaccurate topographic surface.
- Lack of recovery and void data.

It is noted by Wills (1995) that re-assay of the 1970/71 samples gave significantly different results. This casts uncertainty on the validity of assay data within the current drill hole database. Sections produced through the model have also highlighted the need for the natural surface to be surveyed accurately. Old workings have been identified throughout the King Lyell deposit and data to delineate voids is necessary to estimate recoverable tonnes, though probably immaterial to the Upside Model contents. Poor recoveries and contamination were highlighted as issues in previous drilling. Procedures should be put in place in future drilling programmes to counter these problems.

3 GEOTECHNICAL ASSESSMENT

The geotechnical assessment of the King Lyell Copper Clays deposit included a review of data available, assessment of the geotechnical materials for design parameters based on the data available, and recommendations for further investigation and analysis.

The conclusions from the assessment were:

- The six lithologies in the project area can be grouped into three sets of distinguishable geotechnical engineering materials. Each of these materials is expected to differ in behaviour when exposed.
- The inter-ramp slope angles that should be used in preliminary pit design are:
  - 25º to 30º for Material 1, the clays.
  - 35º to 40º for Material 2, the Pioneer Sandstone, the Gordon Limestone and the Gordon Shale.
  - 48º for Material 3, the Owen Conglomerate.
- Based on the interpreted projection of the lithology expected to daylight in the pit, the pit may be zoned into three, with the NE and upper NW walls at an inter-ramp angle of 40º to 48º, the mid section of the E wall at an inter-ramp angle of 35º to 40º and the SE, SW and lower NW walls at an inter-ramp angle of 25º to 30º.
- The number five tunnel, the 323m adit and the Batchelor shaft will be intersected during excavation. Without prior dewatering disturbing the number five tunnel would cause water inflow into the pit and potential pit slope instability. The impacts of intersecting the other two old workings are difficult to predict because little is known about their current condition.
- The general working equipment suitable for use in this material would be a bulldozer to rip where required and an excavator for loading the material.
- Old Iron Blow dumps may have to be removed entirely due to the cohesion less nature of the loose aggregate, which would otherwise leave the material difficult to stabilise and hazardous for operations.
- In light of the limited geotechnical data available, a thorough geotechnical drilling and laboratory testing programme will be required to advance this project to the standard required for pre-feasibility and feasibility studies. Benefits realised will be in the increased accuracy of optimum design parameters and reduce expenditure during the future studies.
- It is recommended that five to six orientated, HQ triple tube geotechnical holes are drilled and logged geotechnically and structurally. A testing programme should include shear and triaxial strength testing, UCS and tensile strength testing, particle size distribution analysis, Atterberg Limits testing and mineralogical identification. Geo-hydrological testing must be included in the programme.

3.1 Data Review and Assessment

Assessing the geotechnical potential for open pit mining the King Lyell deposit required an estimate of the material properties to predict their behaviour, in terms of excavation and stability.

To assist with the geotechnical assessment of the King Lyell deposit CMT provided AMC with lithological logs for drill holes:

- 05KL001, 05KL002, 05KL003, 05KL004, 05KL005 and 05KL006 (incomplete) drilled in 2005.
- 96KLC0001, 96KLC0002 and 96KLC0003 drilled in 1996.
- KL0013 drilled in 1971.
- KL0001, KL0002 and KL0016 drilled in1970.
- ML0008, ML0009, ML0010, ML0011 and ML0012 drilled in1902.

Drill core photographs of:

- 05KL001 (missing photos for core between 0m to 18m), 05KL002, 05KL003, 05KL004 and 05KL006 (missing photos for core between 0m to 52m).

The drill core photographs are included in Appendix H.

Surpac wireframe files:

- Lithology of the King Lyell Copper Clays site.
- Location and inferred dip of local faults.
- Topography for the King Lyell Copper Clays and Iron Blow site and wireframes for the old development at the site including development tunnels, adits and shafts proximal to the proposed open pit.

3.2 Site Conditions

3.2.1 Local Geotechnical – Geological Units

Six rock types were intersected in the core photographs available for the 2005 drill holes. These are characterised in Table 3.1. Due to the limited amount of data, assumptions on the physical properties of the material have been based on the core photographs, visual deductions during the site visit and the lithological logs.

Table 3.1 Descriptions of the Major Geotechnical – Geological Units

Unit	Intersected in Drill holes	Weathering, Structure and Texture	Geotechnical Property Description and Comments
Clays 	05KL001 05KL002 05KL003 05KL004 05KL006	Completely weathered, some oxidation of clays, mottled appearance	Expected to be cohesive but weak in strength, will crumble under firm blows with the point of a geological hammer, may be peeled by a pocket knife, spatially variable weathering and alteration. Described by site personnel as saturated with approximately 20% moisture content. Desiccated appearance when dried out.
Mineralised clays 	05KL002	Completely weathered, oxidised with some black mottling	Expected to be similar to the clays described above, may contain visible mineralisation with variable concentration. Site personnel advised that nodular fragments with native copper within them were common in the mineralised unit of clay. These nodules will be much stronger than the clay but are not expected to alter the geotechnical properties of the clay.
Gordon Limestone 	05KL006	Partially to completely weathered, massive	Strength estimated from core review suggests the limestone is a weak highly fragmented material. The observed limestone outcrop site suggests a shallow indentation will result from a firm blow with the point of a geological hammer.
Pioneer Sandstone 	05KL002 05KL003 05KL006	Slightly to partially weathered, massive	Expected to be medium to weak in strength. The strength will be influenced significantly by the degree of weathering. Likely to be gritty where highly weathered, large amount of fragmented core therefore material probably has a low structural integrity. Occasionally oxidised, indicating water movement through the material.
Owen Conglomerate 	05KL002 05KL003 05KL006	Slightly to completely weathered, large sub-angular fragments	Expected to be strong, requiring more than one blow of a geological hammer to fracture it, low degree of fragmentation and moderate structural integrity. Areas of complete weathering are expected to have a low strength and poor structural integrity. Observed high angle jointing with oxidised surface, though this may not be dominant characteristic.
Gordon Shale 	05KL004	Slightly to partially weathered, fissile appearance	Likely to be weak to moderately strong, requiring more than a single blow to fracture the material, strength is expected to be weaker when loading perpendicular to the bedding. Moderate fragmentation and moderate structural integrity.

Outcrops for two of the six lithology's were observed during the site visit. These were the mineralised clay unit (Figure 3.1a) and the Gordon Limestone (Figure3.1b).

Figure 3.1 A. Mineralised Clay, in a Gully Above the Copper Clay Deposit. B. Gordon Limestone Outcrop at the Site of the Copper Clay Deposit

A.

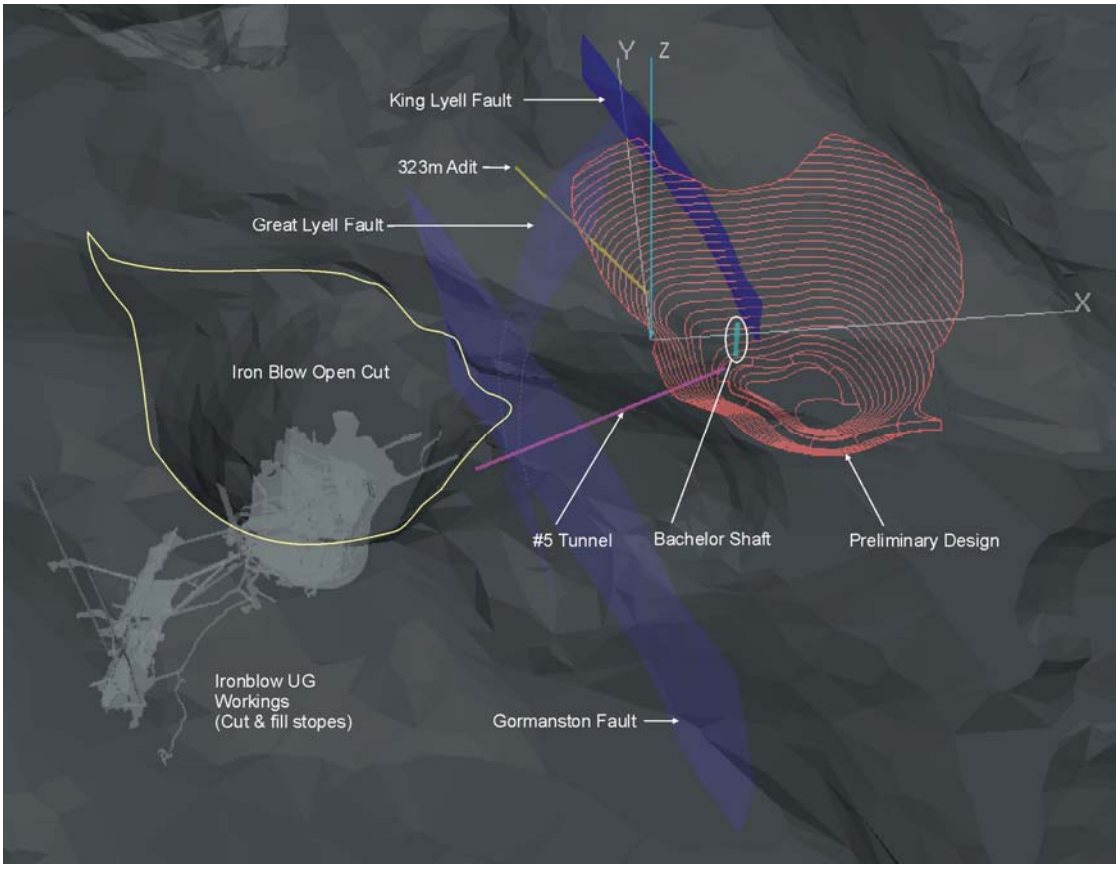
B.

3.2.2 Geological Structures

Three faults roughly bound the King Lyell deposit (Figure 3.2). These are the Gormanston fault which strikes approximately NW to SE, the Great Lyell fault which strikes approx. N to S and the King Lyell fault which strikes almost parallel to the Gormanston fault and cuts through the centre of the deposit.

Very little is known about the faults. Their strike and extension have been digitised from old surface maps. The dips have been inferred and there is no data on the possible presence or flow rate of water on the faults.

Figure 3.2 Structures and Old Development Around the King Lyell Deposit



3.2.3 Old Development and Workings

In conjunction with the geotechnical-geological site conditions and the structural deformation of the area, the influence of any previous development and workings must be considered so as to thoroughly assess the geotechnical potential of the project.

The Iron Blow Open Cut (Figure 3.2), located to the WNW of the King Lyell Copper Clays deposit, was worked from 1823 to 1929. Waste rock from the workings was dumped over the eastern and north-eastern walls of the Iron Blow, creating un-engineered dumps which flank the King Lyell deposit (Figure 3.3).

Figure 3.3 Photograph of the Iron Blow Dumps, Looking South West Down a Gully, Above the Copper Clays Deposit

Old Iron Blow cut and fill stopes exist to roughly the west of the King Lyell deposit (Figure 3.2). These old underground workings extend away from the proposed open pit and not are expected to influence the potential or design of the proposed open pit. However, the No. 5 King Lyell, Iron Blow access tunnel extends approx. northeast from these old stopes towards the King Lyell deposit and potentially daylights in the proposed pit. Unless it is dewatered in advance, intersecting or even just disturbing the number five tunnel may induce water inflows from the Iron Blow catchment and amongst other non-geotechnical ramifications such inflows will seriously compromise the stability of the proposed pit.

Two other old workings exist in the King Lyell deposit. These are the 323m Level adit (1898) which strikes north-west and the Batchelor shaft which sits in the centre of the King Lyell deposit. Both old workings will be intersected in the excavation of the proposed pit. The condition of these workings is not known.

3.3 Geotechnical Assessment

To gauge the preliminary estimates of slope stability properties for the various materials onsite the estimated geotechnical properties (described in Section 3.2.1) are compared to materials of known slope stability properties. Two of the most relevant properties are a materials cohesive strength (c) and angle of friction (Φ).

Hoek and Bray (1981) and Wyllie and Mah (2004) suggest a range of c and Φ values, based on shear strength testing, for a variety of materials. Use of these values is only appropriate where general estimation are required, such as in the case of scoping studies, where actual test results are not yet available.

For cohesive clays the suggested c and Φ values vary between 0.01 Mpa to 0.25 MPa and 7 to 35 degrees respectively. Structures are not expected to significantly influence the geotechnical behaviour of the clay, but if present, will act to weaken the rock mass as the surfaces will likely be smooth or polished and planar. CMT personnel on site describe the clays as having roughly 20% moisture content. This moisture will decrease

the cohesive strength and angle of friction of the clay, therefore decreasing the slope angle at which this material may be excavated without failure. If it is not possible to drain the clays an optimal overall slope angle will not be achievable and it is expected the bench and inter-ramp slopes will bulge at the toe and will most likely result in failure. Mottled colouring indicates fluctuating water table levels, which will again affect the optimal overall slope angle achievable if the material cannot be drained.

Soft sedimentary rocks such as shale may vary in c and Φ values between 1 Mpa to 2 MPa and 25 to 35 degrees, respectively. Considering the high weathering and low structural integrity of the Gordon shale observed in the core photographs these values may overestimate the slope stability properties of the site material. Slightly more conservative values are recommended for the preliminary design of the pit where the pit limits intersect this rock type. Also to be considered in assessing the slope stability of the Gordon shale is the laminated nature of shale and the fissile behaviour when the material is exposed. From the limited observable Gordon shale intersected in drill hole 05KL004, artificial breaks sub-parallel to the bedding are frequent, forming high alpha angles to the drill core (which dips between 80 and 90 degrees). Slope instability and failure are probable where the exposed shale-bedding dips into the pit.

Limestone and sandstone are described by Hoek and Bray (1981) as being hard sedimentary rocks with c and Φ values ranging between 10 to 30 MPa and 35 to 40 degrees respectively. However, both the Gordon limestone and the Pioneer sandstone are, in some places, highly weathered and therefore likely to be weak resulting in lower cohesive strengths and friction angles. As with the Gordon shale the conservative values, compared to the typical values, should be used when these materials daylight in the pit limit.

A conglomerate is a hard sedimentary rock, but depending on the matrix binding the conglomerate fragments, the strength will vary significantly. The matrix of the Owen conglomerate is unknown but higher range c and Φ values may be expected as the rock in the core photographs appears to have an overall high structural integrity, with infrequent fragmentation. Wyllie and Mah (2004) suggest the angle of friction for a conglomerate is more likely to be similar to that of high friction rock types such as basalts and granites. The strength of the conglomerate is expected to be greater than all the other materials in the proposed pit. An area of intense weathering was identified from the core photographs. The boundaries defining the weathering are sharp, suggesting the weathering occurred along a structure and is not indicative of the general geotechnical condition of the conglomerate.

Little is known about the materials which make up the dumps flanking the King Lyell deposit, other than the placement of the material was not engineered. It can be assumed that the dumps will be cohesion less (Φ = 0 degrees), un-graded and uncompacted, baring slight compaction due to environmental exposure over time. The loose aggregate material is likely to have angles of friction between 30 to 50 degrees and taking into consideration the method of placement, the material will be sitting at the meta-stable angle of repose, between stability and failure. Excavation of the dumps will require stabilisation at the base of the dump but ultimately the entire dump may have to be removed to achieve stability in the walls above the proposed pit.

3.4 Geotechnical Parameters for Pit Design

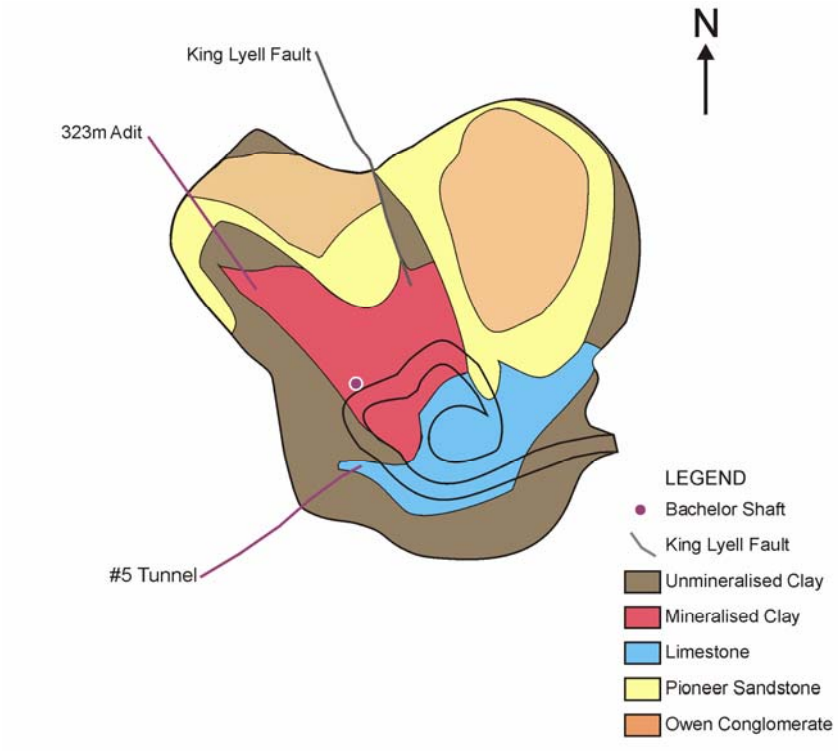
Although there are essentially six different lithologies intersected by the 2005 drill holes, these materials may be grouped into three sets with similar geotechnical properties. Upon exposure these sets can be expected to differ in behaviour. Based on this grouping the inter-ramp design angle for each material has been defined (Table 3.2).

Table 3.2 Preliminary Inter-ramp Design Angles

Material		Inter-ramp angle (degrees)	Comments
1	Clays	25 to 30	Achievable only with active draining
2	Pioneer Sandstone, Gordon Limestone and Gordon Shale	35 to 40	Assumed to be free draining although the shale may need draining
3	Owen Conglomerate	48	

An interpreted projection of the lithology's which are expected to daylight in the pit limit is shown in Figure 3.4. This interpretation has been based on the lithological model provided by CMT and a preliminary pit design by AMC. It is important to note that the lithology model has been designed on a limited number of drill holes, especially up-valley where there are no drill holes to confirm the rock types. For optimised geotechnical design parameters the lithology model will need to be rigorously defined through a planned geological drilling and interpretation programme.

Figure 3.4 Interpreted Projection of the Lithology's Exposed in the Preliminary Pit Design



The preliminary pit design may be zoned into three:

- The NE and upper NW walls (Zone 1), in which the dominant materials are the Pioneer sandstone and Owen conglomerate. The suggested inter-ramp angle is between 40º and 48º.
- The Pit floor and a portion of the E wall (Zone 2), in which the dominant material is limestone, with a suggested inter-ramp slope angle of 35º to 40º.
- The SE, SW and lower NW walls (Zone 3), in which the dominant materials are the mineralised and un-mineralised clay and the suggested inter-ramp angle is 25º to 30º.

It can be expected that the rock types which daylight may change with the collection of more exact data and the re-optimisation of the pit design.

The proposed pit is likely to intersect with the King Lyell fault. Although it is unknown whether there is water in the fault, it can be assumed that the rock mass around the fault will be poorer in quality and instability more likely.

The number five tunnel, the 323m adit and the Bachelor shaft will daylight in the pit walls. Water inflow from the Iron blow through the number five tunnel will cause serious pit slope instability and will have to be mitigated before excavation begins. The condition of the remaining old workings is unknown therefore the impact of their exposure, on the stability of the pit walls, is unpredictable. Monitoring will be required during the excavation in these areas.

General working equipment which would be suitable for use in this material includes a bulldozer to rip where required and an excavator for loading the material. Small areas of stronger rock may be encountered, due to special variability in weathering, in which case paddock-blasting may be necessary.

The cohesion less nature of the dump material may make its removal difficult as the load of the working equipment may not be supported, presenting a hazard. Excavating only sections of the dumps may also cause uncontrollable instability therefore it may be beneficial to remove the entire dump if part is required to be excavated.

3.5 Recommendations

The geotechnical material properties have been judged from a limited amount of data. Any further geotechnical assessment of the King Lyell deposit will require a testing programme that includes drilling specific geotechnical-focused drill holes and laboratory testing to validate and quantify the preliminary geotechnical estimates. The benefits from investing time and capital into the data collection for the understanding of the geotechnical-geological model will be seen in the reduced expenditure during the feasibility assessment.

The following work will provide the data required for a pre-feasibility and feasibility assessments on the King Lyell deposit.

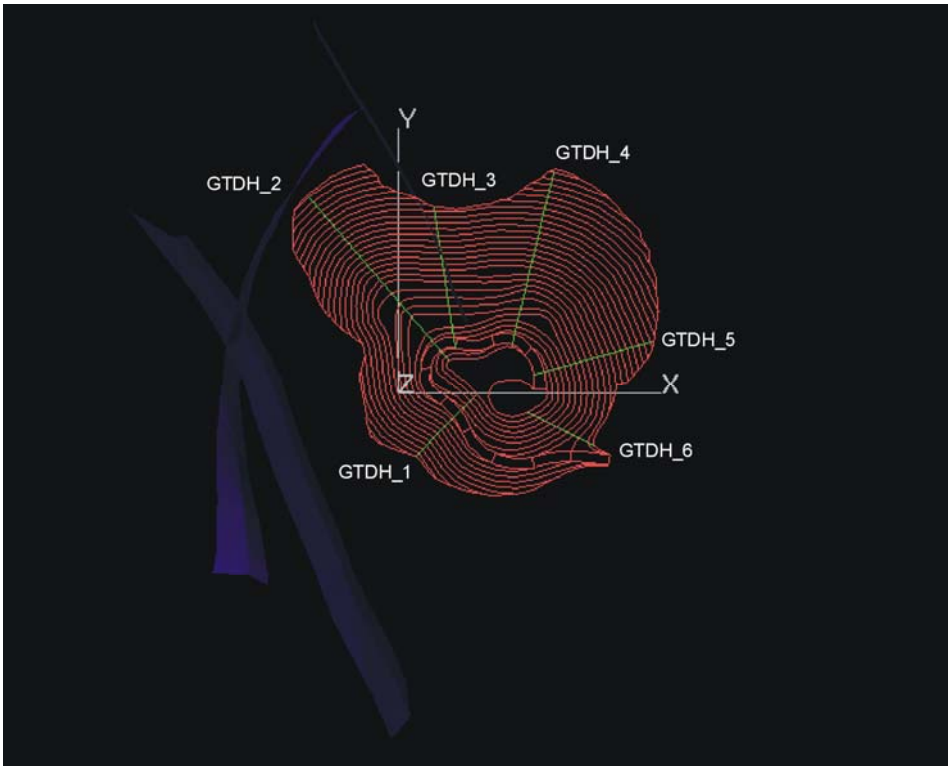
3.5.1 Drilling

Initially five to six geotechnical diamond drill holes will be required. They should be orientated, HQ triple tube drill holes and should be logged geotechnically and structurally. The drill holes should target the final pit walls and align with the pit wall angles. Suggested locations are detailed in Table 3.3 and are shown in Figure 3.5. The exact positions may have access or other such issues and may have to be moved to accommodate for site conditions. Drilling to define a more rigorous geological model should also be logged for geotechnical purposes.

Table 3.3 Suggested Geotechnical Drill Hole Design

BHID	Easting	Northing	RL	Azimuth	Dip	Length (m)
GTDH_1	4003.7	6764.9	2328	46.1	41	111
GTDH_2	3905.4	7000.7	2397	137.7	36	241
GTDH_3	4019.7	6994.1	2363	169.6	37	162
GTDH_4	4127.4	7024.5	2392	190.6	41	209
GTDH_5	4223.6	6872.1	2348	253.5	41	144
GTDH_6	4174.4	6773.1	2302	297.6	37	85

Figure 3.5 Suggested Geotechnical Drill Dole Design



3.5.2 Laboratory and Field Testing

Testing on each material that will be excavated should be carried out to determine material properties. At least three samples per material type, per test type will be required. The testing programme should include shear box and triaxial strength testing to determine c and Φ values, UCS and tensile strength testing of the rock material, particle size distribution (PDS) testing using both the sieve and hydrometer methods, Atterberg limits testing to determine Plastic Limits (PL) and Liquid Limits (LL) of the clays, mineralogical identification of the clays using X-Ray Diffraction (XRD) and hydrological testing using a series of observation and pumping wells to determine the hydrogeology influences on the geotechnical pit design.

4 METALLURGICAL ASSESSMENT

This analysis is based on limited metallurgical data, an inspection of the site, examination of a limited number of samples and discussions with CMT personnel. It has necessarily included an assessment of the potential process response of Copper Clays ore. This is yet to be confirmed by specific metallurgical test work. Hence all the conclusions regarding process routes, recoveries, concentrate grade and costs need confirmation by test work and further analysis.

The analysis identified gravity separation as the process method with the best potential. Operating and capital costs were generated based on a process route with:

- Grizzly and Jaw crusher.
- Surge bin.
- Trommel scrubber.
- Screens and cyclones.
- In-line pressure jigs.

It is estimated this process would recover 65% of the contained copper. The estimated capital and operating cost are shown in Table 4.1.

Table 4.1 Processing Operating and Capital Cost

		Annual Throughput (ktpa)	
		250	**500**
All New Equipment	(MA$)	8.00	13.00
Refurbished 2nd Hand Equipment	(MA$)	6.00	9.50
	(A$/t)	10.00	7.50

4.1 Historical Background

The Copper Clays deposits were originally mined between 1892 and 1910 producing approximately 2,750t of copper. Processing of the ore was carried out using coarse grinding of the ore, hydraulic classification and separation of the copper minerals using shaking tables to produce a gravity concentrate. According to historical records this assayed in excess of 70% Cu from Lyell Blocks and closer to 60% Cu from Lyell Consols.

However a lower grade middlings product was also reported; a product that assayed about 20% Cu and which contained significant ironstone. Copper recoveries were reported at circa 70%. It is noted that the Lyell Blocks ore was higher grade than the Lyell Consols ore, a consideration which potentially contributed to the grade difference in the concentrates. The average ore grade treated in the latter stages of this early operating period averaged circa 1% Cu.

4.2 Mineralogical Assessment

Mineralogical investigations have identified that the copper is present primarily as native copper and cuprite in the upper sections of the deposit transitioning to sulphides at depth. The dominant sulphide mineral is chalcocite although minor chalcopyrite, bornite and covellite have been reported. The dominant gangue minerals include quartz, clays and haematite plus magnetite and carbonates.

A mineralogical assessment of September 2005 on samples from drill hole CMPKL02 reported the copper distribution in sequential depth intervals. From 43m to 47m depth, 91.4% of the copper was present as native copper and 4.5% as cuprite with 2.4% as chalcocite. In the 47m to 51m interval, the proportion of native copper decreased to 84.4% while the proportion of chalcocite increased to 10.3%. At 51m to 55m depth, the proportion of the copper present in chalcocite increased to 94.7% with no native copper being reported.

The mineralogical profile thus indicated dominant native copper in the upper sections of the deposit transitioning to dominant chalcocite at depth. Note however that the samples examined in this mineralogical investigation were fine sized in an approximate 38 µm to sub-10 µm size range such that there was no indication of the coarse particle liberation characteristics.

Drilling logs from earlier Lyell Blocks and Lyell Consols holes indicated a similar copper distribution profile. Typically, native copper is noted in the upper sections of holes and although sulphides are not always reported, when they are observed it is normally at depth.

The native copper was variously described as being present as pebbled or being located along joints and fracture zones. The sulphides were not always described in detail but in some instances were noted to be finely disseminated. The gangue minerals frequently reported included clays, haematite, limonite, quartzite, dolomite and limestone.

While some hard gangue components have been noted such as the ironstone, it has been reported that 75% to 80% of the deposit is present as clays in some form. It will be necessary to quantify the actual materials handling characteristics of the deposit as these factors will have a direct impact on the viability of processing options.

In conjunction with the historical processing data, some interpolation of the mineralogical information might be made in relation to potential mineral processing treatment options for the Copper Clays. It should be stressed however that these interpretations should be regarded as conceptual only and that mineralogical and metallurgical investigations will be required to properly define a process design.

- The Chilean Mill used in the historical operation would have produced a sand sized product which would have been consistent with the operation of shaking tables. The hydraulic separation sequence would have separated the copper-bearing sands from the gangue clay component.
- The production of the historical 70% Cu high grade concentrate from the shaking tables indicated a high native copper content, suggesting a reasonable liberation of the native copper at the sands sizing consistent with the mineralogical

observation that the native copper occurred as pebbles and along joints and faulting.

- The production of the low grade 20% Cu middlings component which contained ironstone conforms with the mineralogical data that noted the presence of haematite and magnetite, high SG minerals that would report to gravity concentrate with the native copper.
- The copper sulphide minerals were apparently not targeted in the historical operation and these minerals would not be amenable to gravity separation. It should be noted that the flotation technology which is generally used to recover sulphide minerals was not available at the time that the historical operation was functioning.

It is assumed that a contemporary gravity processing technique could be applied for the recovery of the native copper but that the copper sulphide mineralisation would require the application of flotation technology. The presence of the high SG gangue minerals would however lower the average grade of a gravity concentrate. It is also understood that whereas the historical operation generally treated ore in a 1% Cu to 3% Cu head grade range, the current resource may be below these limits. The lower head grade will likely produce lower recovery levels. For these reasons it is probable that a gravity circuit on the current Copper Clays will generate a lower average concentrate grade at lower copper recovery levels compared to the historical operation.

It has been estimated that for the treatment of the Copper Clays, a 65% copper recovery into a 45% Cu gravity concentrate might be anticipated in the upper sections of the deposit which contain native copper with some cuprite as the dominant copper mineralisation. It has been assumed that moderate liberation of this copper mineralisation would occur at a relatively coarse size. It has also been assumed that the chalcocite which dominates at depth would be present as locked mineral in harder gangue materials and as such would not be amenable to gravity processing. A contemporary fine grinding and flotation approach would be required for the efficient recovery of the copper sulphide minerals.

4.3 Metallurgical Test Work

To date, little relevant metallurgical test work has been carried out into the treatment characteristic of the Copper Clays. Test work conducted in 1966 considered flotation processing as the sole means of treating the Copper Clays, perhaps because this material had been considered to be a supplementary feedstock for the existing flotation plant. As might be anticipated, it was shown that the Copper Clays had to be de-slimed to remove the clay component of the ore to obtain a controllable flotation operation. Depending upon the sample tested between 15% and 50% of the copper was lost in the de-slime tail. This outcome indicates that the potential for copper loss that might ensue in an ore washing step in any contemporary process design requires investigation.

Of the de-slimed sands product however, the flotation response tended to vary with the head grade of the ore. With a high grade 2.3% Cu ore, a flotation concentrate assaying 27% Cu was obtained at 71% overall copper recovery. As the head grade decreased

to 0.9% Cu however, the concentrate grade and copper recovery level decreased to 17% Cu and 71% recovery respectively.

While processing native copper in a sulphide flotation regime is possible, the recovery efficiency is not usually high. The malleable native copper tends to form platelets during the grinding process that are inherently difficult to physically lift in the flotation operation. As such, it is usually preferable to recover native copper by gravity means. On the other hand the covellite in the ore should be quite amenable to flotation treatment.

In relation to the 1996 test work, there is no indication regarding the constitution of the samples that were tested. However as was concluded at that time, it is apparent that the use of flotation alone would not represent a viable option for the treatment of the Copper Clays. A process route should be developed that utilised the different characteristics of the copper and the gangue minerals as they occur throughout the deposit. As such, the treatment route that is applied to process ore from the upper sections of the deposit in which native copper dominates will likely be different from the optimum treatment route for the lower segments of the deposit in which chalcocite represents the dominant form of copper mineralisation.

4.4 Conceptual Process Design

It has been suggested that the dominant copper mineral in the Copper Clays is native copper which is present as a liberated component in a soft clay gangue. It has also been suggested that the gangue comprises 75% to 80% clays, this is generally supported by statements that in recent drilling exercises only about 10% of the ore was hard rock. The two other major associated gangue lithologies of limestone and detrital sediments have however been described as being generally un-mineralised.

In the 1995 review by KJA Wills entitled ‘Open Cut Potential of the Copper Clays Area’, it was proposed that the copper in the Copper Clays could be distributed as of the order of 80% in native copper with 10% as cuprite and the remainder in sulphides, particularly chalcocite. Because the cuprite is generally associated with the native copper, minerals that together account for circa 90% of the copper in the Copper Clays, it is logical that the conceptual process design focus on the recovery of these minerals in preference to the recovery of the sulphides. That is not to say however that the recovery of copper from the sulphide mineralisation should be ignored totally. It is more that it should not be the principal driver in the selection of a treatment route.

For the reasons previously mentioned, gravity processing techniques would be more appropriate to the recovery of the native copper than would flotation. The process design has therefore been based upon gravity processing.

As well, because the gangue mineralisation comprises circa 80% clays, it is suggested that conventional grinding of the ore would not be an appropriate comminution process ahead of the gravity recovery circuit. With the ore comprising circa 80% clays, it would only be the remaining circa 20% of hard rock that would be amenable to comminution by a grinding process. With minimal hard ore in the deposit, it would be unlikely that there would be sufficient rock media to render SAG milling as a suitable process and although ball milling might well be applied to this ore, the associated capital and operating cost

could not be justified. The equipment would have to be sized to be able to process all of the ore to grind only the 20% hard rock component. Any grinding mill in this case would essentially act as a very expensive slurrying machine.

To that end, it is suggested that the most appropriate treatment approach would be to utilise a trommel scrubber at the head of the circuit to wash out the clay component of the gangue and so nominally remove circa 80% of the mass as a barren gangue early in the process route. A grizzley and jaw grusher would be located ahead of the scrubber control the size of the hard rock component entering the treatment circuit.

The ore would be reclaimed by front-end loader from a ROM stockpile and dumped into a feed bin protected by a grizzley. Ore would be drawn from the feed bin and fed into the jaw crusher. The crushed product would be conveyed to a surge hopper ahead of the scrubber where water would be added and the ore washed to liberate the native copper from the clays. The scrubber discharge product would be separated into a number of size fractions for separate treatment as appropriate.

The appropriate size split for the scrubber discharge would have to be determined by relevant test work. However it might be presumed that the target feed size for the gravity circuit would be circa -20 mm + 0.5 mm. The +20 mm material which would nominally have originated from the hard rock component of the deposit would be conveyed to a coarse rejects stockpile. The copper assay of this product would be monitored and if it ensued that the copper assay was sufficiently high potentially due to the presence of a locked chalcocite fraction, then it would be trucked to the existing flotation plant to be processed through a conventional grinding and flotation sequence.

The clay material in the scrubber discharge product would potentially be separated by a combination of screens and cyclones to be discharged as a barren slimes component. This slimes tailings would be thickened to reclaim process water with the underflow from the thickener being pumped to a tailings dam for residue storage.

The sands component from the scrubber discharge product would become the feed to the gravity separation circuit. For the present exercise, it has been assumed a series of In-line Pressure Jigs would be utilised as the principal gravity recovery technique. This incorporates contemporary processing technology. The jigs would generate a high SG sinks components and a lower SG floats component. The floats would be conveyed to a gravity circuit tailings stockpile while the sinks would become the value concentrate. The floats component would likely contain the copper sulphide mineralisation when that was present in the ore. Hence this product would be analysed and as with the coarse rejects material, conveyed to the existing flotation plant for further treatment when the copper assay justified such action. When the copper assay was low, both the coarse rejects and the gravity circuit floats products would be transferred to a waste materials stockpile.

The gravity circuit concentrate would contain the native copper together with other high SG gangue minerals such as the ironstone type products. It is also anticipated that some gangue will remain locked to the native copper material. Due to dilution it has been estimated that a grade of circa 45% Cu might be anticipated in the gravity circuit concentrate. As has been previously mentioned, with the indicated copper grade of the deposit being around 1% Cu and with the copper recovery being generally influenced by

the head grade, it is anticipated that a copper recovery of circa 65% will be achieved by this process.

In relation to the disposal of the Copper Clays gravity concentrate, it has been assumed that the gravity product would be blended with the filtered concentrate from the existing flotation plant for despatch to market. In this regard however, it will also be necessary to consult with the current customers to ensure that the native copper concentrate is an amenable material for which appropriate payment will be received. There may be issues in this regard as the current flotation concentrate is processed by conventional smelting techniques that are geared to the chemical interactions between sulphide minerals and the various fluxes used in the smelting process.

4.5 Capital and Operating Cost Estimation

Preliminary capital and operating cost estimates have been generated for the indicated process design as shown in Table 4.2. Two ore treatment rates were considered, processing either 250,000 or 500,000 tonnes of ore per annum. In addition, two equipment formats were evaluated, with one scenario considering the use of all new equipment while the alternative case assumed that some refurbished second hand equipment would be incorporated in the circuit.

Table 4.2 Processing Operating and Capital Cost

		Annual Throughput (ktpa)	
		250	**500**
All New Equipment	(MA$)	8.00	13.00
Refurbished 2nd Hand Equipment	(MA$)	6.00	9.50
	(A$/t)	10.00	7.50

It should be noted that with current market conditions, the cost of refurbished second hand equipment is not significantly lower than the cost of new equipment. However it does have the advantage of a reduced delivery time. In the same vein, it should also be noted that these current market conditions mean that well maintained equipment will hold its capital value reasonably well.

4.6 Metallurgical Test work Requirements

At present, the provisional process design has been based upon a liberal interpretation of the limited available data. In order to enable a detailed process design to be carried out, it will be necessary to perform the appropriate metallurgical investigations.

The level of risk associated with the process design will be an inverse function of the extent of the metallurgical information which is available for that design. As essentially no specific Copper Clays metallurgical data exist at present, it will be necessary to generate the full spectrum of design information. As a ball park estimate, a cost of circa A$150,000 would potentially be incurred for a metallurgical test work programme (not including sample collection) at a level that would permit a feasibility standard assessment of the project to be prepared.

The test work programme would be structured to address the gravity recovery of the native copper although an assessment of the beneficiation of the copper sulphide mineralisation would also have to be considered. The test work would also incorporate a mineralogical investigation to assess the ore from a metallurgical perspective in order to identify the mineralogical associations and the likely liberation characteristics.

As suggested, test work would focus on gravity processing, primarily by laboratory investigation. However because of the difficulty in assessing such techniques on a small scale, pilot treatment of a small bulk sample may be justified towards the end of the investigation if a representative bulk sample can be readily obtained. In general however, the metallurgical investigation would have to define the relative proportions of the value mineralisation and gangue, the scrubbing characteristics of the ore and the likely constitution of the scrubbed products as well as the gravity recovery response of the sands fraction.

Although the test work would largely address the recovery of the dominant native copper mineralisation, it will also be necessary to consider the beneficiation route that might be applied to the copper sulphide minerals, particularly if this component of the ore was to be processed through the existing flotation plant. To that end it would be necessary to identify the comminution characteristics of the relative gravity circuit reject products in addition to the flotation characteristics.

Finally, as the gangue mineralisation comprises approximately 80% clays which will be discharged to a tailings dam, it will be necessary to quantify the thickening and settling characteristics of these clays in order to define an appropriate treatment route. It could well ensue that these clays are inherently difficult to settle and thicken such that specific deposition requirements and residence volumes may be required in the tailings dam to ensure that any discharge water has been effectively clarified. Considerable effort may be needed to assess the dewatering characteristics of these clays.

4.7 Tailings and Waste Disposal

Cost has been included in the operating expense for disposal of tailings, but no tails dam location has been identified. For the purpose of the study it has been assumed the tails will be disposed of in the Iron Blow open pit. If this is not the case additional cost will be incurred constructing a suitable tails dam.

Use of the Iron Blow to dispose of tailings may lead to additional liabilities being taken on by CMT above those associated with a new tails dam.

5 MINING OPTIONS AND MINING COSTS

The proposed mining fleet consists of:

- One 80t Excavator.
- Four 40t articulated six wheel drive trucks.
- One 40t tracked dozer.
- One 12t Motor grader.

Six wheel drive articulated trucks will allow operations to continue in the wet conditions regularly seen at Queenstown, which would make the site clays un-trafficable for rigid body trucks.

Due to the potential difficulties operating on narrow benches above 2,330 mRl, in the area of the existing waste dumps, an additional dozer has been assumed and mining rates reduced by 20% above 2,330 mRl.

The estimated operating cost is:

- Mining:
  - Above 2,330 mRl – A$5.00 /t of rock.
  - Below 2,330 mRl – A$3.50 /t of rock.
- Grade control – A$1.00/t ore.

Capital cost is estimated at A$1,750k, this includes de-watering the Iron Blow.

5.1 Mining Method and Equipment

The major issues influencing selection of mining method and equipment are:

- **Ground Conditions** – The rock being mined for the copper clays project will be predominantly clay. The exposed material suggests that it should be possible to excavate without blasting and with only minimal ripping of hard zones. This is supported by exploration geologists who reported that only 5% of the material provided any resistance to drilling.
- **Weather Conditions** – As discussed the area has on average close to 2,500 mm of rain per annum on approximately 200 days of the year. This coupled with the generally soft ground conditions will create difficulties in maintaining running surfaces.
- **Project Size** – Initial analysis suggested the project will have in the region of 5 Mt of rock being mined over a period of three to four years. This will give annual mining rates of between 1.2 Mtpa and 2 Mtpa.

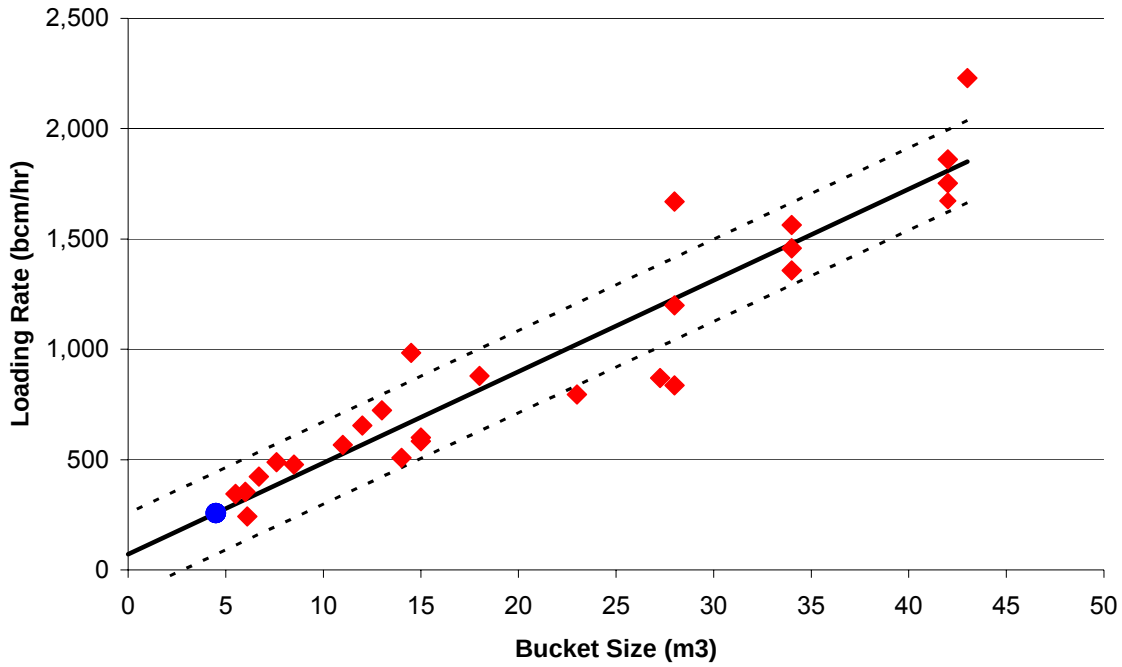
While the ground and weather conditions will make operations difficult at times, and may preclude using rigid body trucks, AMC believes it will be possible to mine the deposit using an excavator and six wheel drive articulated trucks. Due to the poor ground conditions it has been assumed a tracked dozer will be required full time to rip where

required and maintain floors around the digging area. A grader has also been assumed to be required full time to maintain running surfaces.

Assuming an annual production rate of 1.5 Mtpa, this will only require a single excavator. Single shift operations for eleven hrs per day, six days per week and forty eight weeks per year, this gives 2,970 engine hours per annum, if the machine operates for 90% of this time the total effective operating hours will be 2,673 hours per annum. Assuming a 2.2t /bcm bulk density, achieving 1.5 Mtpa will require a rate of 255 bcm/hr.

The graph in Figure 5.1 compares the volumetric loading rate to bucket size for the AMC open pit benchmarking database. The point in blue is a 4.5 m^3 bucket which gives a loading rate of 260 bcm/hr. The volumetric loading rate has been selected as the low density will lead to the bucket being full with material before the machine lifting capacity has been reached.

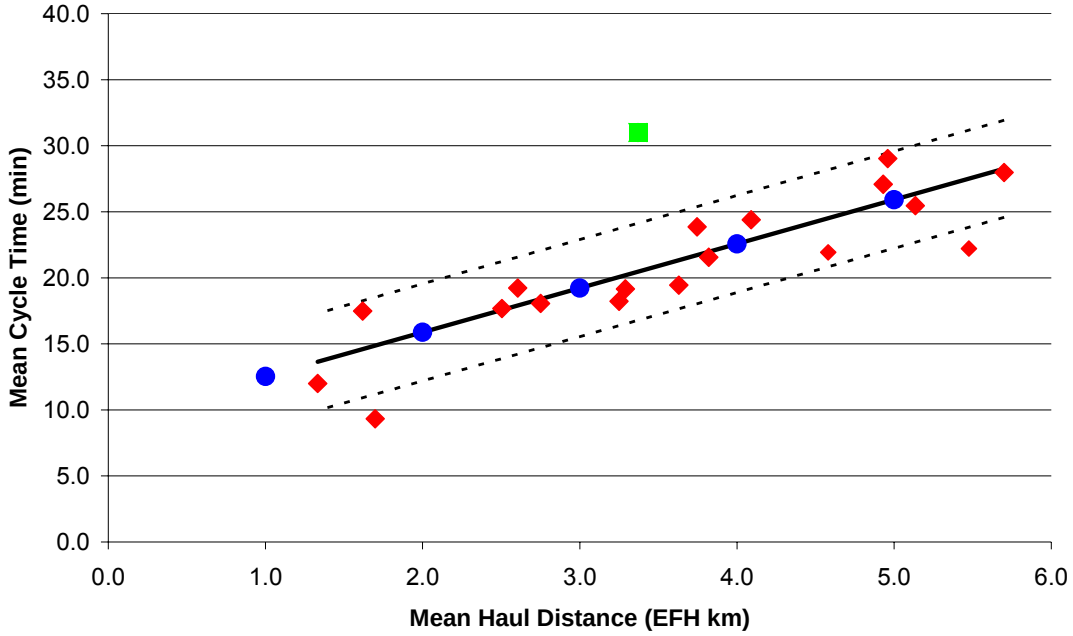
Figure 5.1 Loading Rate vs. Bucket Size – AMC Benchmarking Database



This is the size of bucket normally associated with an 80t excavator such as a Komatsu PC800.

An excavator of this size would be appropriate to load 40t articulated trucks. The graph in Figure 5.2 compares truck cycle time to haul distances for the AMC benchmarking database.

Figure 5.2 Truck Cycle Times vs. Haul Distance – AMC Benchmarking Database



The results of applying these cycle times to a CAT 740 truck with a payload limit of 38t and a heaped capacity of 23 m^3 are shown in Table 5.1.

Table 5.1 Haulage Productivity

Haul Distance	Cycle Time (min)	Productivity (t/op. hr)	Number of Trucks (#)	Total Production (kt)
1 km	12.5	488	3	1,463
2 km	15.9	383	3	1,150
3 km	19.2	317	4	1,270
4 km	22.6	270	5	1,348
5 km	25.9	235	6	1,412

This suggests a fleet of one 80t excavator four 40t articulated trucks will deliver the appropriate quantities of material. For further work it has been assumed this fleet will mine 1.5 Mtpa.

As discussed it has been assumed a tracked dozer and grader will be required full time to support the mining fleet. A 40t dozer such as CAT D8 would be suitable to manage the floors and have sufficient power for any ripping required. A 12t motor grader such as a CAT 120 would be appropriate to manage the running surfaces.

5.2 Mining Costs

5.2.1 Operating Cost

Hourly hire rates (including operator and fuel) for the type of equipment required are based on what is presently being charged on the West coast of Tasmania. The rates are shown in Table 5.2.

Table 5.2 Hourly Equipment Operating Cost

Equipment	Rate ($/eng. Hr.)
Komatsu PC800 Excavator	340
CAT 740 Articulated Truck	220
CAT D8 Tracked Dozer	267
CAT 120 Grader	190
CAT 980 FEL	190

Assuming a fleet of four trucks, and one excavator, dozer and grader are working 2,970 hrs pa, the total cost will be A$5.0 Mpa. With a mining rate of 1.5 Mtpa, the unit cost will be A$3.33 /t mined.

In the upper benches, above 2,330 mRl, due to the mining clays and the waste dumps additional ancillary cost will be incurred. It has been assumed an additional dozer will be required and the mining rate has been reduced by 20%. This makes the total cost A$5.8M for 1.2 Mt, giving a unit cost of A$4.81.

It is not possible to estimate accurately the quantity of material in the waste dumps, but AMC believe it is unlikely to be more than 10% of the material above 2,330 mRl, this is approximately 100,000 m^3.

Additional to this site will require geology and mining supervision, assuming this is the equivalent of one full time person, allowing for a vehicle and on costs this will cost approximately A$250 kpa, or A$0.17 /t.

This gives a total rock mining cost of A$3.50 /t below 2,330 mRl and A$5.00 /t above 2,330 mRl.

Due to the low strength of the clays being mined no drill and blast cost has been assumed. If D&B is required the geotechnical assessment suggests this will be isolated areas requiring paddock blasting. The cost of this is likely to be around A$0.50 /t for a maximum of 20% of the pit. This would equate to an average additional cost of A$0.10 /t. Sensitivity analysis suggests this will not be sufficient to influence the decision to proceed further with the pre-feasibility process.

No additional cost has been included for sheeting. AMC believe that despite excavating a pit in clays, the six wheel drive articulated trucks will operate efficiently with minimal sheeting supplied from material selectively stockpiled from mining the Iron Blow waste dumps. This is based on experience of seeing similar trucks operating in wet clays in

Laos during the wet season. Should sheeting be required it will be for a maximum of 20 benches, assuming a 200m length, 15m wide and 1m thick, this will be a total of 60,000 m^3 of sheeting. If this were to cost A$20 /$m^3$ the total cost would equate to an additional A$0.20 /t. This has been included in the cost sensitivity.

Grade control drilling is estimated to cost ~$60.00 /m and assays presently cost A$13.00 /sample. Hence a total cost has been estimated at A$75.00 /m. Assuming a 10m x 5m pattern, that 70% of the drilling will be in ore and a bulk density of 2.2 t/bcm, grade control cost will be:

$75.00 / (50 m^3 x 2.2 x 70%) = A$1.00 /t of ore

5.2.2 Capital Cost

The mining capital estimate is A$1.75M. This includes A$1.0M for treating 500 kt of water pumped from the Iron Blow at a cost of A$2/t (provided by CMT), and A$750k for general site set up. The items included are shown in Table 5.2.

Table 5.3 Capital Cost

Item	Cost (k$)
Site Access Road	25
Vehicle Washdown	25
Contractor Mobilisation	50
Workshop	50
Hardstand	50
Office and Crib Room	50
Dewatering	100
Power Reticulation	50
IT and Coms	20
Security and Safety	50
Water Reticulation	50
Sediment Traps	25
Fuel Storage	50
Contingency (30%)	179
TOTAL	**774**

5.3.3 Site Rehabilitation Cost

Mine site rehabilitation has been estimated at A$750k, this includes:

- 1,200 hrs of equipment hire at A$250 /hr to A$300k.
- 4 km of fence at A$50 /m to A$200k.
- Revegetation and other activities to A$250k.

It has been assumed the plant will have salvage value of 50% of its capital cost of A$6M, of this A$1.5m will be required for deconstruction, transport and site rehabilitation, leaving a surplus of A$1.5M.

5.3 Dilution and Mining Recovery

The mineralised zone is reasonably wide, ~40m, and the material on the boundary contains some copper so dilutant material will generally have some grade. Hence only minimal dilution, 5% with 95% recovery, has been assumed. The dilutant has been assumed to have no grade.

6 LG OPTIMISATION

Lersch-Grossman Optimisation (LG) identified potential for a pit with approximately 900 kt to 1,000 kt of ore at a grade of 1.9% Cu with a strip ratio of around 6.0 t:t. This would provide between 10 kt and 11 kt of recovered Cu.

Optimisation was performed using the Whittle programming 4X implementation of the Lersch-Grossman open pit optimisation algorithm.

6.1 LG Inputs and Cut off Grades

The parameters used for the LG are shown in Table 6.1.

Table 6.1 LG Optimisation Parameters

Costs		
Mining Rock		
Above 2,330mRl	$5.00	/t rock
Below 2,330mRl	$3.50	/t rock
Treatment	$10.00	/t ore
Grade Control & Geology	$1.00	/t ore
Administration	$1.00	/t ore
Mining and Treatment Factors		
Dilution	5.0%	
Mining Recovery	95.0%	
Metallurgical Recovery	65.0%	
Concentrate Grade	45.0%	Cu
Copper Price		
High Sensitivity	US$8,000	/t
Base Case	US$6,700	/t
Low Sensitivity	US$5,000	/t
Very Low Sensitivity	US$2,000	/t
Exchange Rate	0.86	US$/A$
Discounting Factors		
Cost of capital	8.0%	
Treatment rate	250.00	ktpa
Royalty	5.0%	of net revenue
Realisation Costs		
Transport	US$150	/t concentrate
Treatment Charge	US$45	/t concentrate
Refining Charge	US$0.05	/lb Cu
Consolidated	US$533	/t Cu in conc.
Payable Metal	96.6%	Cu in conc.

The mining, treatment and grade control costs have been discussed earlier. An administration cost of A$1.00/t of ore has been assumed to allow for any additional administration required above the present CMT operation.

The mining and treatment factors were discussed under the relevant sections.

Copper prices were selected in discussion with CMT to cover a range of potential scenarios and test the sensitivity of the project to factors which influence revenue.

Realisation costs were provided by CMT and the royalty was taken from the recent benchmarking study.

The LG input parameters give the cut off grades shown in Table 6.2.

Table 6.2 Cut off Grade

Cut off Grade			
Copper Net Revenue			
	High Sensitivity	$7,948	/t
	Base Case	$6,561	/t
	Low Sensitivity	$4,747	/t
	Very Low Sensitivity	$1,546	/t
Ore Costs		$12.00	/t ore
Mill Feed COG			
	High Sensitivity	0.23%	
	Base Case	0.28%	
	Low Sensitivity	0.39%	
	Very Low Sensitivity	1.19%	
Mining COG			
	High Sensitivity	0.24%	
	Base Case	0.30%	
	Low Sensitivity	0.41%	
	Very Low Sensitivity	1.25%	

The COG for the very low sensitivity at US$2,000 /t gets close to the head grade of the deposit at ~1.5% Cu. The large jump in COG between US$5,000 /t and US$2,000 /t is due to a significant quantity of the realisation costs being fixed per tonne of contained copper regardless of the commodity price.

6.2 LG Slope Parameters

The slope parameters used in the LG optimisation are based on the geotechnical analysis described in Section 3. Initial optimisation and design work indicated a relatively high wall; over 100m in places, to the north of the pit would be formed in relatively competent rock. This area was given an inter-ramp wall angle of 45°. The southern wall exposed two different geological domains; below 2,280 mRl the wall will be excavated in oxidised limestone. This is considered to be free draining and of medium strength and

an inter-ramp wall angle of 37° was applied. Above 2,280 mRl the wall will be excavated in highly weathered clays which will not be free draining; an inter-ramp angle of 27° was applied in this area.

6.3 LG Optimisation Results

LG optimisations were run generating a suite of shells ranging in size from 1.5 Mt of rock to 22 Mt. The contents were then classified as mill feed and waste, and operating cash surpluses estimated based on the parameters in Table 6.1. The results of this are shown in Figure 6.1 and 6.2.

Figure 6.1 LG Optimisation Total Pit Contents vs. Recovered Copper

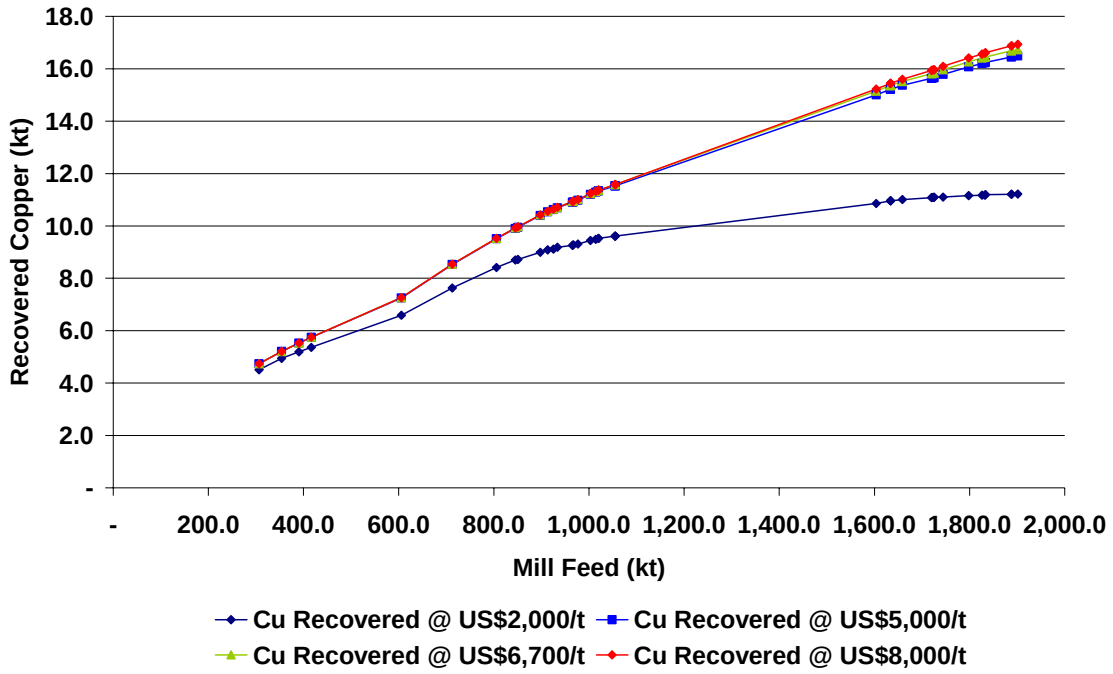
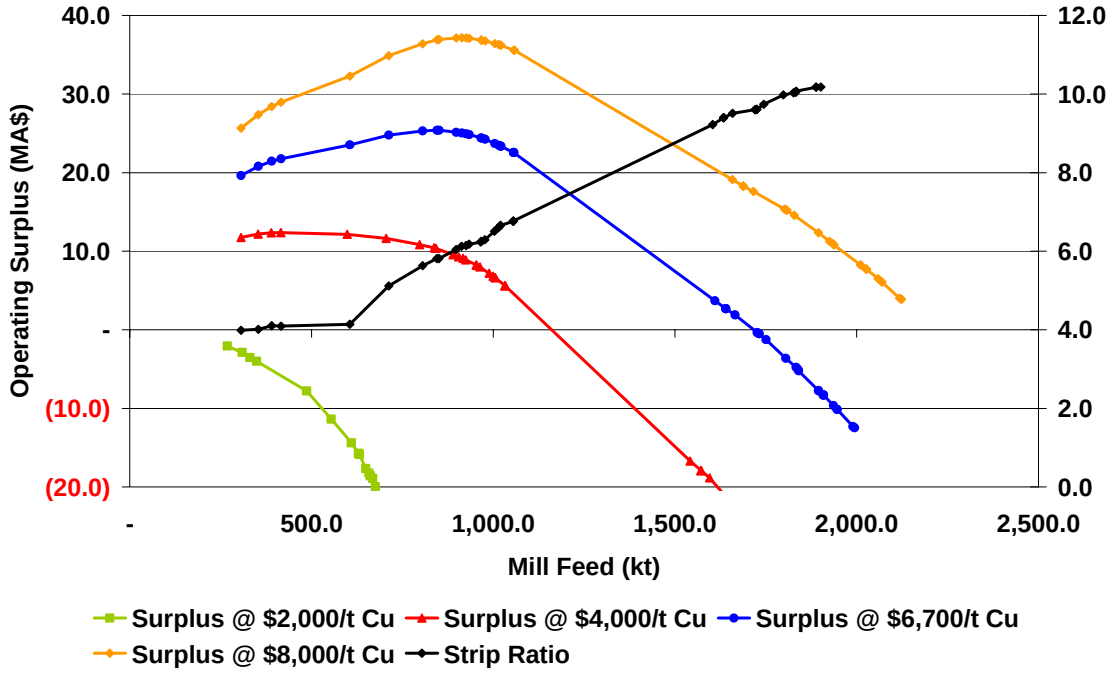


Figure 6.2 LG Optimisation Total Mill Feed vs. Operating Cash Surplus



The graph in Figure 6.1 shows how the shells grow from 300 kt of mill feed to 1,000 kt, steadily adding recovered metal. The slight decrease in the gradient of the line as the pit size increases indicates the material being added is of reducing grade.

Above 1,100 kt the shells jump in size to over 1,500 kt of mill feed, the gradient drops further and the lines begin to diverge. This indicates there is a break in the geological continuity and the shells have to increase in size significantly to add more mill feed. The drop in gradient and lines diverging indicates the material being added is of lower grade and getting close to the cut off grade. The contained metal line for the US$2,000 /t case is much lower than the other three cases due to the COG increasing significantly transferring more of the mineralisation to waste.

Figure 6.2 shows how the operating surplus with copper prices greater than US$5,000 /t grows up to shells with between 900 kt and 1,000 kt of mill feed. Above this the surplus starts to drop, then falls dramatically when the shells jump in size from 1,100 kt of mill feed to more than 1,500 kt. For the very low sensitivity case none of the shells generate a positive discounted surplus.

The flat strip ratio line up between 300 kt and 650 kt indicates the incremental material being added has a similar strip ratio but decreasing grade, above 650 kt the strip ratio begins to increase, implying better grade ore is being added to the shells.

A shell with between 900 kt and 1,000 kt of mill feed with a strip ratio of a little over 6:1 will give close to the optimal outcome for copper prices between US$5,000 /t and US$8.000 /t. This was selected as the basis for detailed design as it provides the optimal outcome at the base case and offers significant upside if the present prices are

maintained. Also, it would be difficult to justify the capital required based on a mill feed quantity less the ~1 Mt as it would not provide a long enough project life.

7 CONCEPTUAL OPEN PIT DESIGN, WASTE DUMPS AND SCHEDULING

The conceptual design has contents which very closely replicate the LG optimisation. This validates the optimisation as a good estimate of the potential mill feed quantity, head grade and strip ratio, when operational and access constraints are applied.

The schedule highlights the need for a twelve month pre-stripping period followed by approximately four years of treatment giving a project life of five years.

7.1 Design Parameters

The design has been based on a fleet of 40t six wheel drive articulated trucks. This equipment has better climbing capability than standard rigid body trucks allowing 1:7 gradient ramps to be used. The ramps are single lane, 10m wide, with potential for a passing bay approximately half way down the pit. This is acceptable as less than half of the total pit contents are below the start of the ramp, and by the time this section of the pit is being mined production will be down to a single fleet as all the benches have significant quantities of ore.

Working areas have been designed at no less than 10m wide.

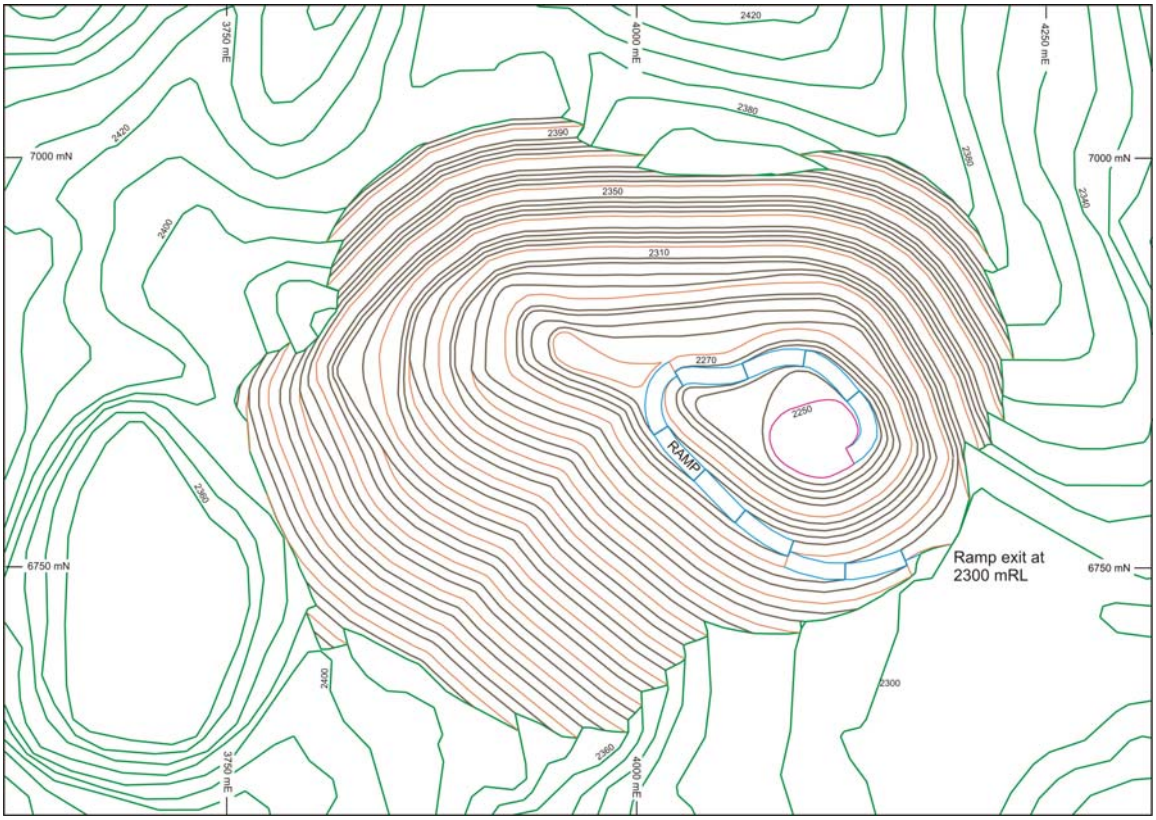
The slope batter configurations have been designed to honour the inter-ramp angle used in the LG optimisation:

- **North Wall** – 20m batters at 60° with 7m wide berms.
- **South Wall below 2,290 mRl** – 20m batters at 50° with 7m wide berms.
- **South Wall above 2,290 mRl** – 10m batters at 40° with 7m berms.

7.2 Conceptual Pit Design

The conceptual pit design is shown in Figure 7.1.

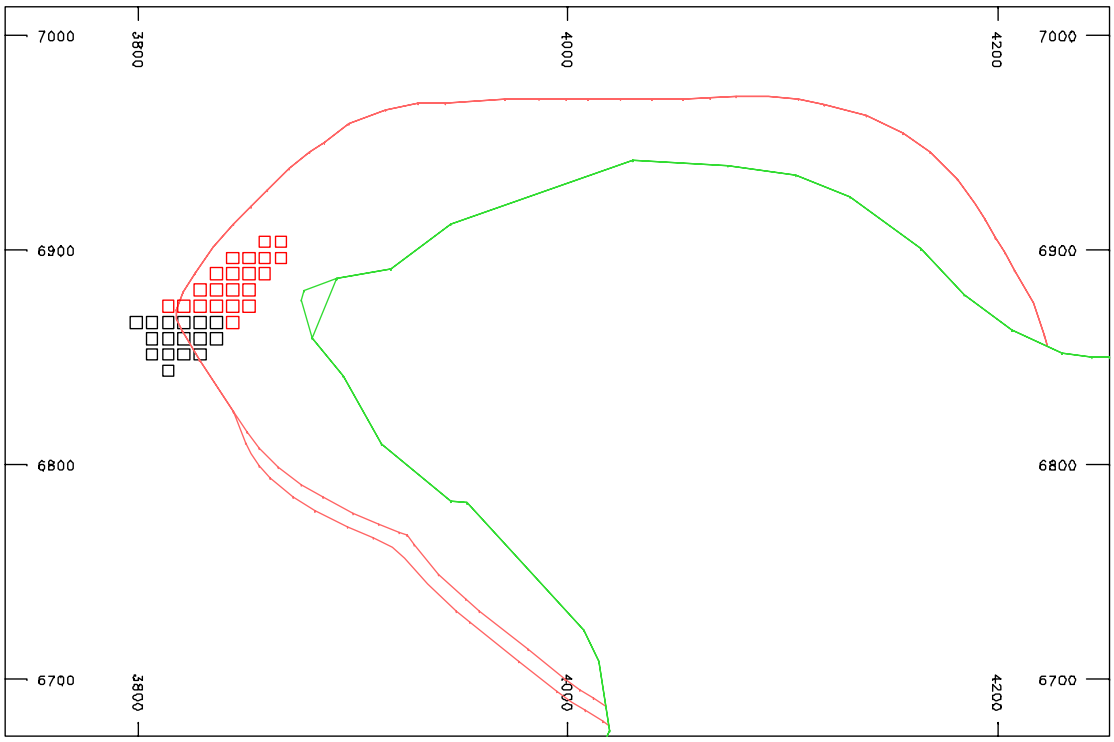
Figure 7.1 Conceptual Pit Design



The ramp has been designed to exit the pit at the lowest point in the topography 2,305 mRl. For the initial 30m vertically the ramp runs along the southern wall, in the hanging wall of the deposit. At approximately 2,270 mRl the ramp switches to the north side of the deposit and runs down the footwall. As the deposit is dipping relatively gently in this area, around 25° the ramp can be accommodated without incurring additional waste movements.

Between 2,305 mRl and 2,360 mRl benches are being mined which run the round the full head of the valley. This is illustrated in Figure 7.2.

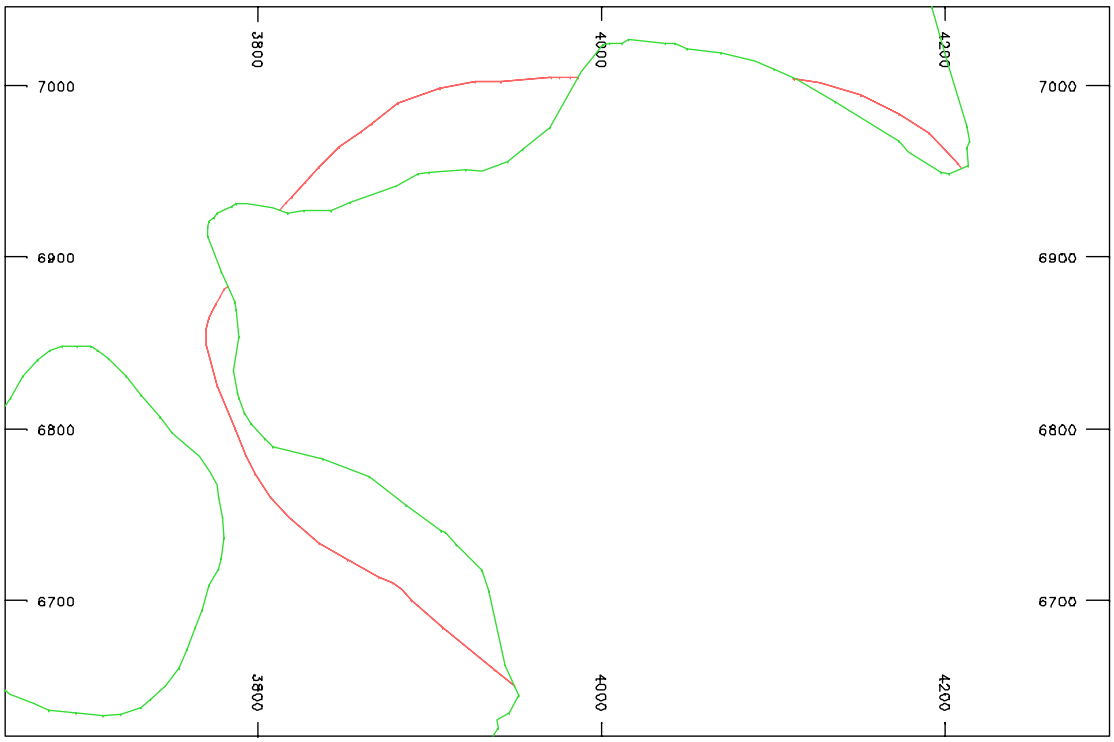
Figure 7.2 Conceptual Pit and Topography 2,340 mRl



In this region it should be possible to develop temporary ramps to access the benches and have two fleets working if required.

Above 2,360 mRl the benches break into distinct sections, this is shown in Figure 7.3. It will not be reasonable to access these with ramps, and the material between 2,360 mRl and 2,400 mRl will require significant dozing to make it accessible with an excavator. Between 2,365 mRl and 2,400 mRl there is around 400 kt of rock.

Figure 7.3 Conceptual Pit and Topography 2,375 mRl



7.3 Conceptual Design Contents

At a COG of 0.5% copper the conceptual design contains 6.4 Mt of rock, of which 950 kt is mill feed at a grade of 1.8% Cu containing 16.6 kt of copper and providing 10.8 kt of recovered copper. This gives a strip ratio of 5.9:1.

The detailed contents are shown in Table 7.1.

Table 7.1 Conceptual Design Contents

Level	Rock	Mill Feed			
mRl	kt	kt	% Cu	t Cu	t Cu Rec
2,390	0	-	-	-	-
2,385	14	-	-	-	-
2,380	59	-	-	-	-
2,375	125	-	-	-	-
2,370	93	-	-	-	-
2,365	176	-	-	-	-
2,360	132	-	-	-	-
2,355	249	-	-	-	-
2,350	143	0	0.8%	2	2
2,345	239	5	1.3%	62	40
2,340	238	10	1.3%	132	86
2,335	403	18	1.2%	221	144
2,330	219	13	1.1%	150	98
2,325	436	20	1.1%	224	146
2,320	230	20	1.4%	283	184
2,315	559	55	1.6%	870	565
2,310	208	43	1.8%	779	507
2,305	753	103	1.9%	2,001	1,301
2,300	203	54	1.5%	836	544
2,295	645	101	2.0%	2,007	1,305
2,290	140	57	2.7%	1,512	983
2,285	434	90	1.8%	1,597	1,038
2,280	84	40	2.2%	866	563
2,275	272	94	1.4%	1,275	829
2,270	96	58	1.6%	936	609
2,265	122	66	1.4%	915	595
2,260	49	36	1.5%	552	359
2,255	65	52	2.3%	1,187	771
2,250	16	13	1.7%	221	144
Total	**6,403**	**948**	**1.8%**	**16,631**	**10,810**

These contents are compared to the LG optimisation results in Figures 7.4 and 7.5.

Figure 7.4 LG Optimisation and Mill Feed vs. Recovered Copper

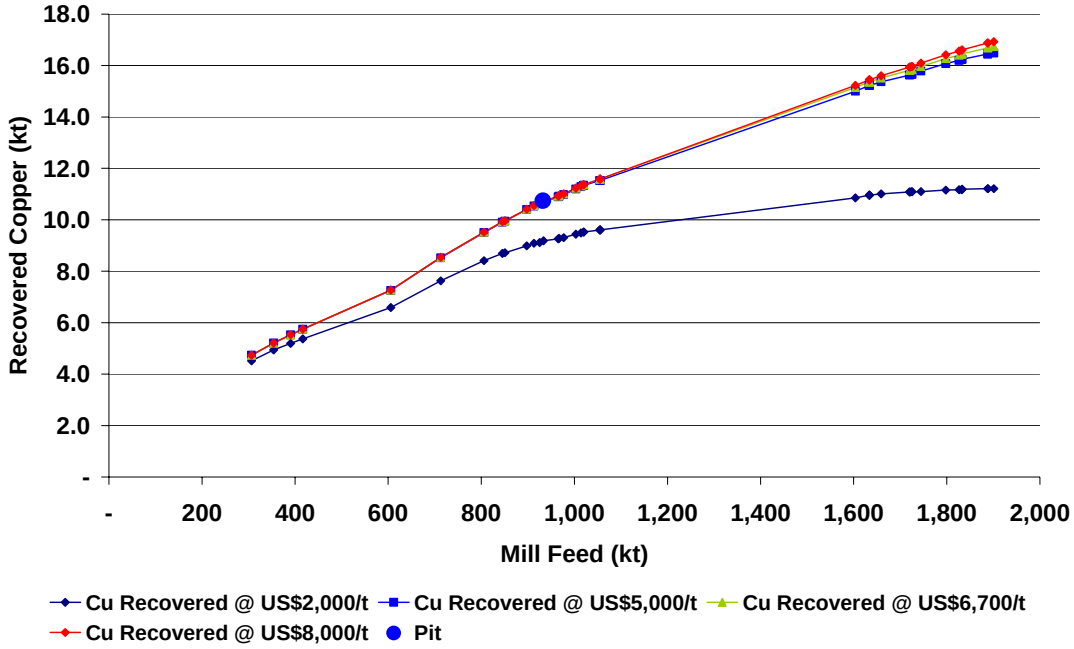
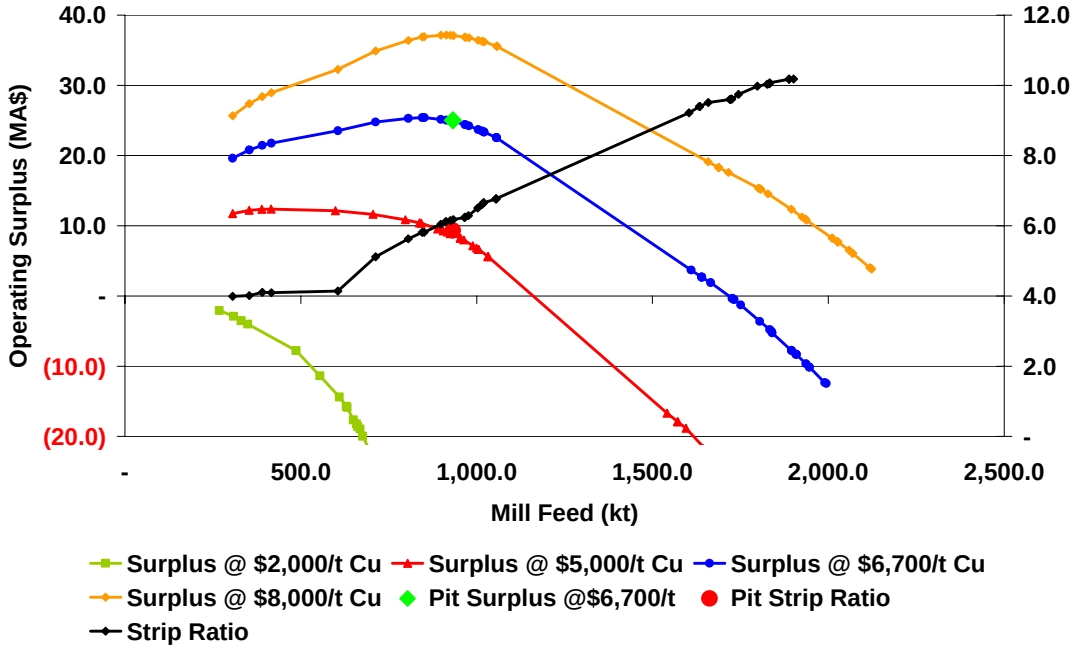


Figure 7.5 LG Optimisation and Pit Total Mill Feed vs. Operating Cash Surplus and Strip Ratio



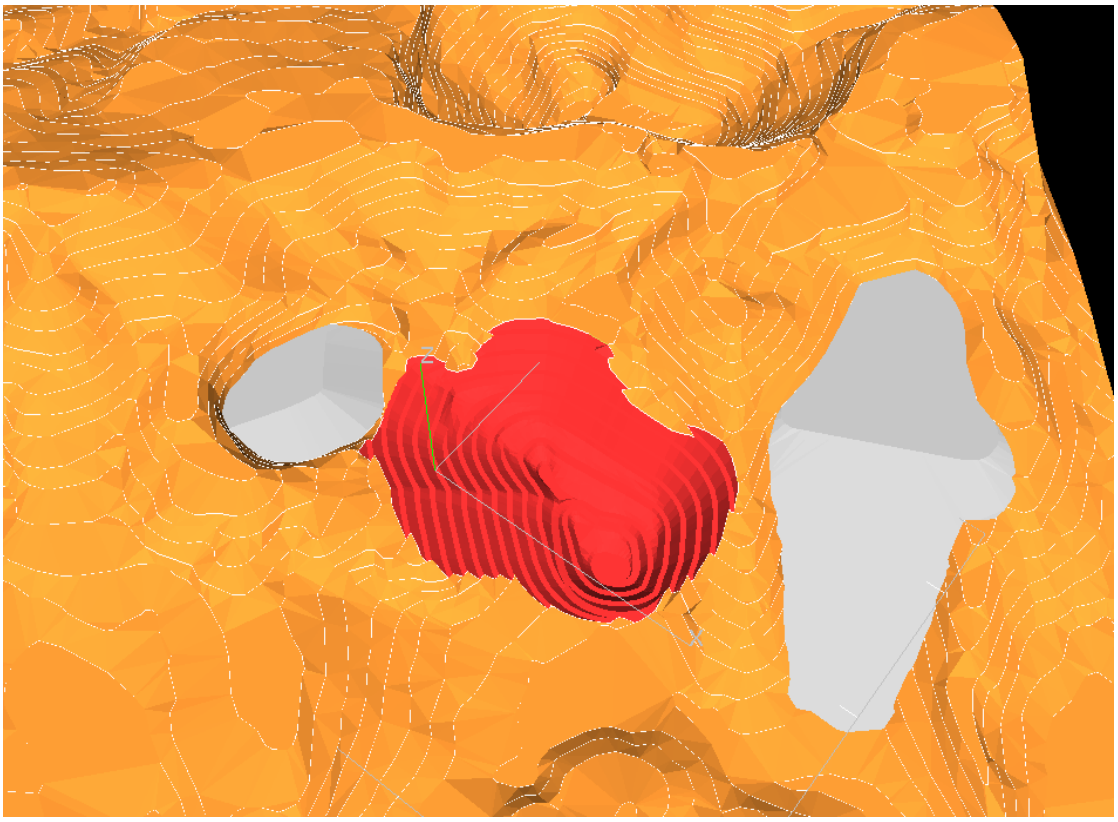
The mill feed provided by the pit design leads to a similar quantity of recovered copper to the optimisation runs, a strip ratio slightly below optimisation shells and a surplus in line with the optimisation shells. This indicates that the design has closely replicated the

optimisation. This validates the results of the optimisation as good estimation of the potential of the copper clays.

7.4 Waste Disposal

Waste disposal is restricted by Lyell Highway to the south and the proximity to Gormanston. Conceptual designs have been prepared to assess the volume of the Iron Blow open pit and valley to the east of the deposit. These designs are shown in the isometric view in Figure 7.6.

Figure 7.6 Isometric View of Conceptual Pit and Waste Dumps



Allowing for a 30% swell the capacities are:

- **Iron Blow Open Pit** – 1,400,000 m^3 or 1,000,000 bcm (assumed an additional 40m available below the present water surface).
- **Valley Dump** – 2.7 Mm^3 or 2.1 Mbcm.

As the design will generate ~2.5 Mbcm of waste these dumps would have sufficient volume, including disposal of the tails in the Iron Blow.

The major issues with these locations would be:

- The Iron blow open pit has heritage values associated with it and back filling the pit may not be acceptable.

- The east valley has recognised copper clays style mineralisation. Although this is generally considered to be low grade and sub-economic. Further more detailed sterilisation drilling and analysis would be required prior to constructing the dump.

Another option would be to use the old Mt Lyell pit. This would have operational issues as the underground cave is making parts of the pit unstable, and the tailings could bleed through to the underground operation. Also there would be potential for resource sterilisation.

7.5 Schedule

A schedule has been produced to indicate the potential life of the project and highlight any required pre-stripping.

The two major constraints applied for scheduling were:

- **Mining Rate** – Maximum of 3 Mtpa has been assumed. This equates to two fleets of 80t excavator and 40t trucks. As the estimate was based on a single shift it is possible this could be achieved by running a twenty four hour operation. This may be a preferable option as the high rate would only be required for a relatively short period.
- **Treatment Rate** – Once treatment commences a rate of 250 ktpa has been maintained.

Figure 7.7 shows the mining rate split by waste and mill feed, and Figure 7.8 shows the mill treatment rate and ROM stock levels.

Figure 7.7 Schedule – Waste and Mill Feed Mining

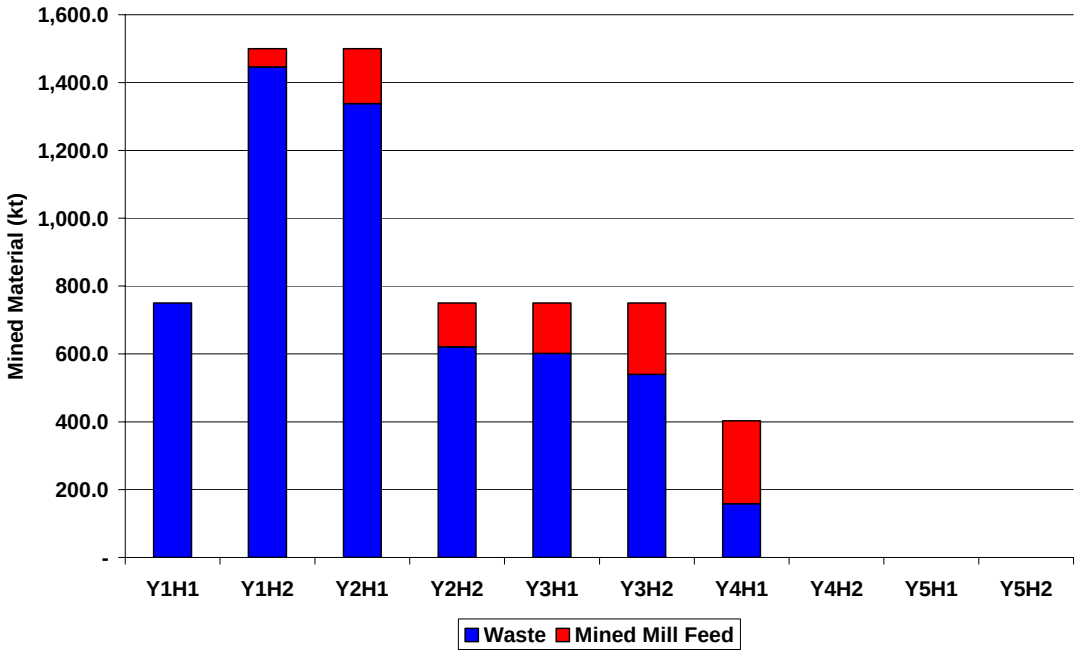
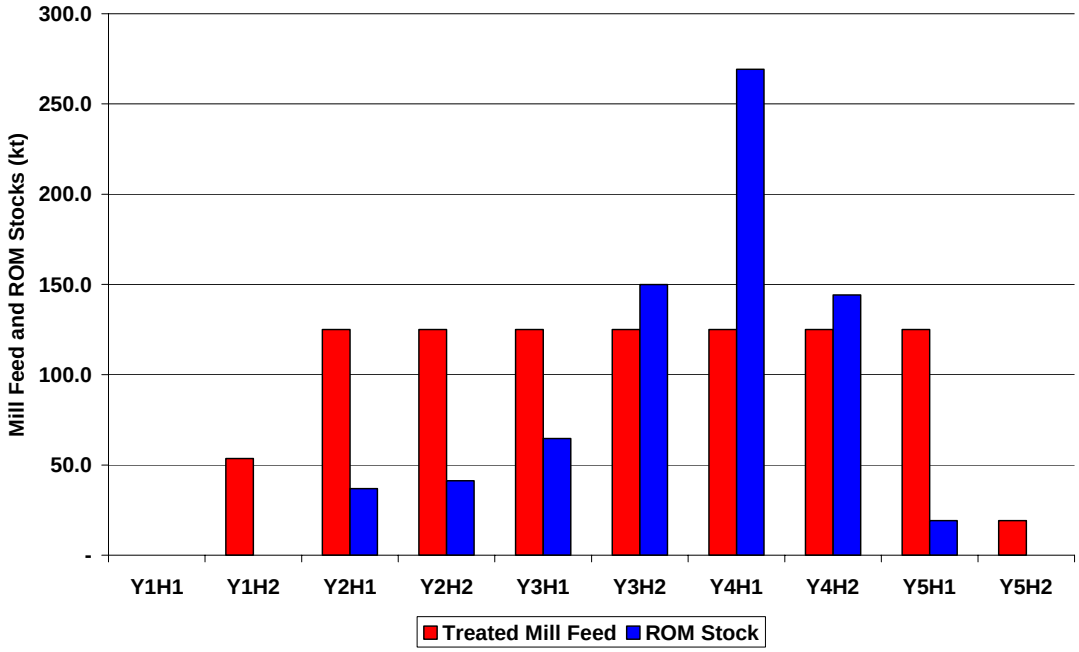


Figure 7.8 Schedule – Mill Feed Treatment and ROM Stocks



A period of almost twelve months of mining is required prior to treatment commencing. This is stripping the sides of the hills at the head of the valley to access the mill feed in the valley floor. The mining rate has been kept low for the initial six months to account for the potential difficulties of mining the upper benches.

After a year of mining at 3 Mtpa to access the mill feed, the mining has been maintained at 1.5 Mtpa, a single fleet of equipment operating on a single shift. Regardless of this low mining rate there is a build up of in excess of twelve months of mill feed. There may be potential to slow the rate further but this will most likely increase the unit cost and increase the risk of losing ore if there is a wall failure.

In the feasibility study there will be potential to optimise the mining and treatment rates. There is no value in doing this presently due to the low level of accuracy of the mineralisation model and treatment parameters.

The schedule highlights the need for a twelve month pre-stripping period followed by approximately four years of treatment giving a project life of five years.

The detail of the schedule is shown in Table 7.2.

Table 7.2 Schedule

Period		TOTAL	Y1H1	Y1H2	Y2H1	Y2H2	Y3H1	Y3H2	Y4H1	Y4H2	Y5H1	Y5H2	Y6H1
Physicals													
Mining													
Total Movements	kt	6,402.8	750.0	1,500.0	1,500.0	750.0	750.0	750.0	402.8	-	-	-	-
Ore Mined													
Tonnes	kt	947.7	-	53.5	161.9	129.3	148.4	210.3	244.3	-	-	-	-
Grade	%	1.8%	-	1.2%	1.7%	1.8%	2.2%	1.7%	1.7%	-	-	-	-
Metal	t	16,631	-	650	2,674	2,299	3,295	3,645	4,069	-	-	-	-
ROM Stocks													
Tonnes	kt		-	-	36.9	41.2	64.6	149.9	269.2	144.2	19.2	-	-
Grade	%		-	-	1.7%	1.8%	2.2%	1.7%	1.7%	1.7%	1.7%	-	-
Metal	t		-	-	610	733	1,435	2,598	4,500	2,411	321	-	-
Mill Feed													
Tonnes	kt	947.7	-	53.5	125.0	125.0	125.0	125.0	125.0	125.0	125.0	19.2	-
Grade	%	1.8%	-	1.2%	1.7%	1.7%	2.1%	2.0%	1.7%	1.7%	1.7%	1.7%	-
Metal	t	16,631	-	650	2,065	2,176	2,593	2,481	2,167	2,090	2,090	321	-
Mill Production													
Concentrate	t	24,023	-	939	2,982	3,143	3,745	3,584	3,130	3,018	3,018	464	-
Grade	%	45.0%	-	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	45.0%	-
Metal	t	10,810	-	423	1,342	1,414	1,685	1,613	1,408	1,358	1,358	209	-

8 FINANCIAL ANALYSIS

The cashflow schedule generated a pre-tax $NPV_{8\%}$ of more than A$30M for the base case copper price of US$6,500 /t. The $NPV_{8\%}$ remains positive for all commodity prices tested greater than US$3,500 /t. The $NPV_{8\%}$ has low sensitivity to changes in cost with the base case NPV changing by ~10% for a 20% change in operating cost.

Cashflow was scheduled based on the physical schedule and the financial and physical inputs shown in Table 8.1. These are the same as the LG optimisation inputs.

Table 8.1 Physical and Financial Parameters

Costs		
Mining Rock		
Above 2,330 mRl	$5.00	/t rock
Below 2,330 mRl	$3.50	/t rock
Treatment	$10.00	/t ore
Grade Control & Geology	$1.00	/t ore
Administration	$1.00	/t ore
Mining and Treatment Factors		
Dilution	5.0%	
Mining Recovery	95.0%	
Metallurgical Recovery	65.0%	
Concentrate Grade	45.0%	Cu
Copper Price		
High Sensitivity	US$8,000	/t
Base Case	US$6,700	/t
Low Sensitivity	US$5,000	/t
Very Low Sensitivity	US$2,000	/t
Exchange Rate	0.86	US$/A$
Discounting Factors		
Cost of capital	8.0%	
Treatment rate	250.00	ktpa
Royalty	5.0%	of net revenue
Realisation Costs		
Transport	US$150	/t concentrate
Treatment Charge	US$45	/t concentrate
Refining Charge	US$0.05	/lb Cu
Consolidated	US$533	/t Cu in conc.
Payable Metal	96.6%	Cu in conc.

Capital allowed for includes:

- Construction of a treatment plant – A$6M.
- Dewatering Iron Blow, mobilisation of mining fleet and set up of site – A$1,750k.
- Site rehabilitation – A$750k.
- Sale of the plant – A$1.5M credit.

Table 8.2 shows the cashflow schedule for the project.

Table 8.2 Cashflow Schedule

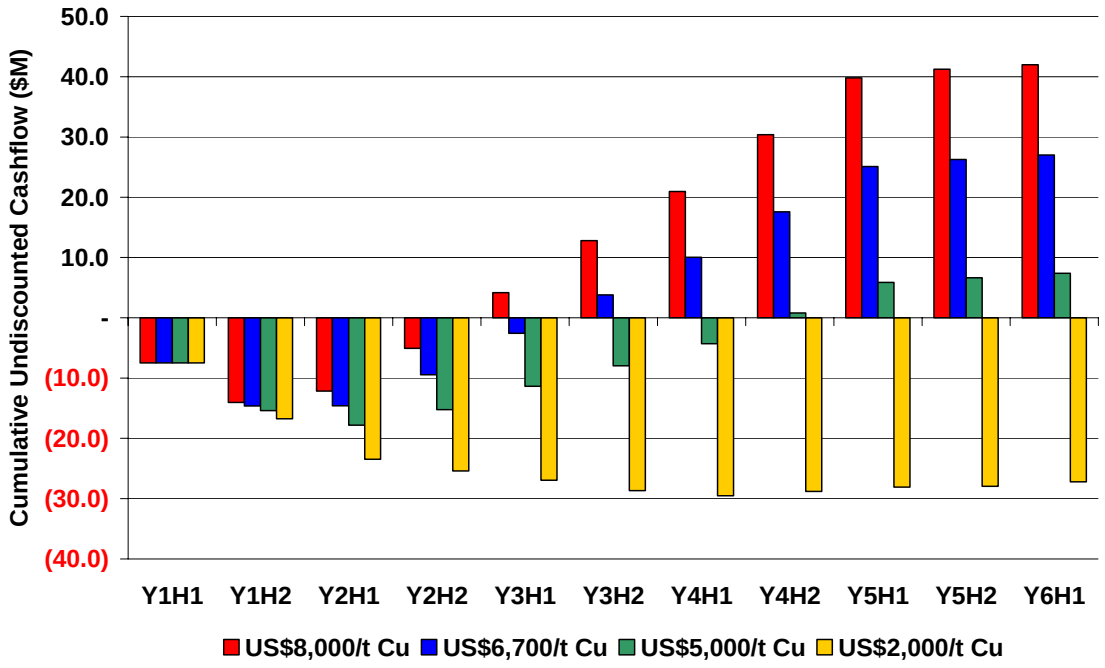
Period		TOTAL	Y1H1	Y1H2	Y2H1	Y2H2	Y3H1	Y3H2	Y4H1	Y4H2	Y5H1	Y5H2	Y6H1
Revenue													
Gross Revenue (On Payable Metal)													
US$8,000/t Cu	MUS$	83.5	-	3.3	10.4	10.9	13.0	12.5	10.9	10.5	10.5	1.6	-
US$6,700/t Cu	MUS$	70.0	-	2.7	8.7	9.2	10.9	10.4	9.1	8.8	8.8	1.4	-
US$5,000/t Cu	MUS$	52.2	-	2.0	6.5	6.8	8.1	7.8	6.8	6.6	6.6	1.0	-
US$2,000/t Cu	MUS$	20.9	-	0.8	2.6	2.7	3.3	3.1	2.7	2.6	2.6	0.4	-
Realisation Costs													
Treatment Cost	MUS$	1.1	-	0.0	0.1	0.1	0.2	0.2	0.1	0.1	0.1	0.0	-
Refining Charge	MUS$	1.1	-	0.0	0.1	0.1	0.2	0.2	0.1	0.1	0.1	0.0	-
Transport	MUS$	3.6	-	0.1	0.4	0.5	0.6	0.5	0.5	0.5	0.5	0.1	-
TOTAL	**MUS$**	**5.8**	**-**	**0.2**	**0.7**	**0.8**	**0.9**	**0.9**	**0.7**	**0.7**	**0.7**	**0.1**	**-**
Exchange Rate			0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86	0.86
Net Revenue													
US$8,000/t Cu	MA$	90.4	-	3.5	11.2	11.8	14.1	13.5	11.8	11.4	11.4	1.7	-
US$6,700/t Cu	MA$	74.7	-	2.9	9.3	9.8	11.6	11.1	9.7	9.4	9.4	1.4	-
US$5,000/t Cu	MA$	54.0	-	2.1	6.7	7.1	8.4	8.1	7.0	6.8	6.8	1.0	-
US$2,000/t Cu	MA$	17.6	-	0.7	2.2	2.3	2.7	2.6	2.3	2.2	2.2	0.3	-
Costs													
Royalty (5.00% of Net Revenue)													
US$8,000/t Cu	MA$		-	0.2	0.6	0.6	0.7	0.7	0.6	0.6	0.6	0.1	-
US$6,700/t Cu	MA$		-	0.1	0.5	0.5	0.6	0.6	0.5	0.5	0.5	0.1	-
US$5,000/t Cu	MA$		-	0.1	0.3	0.4	0.4	0.4	0.4	0.3	0.3	0.1	-
US$2,000/t Cu	MA$		-	0.0	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-
Operating													
Mining	MA$	25.5	3.8	7.3	5.3	2.6	2.6	2.6	1.4	-	-	-	-
Grade Control	MA$	0.9	-	0.1	0.2	0.1	0.1	0.2	0.2	-	-	-	-
Treatment	MA$	9.5	-	0.5	1.3	1.3	1.3	1.3	1.3	1.3	1.3	0.2	-
Admin	MA$	0.9	-	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.1	0.0	-
TOTAL	**MA$**	**36.9**	**3.8**	**7.9**	**6.8**	**4.1**	**4.1**	**4.2**	**3.0**	**1.4**	**1.4**	**0.2**	**-**
Capital													
Mining	MA$	1.8	1.8										
Plant	MA$	6.0	2.0	2.0	2.0								
TOTAL	**MA$**	**7.8**	**3.8**	**2.0**	**2.0**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**
Re-habilitation													
Mining	MA$	0.8											0.8
Plant	MA$	(1.5)											(1.5)
TOTAL	**MA$**	**(0.8)**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**(0.8)**
Cashflow													
Undiscounted													
US$8,000/t Cu	MA$		(7.5)	(6.5)	1.9	7.1	9.2	8.6	8.2	9.4	9.4	1.4	0.8
US$6,700/t Cu	MA$		(7.5)	(7.1)	0.0	5.1	6.9	6.4	6.2	7.5	7.5	1.2	0.8
US$5,000/t Cu	MA$		(7.5)	(7.9)	(2.4)	2.6	3.9	3.4	3.7	5.1	5.1	0.8	0.8
US$2,000/t Cu	MA$		(7.5)	(9.3)	(6.7)	(1.9)	(1.5)	(1.7)	(0.9)	0.7	0.7	0.1	0.8
Discounted													
US$8,000/t Cu	MA$		(7.2)	(6.1)	1.7	6.1	7.6	6.8	6.2	6.9	6.7	1.0	0.5
US$6,700/t Cu	MA$		(7.2)	(6.6)	0.0	4.4	5.7	5.1	4.7	5.5	5.3	0.8	0.5
US$5,000/t Cu	MA$		(7.2)	(7.3)	(2.2)	2.2	3.2	2.7	2.8	3.7	3.6	0.5	0.5
US$2,000/t Cu	MA$		(7.2)	(8.6)	(6.0)	(1.7)	(1.3)	(1.4)	(0.7)	0.5	0.5	0.1	0.5
$NPV_{8\%}$													
US$8,000/t Cu	MA$		30.3										
US$6,700/t Cu	MA$		18.3										
US$5,000/t Cu	MA$		2.6										
US$2,000/t Cu	MA$		(25.1)										

The project produces a positive $NPV_{8\%}$ for copper prices greater than US$5,000 /t, but has a negative $NPV_{8\%}$ for the US$2,000 /t sensitivity case. The base case has an operating margin of around 50% of revenue. While this is a good margin the short project life, 3½ years of full plant production keeps the base case pre-tax $NPV_{8\%}$ low

at ~$18M. This does offer an opportunity should more mill feed be identified to extend the mine life and increase the $NPV_{8\%}$.

Figure 8.1 shows the cumulative undiscounted cashflow for each of the copper price scenarios.

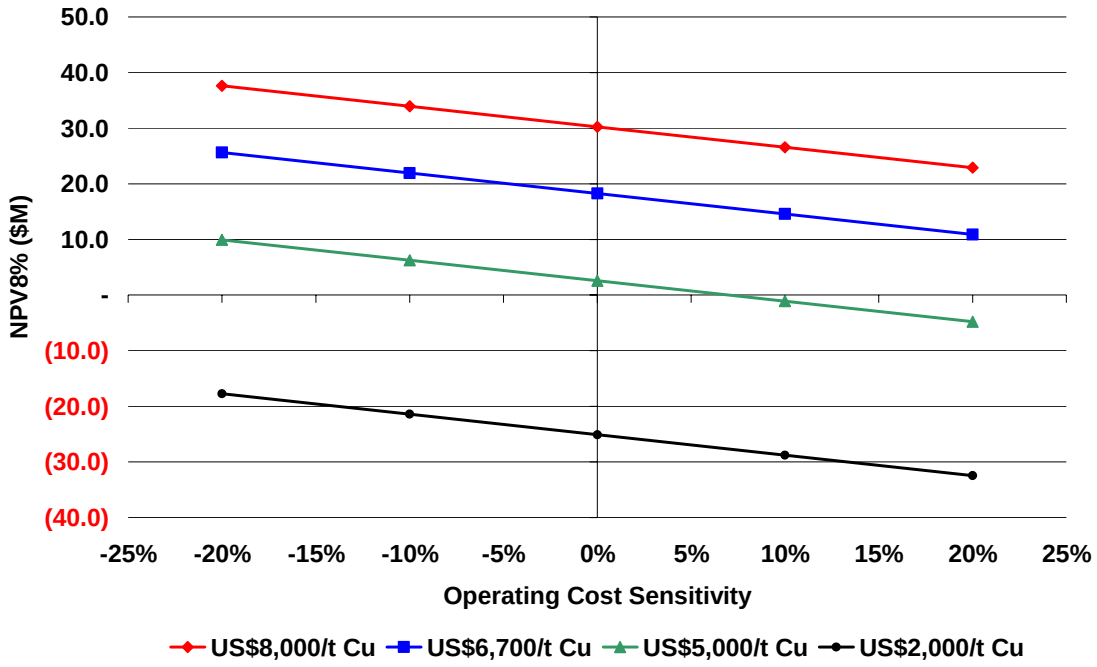
Figure 8.1 Cumulative Cashflow



For both the base case (US$6,500/t Cu) and the optimistic case (US$8,000 /t Cu) the project is cash positive by the beginning of Year three. The conservative case (US$5,000 /t Cu) becomes cash positive during Year three, while the very conservative case (US$3,500 /t Cu) struggles to pay back the capital.

The projects sensitivity to changes in cost was tested for a range from -20% to +20% for all costs including TC and RC charges. The results are shown in Figure 8.2.

Figure 8.2 Operating Cost Sensitivity



As would be expected the project is less sensitive to cost changes than revenue changes, with a 20% change in operating cost only giving 50% of the difference between the commodity price scenarios. This is due to the operation being relatively low cost, while the unit mining cost is relatively high due to the small size of the mining equipment; the strip ratio of 6:1 is not particularly high keeping the absolute mining cost relatively low. Also the simple gravity circuit proposed is a low cost treatment option.

Increasing the mining cost to A$5.00 /t for rock and the treatment cost to A$15.00 /t ore has a combined effect to reduce the base case pre-tax $NPV_{8\%}$ to A$9M. Increasing the recovery for 65% to 75% increases the base case pre-tax $NPV_{8\%}$ to A$27M.

9 PROJECT DEVELOPMENT SCHEDULE AND COST

Prior to making the decision to proceed with the project the following tasks will need to be completed:

- Diamond drilling programme – This will be required to gather data for geological analysis, metallurgical test work and geotechnical test work.
- Geological interpretation and modelling.
- Metallurgical and geotechnical test work.
- Pre-feasibility and feasibility studies for:
  - Mining.
  - Metallurgy and plant design.
  - Tails dam location and design.
  - Environmental and cultural impact.

AMC estimates the diamond drilling, geological interpretation and test work programs will take nine to twelve months to complete and cost approximately A$3M. The pre-feasibility and feasibility studies will require at least twelve months after completion of the drilling and test work programmes. The studies will cost around A$1M to A$1.5M.

Hence to take the project to the feasibility decision point is likely to take two years and cost approximately A$4M. This will need to be confirmed as the project progresses, particularly the cost of the feasibility study as this will be dependant on the outcome of the pre-feasibility study.

The first major milestone would be completion on the drilling and geological interpretation and modelling. The scoping study should be revisited at that stage to confirm there is value in moving to pre-feasibility.

10 RISK ANALYSIS

The major risks which will require to be addressed before a decision can be made to proceed with the project are:

- **Geological Model** – The geological model is based on poor quality data. Further drilling and analysis is required to bring it to a JORC Indicated or measured standard.
- **Iron Blow Water** – The water presents a danger of sudden in rush. A plan is required to remove the water and ensure it remains dry for the period of mining. This may require significant remedial action.
- **Historic Waste Dumps and Clay Slopes** – Both these present a significant danger of collapse. The extent of the risk needs to be further evaluated through geotechnical analysis, and design and operating protocols developed to minimise the danger.
- **Metallurgical** – There is presently limited metallurgical understanding of the mineralised zones. There is a danger of the material needing crushing to liberate the native copper or the copper being in a form which cannot be separated by gravity. A well structured metallurgical test programme will be required to address this issue.

There are a number of other waste dumping, environmental and heritage risks which need to be addressed prior to mining. While these are important they do not have the similar potential to lead to failure of the project as the issues listed above.

10.1 Geological Model

The geological model is based on sparse data, some of which is very difficult to verify and maybe of dubious quality. This is discussed in detail in Section 2.

Risk – The resource modelled is not present (tonnes or grade).

Likelihood – There is a very high likelihood of this occurring without significant drilling and resource estimation work.

Remedial Action – A structured drilling and resource interpretation process is essential before any decision on mining can be made.

One of the key issues with the model is density. The density used was 2.2t /bcm. As there is potential that this is too high, the financial sensitivity was tested by changing the density and hence tonnage of the feed. The effects on the $NPV_{8\%}$ were:

- 2.2t /bcm $NPV_{8\%}$ \$18M
- 2.0t /bcm $NPV_{8\%}$ \$16M
- 1.8t /bcm $NPV_{8\%}$ \$14M

Clearly confirming the density of the ore is a key risk which needs to be addressed.

10.2 Iron Blow Water

A significant quantity of water is contained in the Iron Blow open pit, the surface of which is between 50m and 100m above the base of the valley where operations will occur. An adit is known to run from below the water level out into the hill side likely to be mined for the copper clays project. It is not know if this has been deliberately plugged or if it has collapsed, the latter is more likely. The position of the blockage is not known. There is no ramp into the pit to allow water to be easily pumped and provide access to the adit for remedial work.

Risk – There is significant energy stored in the water and mining the slope below before remedial work has removed the water exposes CMT to a potentially catastrophic sudden inrush with a risk of major damage to equipment and injury or death to personnel.

Likelihood – There is a very high likelihood of this occurring if remedial work is not performed prior to mining.

Remedial Action – The water will need to be removed prior to mining commencing. The contained water will require treatment prior to release. Pumping will need to continue to keep the water low and/or the adit will need remedial work to allow drainage. Lack of access will make these major tasks.

Opportunities – The Iron Blow water also presents a risk to the underground mine. A vent shaft is close by (~100m). Should a connection occur this presents a risk of a sudden inrush into the underground mine. Hence the cost of the remedial work may not need to be born by the open pit project.

Threats – Environmental responsibility for Iron Blow passes to CMT once any work is done on the site. If draining the water or other remedial work triggers this liability the additional cost may need to be accounted for (fully or partially) by the project.

10.3 Historic Waste Dumps

Waste from the Iron Blow has been tipped directly onto the slope above the working area. The material underneath the waste dumps is clay. There is seemingly no engineering design that has ‘keyed’ the waste into the material below. As discussed earlier, AMC believe the volume is unlikely to be more than 100,000 m^3.

Risk – As the waste dumps have been tipped from the top they have formed at the angle of rill, this means there is no factor of safety and they will be close to collapse. Mining the slope below the waste dump presents a risk of a failure.

Likelihood – There is a high likelihood the dump will collapse if mining occurs below.

Remedial Action – The waste dump will require to be removed before work on the valley floor commences. Mining the dumps independently is not an option as equipment working on the top of the waste dump could trigger a failure. The waste dumps will require to be mined with the clay underneath them.

10.4 Clays in Slopes

The natural slopes above the valley floor are clays sat on more competent weathered rock. These are likely to be saturated due to high rainfall, and below the Iron Blow, due to the adit running from the pit.

Risk – Mining the toe of the slopes at angles steeper than the natural surface angle will present a risk of failure of the slope. This is exacerbated by the waste dumps sitting on the slopes. Mining from the top will expose the clay and the contact between the clay and harder material to rainfall. This will increase the likelihood of a failure.

Likelihood – There is a high likelihood of a failure.

Remedial Action – If the slope is to be mined the pit designs will need to take the walls back to competent material, and mining protocols will need to be developed which will allow the slopes to dry after significant rainfall prior to mining recommencing. This may prevent mining from occurring during the winter months. Where a slope has had waste dumped on it protocols will be required which limit the time and the manner in which equipment works on the dumped material. This may involve using a dozer push down the dump to a lower level before it is excavated.

10.5 Metallurgical

It is believed the mineralised material is predominantly soft clay with liberated native copper which can be extracted by feeding directly to a gravity circuit. Some of the associated material may be hard and require comminution if any contained copper is to be extracted. The copper clays mineralogy is not presently well defined to assess this risk. Although it has been suggested that native copper represents the dominant copper mineral, other copper minerals such as cuprite, chalcocite, covellite, bornite and chalcopyrite have also been reported, with the sulphides likely becoming more abundant at depth. These copper minerals could be potentially recovered by gravity processing if they were present as liberated components in low SG clays, but more complex treatment options would be required if a significant proportion of the copper mineralisation was contained within the hard rock segment of the deposit.

Risk – Copper which is not in the form of native copper in soft clay will have a very poor recovery. It is unlikely that the amounts of hard rock material will be sufficient to justify a comminution circuit. The grade of any gravity circuit tail generated is likely to be lower than the underground ore head grade so that feeding a low grade gravity circuit tail into the already overloaded comminution circuit in the existing mill will displace better grade material. Also, feeding clay ore directly into the plant in conjunction with underground sulphide ore will have a deleterious effect on the overall process response and hence reduce copper recovery. Thus, combining either the clay ore or a gravity circuit tail with the main plant feed is not acceptable.

Likelihood – Hard zones, ~5% of the drilling, and areas of chalcopyrite have been noted. The extent of these is not fully understood, so the risk of a significant drop in recovery is moderate to high.

Remedial Action – A well structure metallurgical test programme is required estimate the true quantities of material types.

10.6 Waste Dumps

The areas presently identified as potential waste dump sites have potential heritage and resource sterilisation issues. There are limited other options for waste disposal. Prior to commencement further investigation and analysis will be required to estimate the potential of resources which could be sterilised to assess the trade off between other not identified options and loosing access to the resources in the dump area.

10.7 Environmental and Heritage

Additional Environmental and heritage risks include:

- **Water Discharge** – Control of pit and site stormwater, and process discharge water quality, particularly turbidity. Also the impact of discharge on Linda Creek and Lake Burbury, this will require sediment retention / water treatment system.
- **Tailings management** – This presents a risk due to high clay and water content leading to poor settling. A suitable tailings dam site not yet identified.
- **Acid Mine Drainage** – Generation of acid drainage from waste rock and open cut during and post operation. Waste will need to be assessed for acid generation capacity (NAG etc.).
- **Local community** – Noise, vibration and dust from operation has potential to disturb the local community.
- **Disturbance of cultural heritage sites** – There are a few sites with cultural significance identified in the Godden Mackay assessment of 1994 upslope of the resource. The Iron Blow open cut is also an identified site that may require appropriate approvals if it is to be used for tailings / waste disposal.
- **Disturbance of geo heritage features** – Part of the area includes the listed geo-heritage feature of the 'Linda Moraine'. This is a regional feature and exposure could be useful for further academic study.
- **Disturbance to flora** – Potential for threatened species will require a flora and fauna habitat survey prior to disturbance.
- **Hydrocarbon management** – A protocol for spill / contamination prevention and incident response will be required.
- **Weed Management** – Control of weed and plant disease (phytophthora) will require equipment wash down and inspection.

11 CONCLUSIONS AND RECOMMENDATIONS

While the project still has a very high level of uncertainty AMC believes there is a justification to proceed with the work required to take the project to the next level of evaluation, which would be a Pre-feasibility study.

The Copper Clays project has the potential provide significant value to CMT, the analysis suggests the NPV at an 8% discount rate ($NPV_{8\%}$) around A$10M to A$20M for the base case copper price of US$6,700 /t, should prices remain at the present level of US$8,000 /t the project will generate a pre-tax NPV of approximately A$25M. This is based on a pit containing approximately 900,000t of ore at a grade of 1.9% Cu and a mass based strip ratio of 6:1.

Presently there is a very high level of uncertainty as there is limited geological, geotechnical and metallurgical understanding of the deposit. This would need to be addressed before the project could commence. As the geographical extent of project is limited, the total cost of addressing these issues should not be high.

The Pre-feasibility study should address the various mining and processing options, selecting the best for final analysis and costing in a Feasibility Study that would then form a basis for the investment decision. The cost of the final Feasibility Study would depend on the results from the Pre-feasibility study and the level of comfort required by CMT prior to proceeding with the project.

As part of the Feasibility Study plans and protocols would need to be developed to manage the risk presented by the water in the Iron Blow open pit.

APPENDIX A

AMC PRE-SCOPING REPORT

AMC Consultants Pty Ltd

ABN 58 008 129 164

10 December 2007

Scot Clyde
General Manager
Copper Mines of Tasmania

Dear Scot

DRAFT Copper Clays Pre-scoping Review – AMC 107122

AMC has performed an initial review of the Copper Clays project and recommends that the scoping study should proceed, but the analysis in this report should be reviewed at the end of or shortly after the initial site visit to confirm it is worth progressing with the full proposed scope of the scoping study. This review will include more expert opinion on the geotechnical and metallurgical aspects of the study, more detailed analysis of the mining issues, and use the geological model which AMC understands CMT are presently completing.

The analysis suggests that the project is marginal applying conservative but realistic parameters, but if slightly more optimistic parameters could be achieved the project has potential to provide a significant operating surplus, possibly in excess of A$40M. This is undiscounted and includes no capital or depreciation.

Yours sincerely,

James Stoddart
Principal Mining Engineer

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Project Manager

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Peer Reviewer

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1 Introduction

AMC was requested to perform a quick, three day, review of the Copper Clays project prior to commencement of a scoping study. The purpose was to identify if there was any possibility of a project being developed on the potential resources and hence any value in proceeding with the proposed scoping study.

2 Analysis

CMT provided wireframes of the topography and the mineralized zone. To date no resource model has been provided. The wireframes were filled with blocks to produce a rock model with the mineralised zone flagged. The Whittle Programming 4X software was used to generate two suites of shells based on this model; one with 35° wall angles and one with 45° wall angles. These shells provided ore and waste volumes which could then be analysed for a range of physical and financial parameters.

The base case set of parameters used was:

- Head grade to mill 1.5% Cu
- Dry bulk density 2.2 t/bcm
- Mill recovery 65%
- Mining cost A$7.50/t rock
- Treatment cost A$10.00/t ore
- Realisation cost US$700/t Cu
- Copper price US$5,000/t
- Exchange rate US$0.85 : A$1.00

The realisation cost is based on:

- Treatment cost US$50/t concentrate
- Refining cost US$0.05/lb (US$110/t Cu)
- Transport US$100/t concentrate
- Concentrate grade 45% Cu
- Payable metal 95%

These parameters are considered as conservative but realistic. To test the potential upside a set of optimistic but realistic parameters were tested. The changes were:

- Head grade to mill 1.8% Cu
- Dry bulk density 2.4 t/bcm
- Mill recovery 70%

The head grade chosen is marginally higher than that used in the 1997 analysis (1.4% Cu) but lower than the 1997 estimate for the western region of 2% Cu. As the LG shells focus on the better grade material near the Batchelor shaft the base case and upside cases seem appropriate.

The density is higher than AMC would have anticipated; our experience is that the bulk densities of clays can be 1.6 or even lower. The 1997 analysis used ~2.5. 2.2 was selected in discussion with the site geologists.

Mill recovery and costs are based on experience of similar gravity plants.

Mining cost is high, but the conditions will require either significant sheeting or use of six wheel drive trucks. Additionally, the mining rate will be low and the potential for wet weather conditions will reduce the utilisation of the mining fleet. The cost is based on recent experience with 40t six wheel drive trucks mining clay in tropical conditions.

The ore contents, strip ratios and un-discounted operating base case and upside case surpluses for the 35° and 45° suites of shells are shown in Figures 1 and 2. The operating surpluses have no capital or depreciation included and are only indicative of potential results.

Figure 1 35° Shell Pit Contents, Strip Ratio and Base Case and Upside Case Operating Surpluses

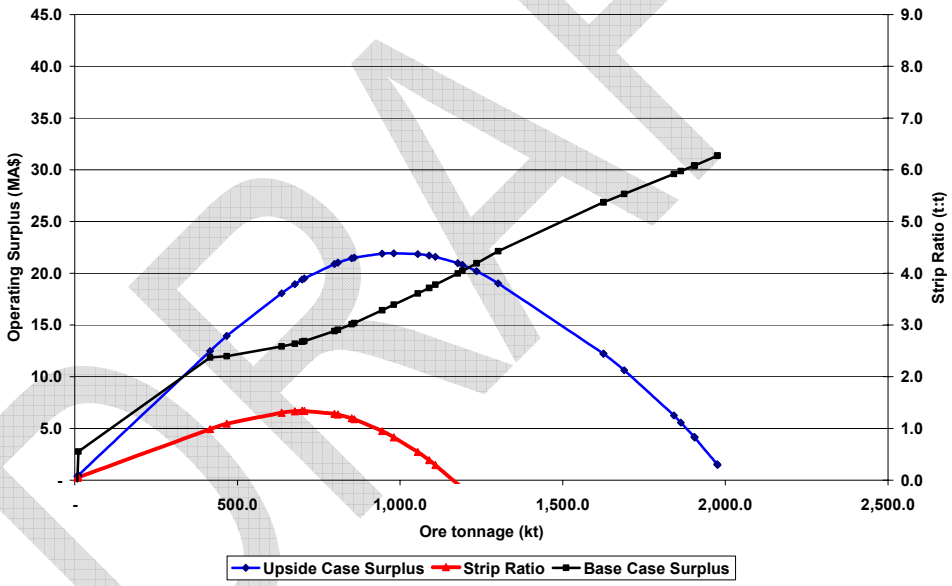
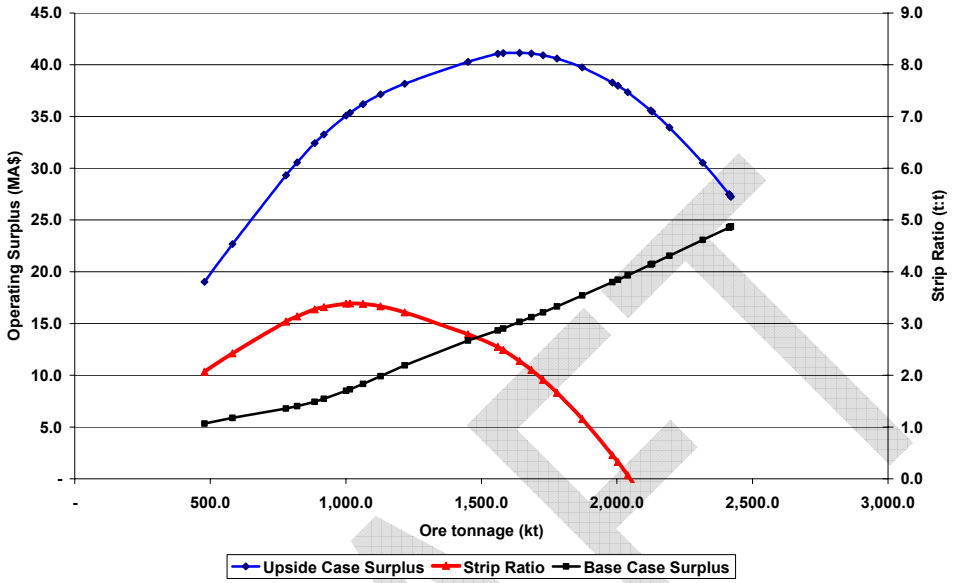


Figure 2 45° Shell Pit Contents, Strip Ratio and Base Case and Upside Case Operating Surpluses



The analysis suggests that the base case using 35° slopes generates little or no surplus, but the upside 35° case generates around A$20M surplus from a pit containing approximately 1Mt of ore.

The 45° base case highlights that the result is sensitive to slope angle generating an additional A$15M compared to the 35° base case. When the upside parameters are applied the contained ore increases to over 1.5Mt and the surplus increases to over A$40M.

3 Risks

The major risks for the project are:

- **Resource** – The drilling is limited, some dating back to 1902. This represents a major risk for grade, extent and density. Clearly this is a potential opportunity with improved grade or quantities.
- **Mining** – The material is predominantly soft clay which will be difficult to traffic and may require specialist mining equipment or transporting significant amounts of material into the pit.
- **Weather/Water** – The pit will go below the valley floor level at the head of a drainage. Coupled with the high rainfall this presents a risk of flooding and will most probably limit when the pit can be mined. Additionally, the Iron Blow pit is flooded and sits above the likely pit location; this may present a significant risk of water in-rush. It will also lead to high levels of moisture in the wall rocks.

4 Conclusions

The analysis suggests that while the project is marginal applying conservative but realistic parameters, if slightly more optimistic parameters could be achieved the project has potential to provide a significant operating surplus.

AMC recommends that the scoping study should proceed, but this analysis should be reviewed at the end of or shortly after the initial site visit to confirm it worth progressing with the full proposed scope of the scoping study.

APPENDIX B

PREVIOUS STUDIES

COPPER MINES OF TASMANIA PTY LTD
ACN 065 339 835

OPEN CUT POTENTIAL OF THE COPPER CLAYS AREA MOUNT LYELL, TASMANIA

REPORT NO: 1995-50

Author : K J A Wills

Report No. : 1995-50
Date : June 1995
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: K J A Wills
Ref: Reps: 1995-50

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Tony Weston
& NDEC Chm

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1. SUMMARY

This report synthesises 112 years of mining and exploration history at the Mt Lyell Copper Clays. These are a series of native copper in clay deposits located at the head of the Linda Valley, just below the Mt Lyell Saddle. The main deposits are Lyell Blocks, Lyell Consols and King Lyell. They occur in valleys and are generally covered by thin fluvioglacial sediments. The data on which this report is based have been available for the last 25 years but have not been previously reported or interpreted.

During the 1886 Linda gold rush, in addition to gold, up to 1 kilogram per dish of native copper could be panned from Cooney's Creek (King Lyell) and on Jimmy Watson's Claims (Lyell Blocks). Early copper production commenced with sluicing operations and in 1902 the Blocks commenced gravity treatment of underground ore. The Blocks mine had a large and rambling gravity plant that was visited by metallurgists from all over the Commonwealth, and pronounced first class.

Total production between 1892 and 1910 was 243,000 tonnes of 1.6% ore for 2,750 tonnes of copper. The Blocks and Consols underground mines had severe problems with ground stability and both were closed following collapses and flooding. During its brief life, the Blocks produced a concentrate averaging 69% copper with a recovery of 72%. The Mount Lyell Blocks Mining Company NL was one of the few companies on the field apart from the MLMRC to declare dividends. The mine was eventually sold to the MLMRC for £5,000 in 1919. Blainey summed up its history succinctly as "a courageous and enterprising company; it deserved a better fate."

Several early factual geological descriptions of the deposits have been very useful. Native copper was found outcropping in 'pug' in the form of: sheets in joints; dendritic masses coated in chalcocite; as pebbles, nuggets, shots and spangles; as masses up to 75 kgm and as limonite geode fillings of interlayered cuprite and native copper. Batchelor's original 1902 diamond drill logs provide neat and factual descriptions which showed that native copper was the main ore mineral throughout the clay hosted deposits and into the underlying unconsolidated Pioneer Sandstones.

Previous geological descriptions and genetic ideas are reviewed and it is shown that recent workers have suffered from a lack of access to good factual data. This led to unjustified theories which were tested by expensive and unsuccessful exploration programmes. The worst example was the drilling of two long diamond holes in 1984 which showed there was no ore where the Block's main shaft had previously reached the same conclusion in 1915.

Elements of previous genetic ideas have been combined to produce a new model which is factually well supported. It involves the dissolution of copper during weathering into acid oxidising drainage and the neutralisation of solutions during mass formational dissolution of Gordon Limestone. Copper is precipitated by the reducing effects of clay, carbon or humic acid in the limestone residue or pug. The process has been termed natural Cainozoic hydrometallurgy and has a predictive capacity which will hopefully aid future exploration.

A total of 46 diamond drillholes for about 4,000 metres has been drilled during exploration of the Copper Clays between 1902 and 1970. New summary logs have been produced, plotted and interpreted. A sectional resource estimate using a cut off

grade of 0.1% Cu gave a global inferred resource of 8Mt at 0.6% copper. Core recovery averaged between 56 and 70% which results in a higher than normal uncertainty on the grade of the resource. Simple pit designs lead to an estimated recoverable resource at a waste to ore ratio of 2.3 to 1 of 6.9Mt at 0.6% copper. Estimates of total resource potential including inferred resources have also been made. They suggest that 19Mt at 0.8% Cu may be present with a target of potentially recoverable copper in the order of 90,000 tonnes.

Preliminary economics are examined. For contract mining and a gravity copper separation plant, operating costs of $11.28 per tonne are estimated. This equates to a break-even head grade of 0.45% copper. To produce copper with operating costs of 50% of revenue, a head grade of 0.9% is needed, and believed achievable. This suggests that mining of the Copper Clays could be highly profitable. It also justifies an exploration programme to upgrade the status of the resource and provide information for a pre-feasibility study.

A 64-hole, 5,590m programme of vertical air core and face sampling hammer drilling is proposed. This will lead to a better understanding of the deposits and possible estimation of an indicated resource. By weighing drill samples, more reliable estimates of recoveries and tonnage factors will be obtained. It is also proposed to commence collecting metallurgical and mining information.

Finally, the long term exploration potential for the entire Copper Clays area is assessed. Four main targets are considered worthy of exploration attention. Firstly, the Copper Clay deposits as described above. Secondly, there are four locations of limited Tabberabberan quartz vein gold known. These suggest leakage from potential Henty-style gold orebodies in structures at depth. Thirdly, carbonate hosted lead-zinc mineralisation is suggested by the presence of fine-grained galena and honey-yellow sphalerite in concentrates from non cupriferous clays. Fourthly, there is the much greater prize of a continuation of the Mt Lyell Copper-Gold-Silver Field at a depth of about 1,500 metres under the Copper Clay deposits and Owen Conglomerate on the eastern side of the Great Lyell Fault.

The Copper Clays are a rare to unique type of ore deposit and their successful evaluation is a challenging task. However, potential rewards are high and it is believed that, like the impact of the 1893 Mt Lyell silver bonanza on the MLMRC, the Copper Clays have the potential to produce a significant cash injection during Copper Mines of Tasmania's ("CMT") early years at Mt Lyell.

2. INTRODUCTION

The Copper Clays at Mt Lyell are an unusual group of native copper in clay deposits at the head of the Linda Valley which have been promoted by many but with very little long term effect. The deposits are all located under modern water courses at altitudes of between 300 and 600m above sea level as shown on Figure 1. There are three previous mines and a large area of prospective ground (shown on Figure 1 as Mpc) which may be mineralised.

As part of a brief to examine the future open cut potential of the Mt Lyell area, it became clear in the initial assessment (Wills, 1995a), that excellent potential existed in the Copper Clays area. This was not difficult to conclude as estimates of 'indicated' ore of 1.99 Mt at 1.92% Cu, with an additional 6 million tonnes possible, were made by Wade (1957b). Subsequent workers appear not to have been aware of Wade's conclusions as they have not been mentioned again until Flitcroft and McKeown's (1992) report and subsequently by Newnham (1993), Snowden (1994) and Ellis (1994).

After producing a first-pass assessment of the West Lyell resource, (Wills, 1995b), and while work on an upgrade of the West Lyell resource estimation was under way, an early assessment of the Copper Clays area was requested by Gold Mines of Australia Limited's ("GMA") management so that decisions on future metallurgical and exploration work could be made.

The work for this report has taken about 5 weeks, mostly compiling data, but results have exceeded expectations. It is thought that this has resulted from a situation, particularly during the 1980's, where 'theory' based exploration had gained the upper hand and participants were either unable or unwilling to consider, or unaware of the existence of, the large amount of factual descriptive information which conflicted with their concepts.

This report has been designed to provide a comprehensive overview of the history and economic geology of the Copper Clays deposits. Very little new work other than drawing up and interpreting drill sections has been necessary.

Historical data from the 36 years of operations between 1883 and 1919 is a vital part of the story. Three operations demonstrated that production of high grade copper concentrates by gravity methods was possible. However, underground mining of unconsolidated materials has obvious dangers and after collapses and flooding, the early mines were forced to close. If the deposits are amenable to cheap open cut mining, as well as to gravity concentration, then hurdles to their larger scale development can now be successfully overcome.

3. CONCLUSIONS

3.1 Copper Clays Neglected

Information used in this report has lain dormant at Mt Lyell for 25 years. Between 1902 and 1970, 46 diamond drillholes were completed and neither the results nor their implications have been properly assessed.

3.2 History

The Copper Clays were the first producers of copper metal at Mt Lyell with a total of about 2750 tonnes of copper from 243,000 tonnes of ore averaging 1.6% between 1892 and 1910. The largest mine at Lyell Blocks was in production between 1902 and 1907

with a peak labour force of 282 men. They produced high grade native copper rich concentrate averaging 69% with a recovery of 72% by gravity separation.

3.3 Geology

The Copper Clays are a rare to unique group of mineral deposits hosted in clays formed as a residue after the mass formational dissolution of Gordon Limestone in post landscape-formation times. A natural Cainozoic hydrometallurgical model is proposed for their genesis. In this model, acid oxidised drainage is neutralised by reaction with limestone and native copper and cuprite are precipitated by reduction within the clays.

3.4 Resource

Resources for the three main deposits total 8.0 mt at a grade of 0.6% copper. Due to poor core recovery from unconsolidated sediments, this resource can only be classified as an inferred resource. Simple open cuts have been designed and an overall waste to ore ratio of 2.3 to 1 with a recoverable resource of 6.9 mt at 0.6% Cu has been derived. Due to the lack of drillholes in the central higher-grade cores of the deposits, the grade is lower than that expected after further systematic drilling.

A number of extensions are likely with a target resource potential of 19 mt at 0.8% Cu containing approximately 90,000 tonnes of recoverable copper.

3.5 Economics

Using preliminary costs for a contract mining and gravity separation type of operation, with waste to ore of 2.3:1, operating costs total $11.28 per tonne. At a copper price of $35 per % Cu this is equivalent to a break even recovered grade of 0.32% Cu and head grade of 0.45% Cu. To operate with costs at 50% of revenue a head grade of about 0.9% Cu is necessary. The clays have the potential to be a highly profitable operation and an exploration programme to upgrade the status of the resource is justified.

4. RECOMMENDATIONS

4.1 Proposed Exploration Programme

A 64-hole, 5,590m programme of mixed vertical air core and face sampling hammer drillholes is proposed. This will lead to a better understanding of the deposit and possibly enable classification as an indicated resource so that a pre-feasibility study can be undertaken.

4.2 Treatment of Drill Samples

During the programme, careful work collecting, weighing and describing drill samples should lead to more reliable estimates of recoveries, tonnage factors and mineral distribution. Several types of copper analysis such as strong acid, weak acid and gravity recoverable copper will also help in later assessments.

4.3 Metallurgical and Mining Investigations

Representative metallurgical samples can be obtained during the drilling programme. Composite samples should be collected and size analysis and gravity recoverable copper determined. Drilling techniques need to be recorded in detail for each hole to build up a model for the proportion of each orebody which is amenable to free digging, and that requiring drill and blast.

5. HISTORY

The original Mt Lyell discovery in 1883 was based on following up alluvial and eluvial gold shed from the Iron Blow. When operating capital became scarce in 1886, the Mt Lyell Prospectors Association received a boost from a very bullish report by the Government Geologist, Gustav Thureau. He claimed the deposit was phenomenally rich in gold and so large that the deposit would be practically inexhaustible (Thureau, 1886, p. 7). In 1887, the Prospectors Association engaged Sydney geologist, J R Robertson, to assess Mt Lyell. He contradicted Thureau by saying that the average ore was poor in gold and silver and rich in copper, and was similar to the Tharsis Mine in Spain (Blainey, 1993, p. 43).

Despite Robertson's assessment, the Prospectors Association was dissolved and the Mt Lyell Gold Mining Company formed in 1888. Capital was raised and an eight-head stamp battery set up to recover the gold bonanza. Both Robertson and Thureau protested that abundant barite in the ore would severely hinder gold recovery. The battery operation was not viable and in 1891 the company was put up for sale. It was purchased by Bowes Kelly who then floated the Mt Lyell Mining Company in January 1892.

After another bullish, though this time reasonably accurate report by Peters (1893), the Mt Lyell Mining and Railway Company was formed in March 1893. The company did flourish in its early years from a bonanza; but from silver rather than gold or copper. A rich vein of stromeyerite [(Cu, Ag)S] containing 862 tonnes at 3.1% produced 24.4 tonnes of silver and provided the young company with a windfall gain of some £106,000, and some very credible publicity.

From reports by Peters (1893) and Thureau (1886), early gold production of about 4000 ounces can be estimated in the 1880's. This included some 2,200 ounces from the Blow and 1,800 ounces from the Upper Linda Valley alluvials. Thureau (1886, p.8) also noted that native copper was abundant in creeks about 200m east of the Blow (King Lyell) and from Watson's Claim (Lyell Blocks). He noted that: "besides gold, copper is found in its pure malleable state embedded in a kind of hard brown clay. Lumps of pure copper weighing between 1 and 3 kg could be found and up to 1 kg of native copper could be washed in a dish." No records of copper production for the 1883-1897 period have yet been located, but it seems clear that the first significant copper production from the Mt Lyell Field was not from sulphide deposits, but actually from the Copper Clays.

Power (1891, p. 32) also examined the alluvial deposits and provided a useful early description of them: "On tracing the alluvial copper to its source we are led to a zone of

decomposed schists forming a pug; grey, yellow and red, some hundred feet broad ..... the copper is found in sheets as if occupying the joints and cleavage planes of the original rock. These sheets of metal which at times are fairly thick, get broken up into small nuggets, shots and spangles which are more or less coated with the black oxide of copper (chalcocite). Below the native copper we come across cuprite which occurs in beautiful crystals, mostly octahedrons. No doubt the native copper has been reduced from this by the agency of the peaty waters. Still deeper than the cuprite, we come across copper pyrites and/or iron pyrites."

Another useful early description from underground observations at Lyell Blocks is by Gregory (1905, p. 140). "From observation it would appear as if the deposition of the ores had taken place subsequent to some of the transverse faulting as is evidenced by the fact that these smaller dislocations intersect the impregnated areas without any apparent alteration in the peripheral walls (such as they are) of the ore occurrences .....(Later, p. 142) the 'eastern orebody' within the North Mt Lyell property ..... is quite different from any other in the mine, and is made up of quartzitic boulders and decomposed matter carrying chalcopyrite and bornite. The soft dark decomposed matter is by far the richer in metallic contents, probably being enriched through decomposition of the original ore."

There are also some useful descriptions in W T Batchelor's 1902 drill logs of holes ML8-12 at the King Lyell deposit. Holes 9 and 10 passed through native copper bearing clay over their entire lengths and bottomed in native copper bearing conglomerates. 'Pebbles' of native copper as well as associated 'fossil wood' were also described.

It is clear that the early geologists, some of whose valuable observations are cited, had recognised many features of the Copper Clay deposits which appear to have been overlooked in more recent times.

The early history of the Mt Lyell Field was fiercely competitive and, after discovery of rich ironstone-hosted gold at the Blow in June 1886, news spread fast and sparked a gold rush. By September, over 200 diggers were working in the upper Linda Valley and any remaining ground was blanket pegged. In the years to about 1910, the cumulative effect of hundreds of miners and prospectors working for 40 different companies putting in exploration adits and shafts is thought to have been far more effective than the modern era exploration.

Only four companies have ever paid dividends from activities on the Mt Lyell Field. Of these, MLMRC, Lyell Tharsis and South Tharsis were all dependent on the financial success of the MLMRC for their production. The only other company to pay a dividend from its own efforts was the Lyell Blocks Company in 1906. The combination of the first copper production and an ability to generate dividends are the two most notable features of the early history of the Copper Clay deposits.

More detailed descriptions of the mining and production history of the main deposits are given below.

5.1 Lyell Blocks

Native copper mineralisation at the Blocks is mentioned in the first known geological report on the Mt Lyell Field by Thureau (1886). At this time, title was held by Jimmy Watson whose partners had located alluvial copper and gold. This included the largest gold nugget from the Goldfield which was located near the topographically highest part of the diggings. This 6.5 ounce nugget contained 4.5 ounces of gold associated with well-formed quartz crystals. Two creeks draining the southern slopes of Mt Lyell were named Gold and Copper Creeks after their alluvial contents (Figures 1 and 26). To the writer's knowledge, a primary source for the gold in Gold Creek has never been identified and could lie under scree cover along the North Lyell Fault near grid position 6100N-6350E.

Some early workers, for example Power (1891, p. 30) thought that the source of the gold was the conglomerates surrounding the upper Linda Valley. Power noted that where gold still clings to its matrix, it is sometimes in contact with hematite and sometimes with quartz. He also mentioned that he and local miners Jack Fehey and Steve Karlson had obtained gold from the products of decomposing conglomerate boulders.

After the 1886 rush, Watson's six claims were floated into the North Mt Lyell Prospecting Association who unfortunately found nothing to develop (Blainey, 1993, p. 102). In 1889, the leases came into the possession of a Sydney syndicate known as the Stanley Company (Mining Standard, 1898). They spent £3,000 but were unable to make the property pay. However, they did prove the existence of a large lode of auriferous copper clay. They also held the ground without working it for a long time. Some time before 1892, the leases became owned by the Idaho Copper Mining Company. Two reports by Thureau (1892; 1893) give information on this period. Twenty tonnes of concentrate were sent to Newcastle (NSW) where they yielded 90% copper.

Some time before 1897 (Russell, 1898), the ground was renamed the Blocks and ownership passed to the Melbourne-based Mt Lyell Blocks Copper Corporation. The new manager was Robert Schloesser who placed an emphasis on underground exploration for which he was congratulated by the Press (Mining Standard, 1898). During the period 1897-99 significant sulphide and native Cu ore was discovered and production was said to be imminent (PMIT).

Production did commence during the September 1900 quarter when 3,583 tonnes of bornite ore, said to average 10% Cu, were produced. About the first 10,000 tonnes of bornite ore produced came from the North Lyell orebody on the Blocks leases. However, about the last 8,000 tonnes were stolen from the North Mt Lyell company's ground. The MLMRC were the main benefactors (Blainey, 1993, p. 143). A total of 18,568 tonnes averaging 7.6% copper with similar gold and silver contents to the North Lyell orebody grades were sold to MLMRC between the September 1900 and December 1901 quarters. This production has been included with statistics on the North Lyell Orebody and is not listed in Table 1. Before the end of 1901, probably to avoid legal prosecution, the Mt Lyell Blocks Copper Corporation was dissolved and reformed as the Mt Lyell Blocks Mining Company NL.

Table 1

Copper Clays, Historic Production Details

Mine	Period	Data Source	Tonnes Treated	*Head Grade % Cu	Recovered Grade % Cu	Tonnes Produced Cu
Lyell Blocks	pre-1892	T	20.3	Cons	90.0	18.3
	1902	P	7,500	2.81	2.02	151.7
	1903	P	No production			
	1904	P	17,429	1.72	1.24	215.4
	1905	P	64,410	1.80	1.28	823.2
	1906	P	71,926	1.51	1.09	781.6
	1907	P	45,089	1.14	0.82	368.3
Total Lyell Blocks			206,354	1.58	1.13	2,385.5
Lyell Consols	1909	P	8,700	1.11	0.80	69.4
	1910	P	14,906	1.07	0.77	115.3
Total Lyell Consols			23,606	1.09	0.78	184.7
King Lyell	pre 1897	R	10,000	2.08	1.5	150
	1898	P	2,760	2.08	1.5	41.4
	1899	P	273	2.08	1.5	4.1
	pre 1903	B1	21	56.1	40.4	8.5
	1904	B2	2	1.53	1.1	0.02
	1906	B2	4	2.36	1.7	0.07
Total King Lyell			13,060	2.17	1.56	204.1
Total Copper Clays 1883-1910 (28 years)			243,020	1.56	1.12	2.747.3

* Head grades estimated assuming 72% Recovery from Table 2.

Data Source Abbreviations

T = Thureau (1892)

P = Progress of the mineral industry of Tasmania quarterly reports 1897-1913.

B = Record of Mining and Treatment operations. "The Black Book" CMT Ref T1991-008. B1 purchased prior to Mt Lyell - North Lyell amalgamation in 1903. B2 during MLMRC ownership.

R = Reid (1970) from G W Blainey's personal records.

During the period of bornite mining, a significant amount of development work had been carried out (Figure 27). Therefore, as early as the March quarter of 1902, the new company was able to commence underground production for the first time on copper-clay ore. In this quarter, 592 tonnes of ore was treated by two puddling machines and blanket tables for 34.4 tonnes of concentrates grading 49% copper (Table 2). Over the next two quarters, concentrate grade increased initially to 58.1 and then to 64.3% copper. The entire 1902 production was 7,500 tonnes of ore for 267.3 tonnes of concentrates averaging 57% copper (Tables 1 and 2).

2.8% Cu rec'd.

Production was halted in late 1902 by a strike over wages. Eventually, work was resumed with no changes to pay, but with falling copper prices the result was a loss making situation, so the mine was closed and most workers dismissed (Blainey, 1993, p. 201). The copper price from April to December 1901 averaged £71 per tonne and from January to September 1902, it averaged £54 per tonne.

The period to August 1904 was spent reconstructing the company, carrying out underground exploration and development and in constructing a new mill. The mill construction is described in detail by Grayson (1903). The clay was initially desegregated in puddlers and oversize material separated. This was hand sorted and any copper-bearing materials put through a Chillian mill. The fines and crushed material were then passed over a hydraulic classifier and treated on Wilfley and Ferraris tables. Grayson predicted an 80% recovery and concentrates grading 75% Cu. A second product, largely composed of ironstone, assaying 20% Cu was also produced. The plant was planned to treat 1,500 tonnes per week and for three years from August 1909 to July 1907, it averaged 1,305 tonnes per 50-week year.

The mill was described by Blainey (1993, p. 187) as 'large and rambling' and was visited by metallurgists from all over the Commonwealth and pronounced 'first class'.

During peak production, a maximum of 284 men and an average of 242 men per quarter were employed at the Lyell Blocks Mine. The company recovered at least 70% of the copper and successfully treated about 200,000 tonnes of ore producing concentrates averaging 69% Cu for about 2,200 tonnes of copper (PMIT statistics, listed in Tables 1 and 2). Concentrates were railed to the Kelly Basin and shipped to the Wallaroo smelters. The stock was popular and in 1906 the Blocks paid £15,000 in dividends as well as spending additional funds on exploration properties at Mt Darwin and Zeehan (Blainey, op. cit.).

Unfortunately, the clay in the mine tended to swell and smash the heavy timber supports. In July 1907 the timber, clay and slurry caved in over a wide area and buried two men; only one survived. Officially, the mine was closed due to the low copper price and the low ore grade. The recovered grade had gradually dropped from 3.09 to 0.87% Cu between 1902 and 1907. By December 1907, the mine was idle (PMIT, 1907).

The mine remained closed for some time and there was consideration of a merger with the adjacent Consols mine which was preparing for production in 1908. The Blocks was re-opened in December 1909, but copper clay production did not recommence.

Table 2

Available Details of Gravity Concentrates

2.1 Lyell Blocks

Note:
Plain numbers are as reported or calculated from reported numbers.
Numbers with superscript [e] are estimates based on averages of other reported numbers.

Quarter Ending	Ore Tonnes T	Head Grade % Cu	Recovered Grade % Cu	Conct. Produced T	Conc. Ratio	Con Grade % Cu	Cu Metal Produced	Recovery %
31.03.02	592	4.29[e]	3.09	37.4	15.8	48.9	18.3	72[e]
30.06.02	3,587	2.90[e]	2.09	129.0	27.8	58.1	75.0	72[e]
30.09.02	2,399	3.19[e]	2.30	85.7[e]	28.0[e]	64.3[e]	55.1	72[e]
31.12.02	908	0.51[e]	0.37	15.2	59.7	22.4	3.4	72[e]
30.09.05	14,242	1.90	1.37[e]	267.0	53.3	73.0[e]	195.1[e]	72[e]
31.12.05	19,203	1.90	1.39	335.3	57.3	79.4	266.2	73
31.03.06	12,371	1.58[e]	1.13	196.1	63.1	71.5	140.2	72[e]
30.09.06	17,927	1.50	1.05	301.2[e]	59.5[e]	62.0[e]	188.2	70
Totals	**71,229**	**1.84**	**1.32**	**1,366.9**	**52.1**	**68.9**	**941.5**	**72**

2.2 Lyell Consols

30.09.09	4,267	1.02[e]	0.73	46.7	91.3	67.0	31.3	72[e]
31.12.09	4,433	1.19[e]	0.86	66.7	66.5	57.2	38.1	72[e]
31.03.10	3,338	1.33[e]	0.96	49.7	67.2	64.4	32.0	72[e]
30.06.10	7,302	1.74[e]	1.25	78.7	92.7	61.8	48.7	72[e]
30.09.10	4,267	1.13[e]	0.81	58.8	72.5	58.8	34.7	72[e]
Totals	**23,607**	**1.09**	**0.78**	**300.6**	**78.5**	**61.5**	**184.8**	**72**

Data from Progress of the Mineral Industry of Tasmania (PMIT).

Between late 1909 and 1915, the Blocks carried out development work in the search for more bornite ore. This was largely by sinking of the 414m deep Blocks shaft. This was located only 10m from the North Mount Lyell Company's lease boundary. The search for bornite ore was unsuccessful, as were the two diamond drillholes NL1101 and 1102, which tested the same target area in 1984 (Bird, 1984; 1985). The Blocks Mine was inactive until 1919 when it was sold to the MLMRC for £5,000. Blainey (1993, p. 193) summed up the Blocks Mining Company's history succinctly: "A courageous and enterprising company, it deserved a better fate."

The MLMRC have done little with the ground except for the Corridor mining activities which impinge on the Blocks western boundary and the drilling of 16 diamond drillholes between 1963 and 1984, most of which were neither plotted up nor reported on. In contrast, a very useful report on the Copper Clays which concentrates on the Blocks was completed by Wade (1957b), but this appears to have been temporarily mislaid until it was mentioned by Flitcroft and McKeown (1992). The MLMRC's work is described in later sections of this report.

5.2 Lyell Consols

The Mt Lyell Blocks Mining Company was the most successful company on the field after the MLMRC. Other Copper Clay mining companies fared badly by comparison with the Blocks and much less information is available.

The first mention located of the Lyell Consols Mining Company NL is by Russell (1898). This was one of James Crotty's companies and preliminary work was being undertaken in 1897. It appears from early publicity that Crotty expected to find additional North Lyell type rich bornite ore on the leases, but despite tunnelling, this was unsuccessful.

Schloesser (1900) reported that hematite and gossan carrying native copper were found in the Consols tunnel under the Linda Valley Creek but the value bulked under 2% Cu. Location is shown on Figures 1 and 8 and the extent of underground workings on Figure 24. There were rumours of a production start from early 1906 and a similar mill to that at Lyell Blocks was built during 1907.

The Consols Plant consisted of puddling machines, a 5-head battery and Wilfley concentrating tables. Commissioning began in the September quarter of 1907, but alterations were necessary and underground development continued until production commenced in the September quarter of 1909. The mine remained in production for two years until 10 September 1910. A total of 23,600 tonnes of ore for 185 tonnes of copper was produced. Production statistics are shown in Table 1 and the concentrate information in Table 2.

The average treatment rate of 236 tonnes per week is significantly less than the 1,250 predicted which must have made the operation a financial failure. It is of interest that actual concentrate grades averaged 61.5% and that the Consols did not show a similar gradually declining grade trend to the Blocks. It is thought that problems with ore supply or plant performance were the main reasons for their financial position.

Mine closure was again caused by ground weakness. One day in August 1910, the miners heard the ground creeping and the timbers groaning so they promptly left the mine. A few hours later, creek water broke into the workings which were so badly damaged that the mine had to be abandoned (Blainey, 1993, p. 193). Development including diamond drilling for which we have no records was continued until about June 1911, after which there are no entries in the PMIT records.

Reserves at closure were estimated to be 95,600 tonnes at 3.0% Cu (Wade 1957b). The grade is thought to be over-stated, and a much larger area was mineralised than the reserves suggest.

5.3 King Lyell

The history of King Lyell is also not very well covered by available documents. The presence of Native Copper was mentioned by Thureau in 1886 and the ground was probably pegged soon after the discovery of the Iron Blow in 1883. The Blow is located only 200 metres uphill to the southwest (Figure 1). Reid (1970) mentions that the King Lyell company was sluicing copper from Cooney's Creek as early as June 1895. The deposit was well described geologically by Power (1891, p. 32), and Schloesser (1900) mentioned that the deposit had been mined by hydraulic sluicing by the King Lyell Gold and Copper Mining Company NL for the previous 3 years.

The main areas sluiced to shallow depth were on the Southern Slopes of Pioneer Spur (Figure 30). Reid (1970) comments that Blainey's personal records amount to a total of at least 150 tonnes of copper on a very conservative reckoning. Like the Blocks, King Lyell was also probably selling copper before the MLMRC commenced production on 25 June 1896 (though they had produced Ag-Cu-Au concentrates from the Mt Lyell Bonanza in 1893/5).

In November 1901, King Lyell was purchased by the MLMRC for £2,600 as a prospective overburden dump. They set about exploration by the diamond drilling of four holes, ML8-12, in 1902 and the development of the No. 5 Adit for the Iron Blow Mine (Figure 15).

Apart from the diamond drillholes, trenches, sluicing, 2 tunnels and several shafts help to define the mineralised area. There was also Mr Batchelor's prospecting shaft, recommended by Thureau (1899; 1900), put down in 1903 and collapsed soon after. Thureau (1900) also suggested that "the boys in the district might be induced to collect scrap iron for copper precipitation from the mineral waters impregnated with dissolved copper."

Production statistics are recorded in Table 1 from which it can be seen that King Lyell has been the second largest copper clay producer, despite its lack of underground mining. Reid (1970) wrote a comprehensive review of King Lyell in which he recognised the potential and recommended an initial programme of 10 vertical diamond drillholes on 122m centres for a total of 945 metres. If encouragement was obtained, the second phase was to consist of 61m spaced holes. Four of these holes were drilled in 1970/71 (KL1, 2, 13 and 16) but the results were disappointing and have never been written up.

Reid (1970) was aware of problems of high core loss in the 1902 drillholes and also of contamination by material falling down the holes. He concluded that with unrepresentative sampling and errors in assay techniques, the information should not be used for evaluation purposes. Despite the unreconcilable factors, Reid thought that the results were of considerable interest.

5.4 Balance Shaft

The Balance Shaft is shown as a separate deposit on the Queenstown 1:25,000 geology map and receives some separate comments in PMIT. However, it was really part of the Lyell Blocks orebody to which it may be connected by undiscovered ore.

5.5 McDowell's Prospect

This prospect has been included with the Copper Clays Group for geographic rather than geological reasons. The old open cut is located just east of Figure 1 at about 5600N/7500E. The prospect is shown on the Queenstown 1:25,000 sheet as lying on the North Lyell Fault and is similarly described by Sillitoe (1984, p. 21).

The prospect was explored by Bird (1985), who decided that the North Lyell Fault was really 400m further up the hill. He suggested the epithermal mineralisation was associated with a hydrothermal explosion crater which created free-standing Owen Cliffs perhaps 1,000m high. Together with Bird's other explosion crater at Gormonston, these rocks were thought more likely to be glacial sediments by Arnold (1985), Sillitoe (1985) and Flitcroft and McKeown (1992).

Sillitoe's (1984) description of McDowell's appears realistic, that the gold mineralisation is associated with quartz vein stockworks parallel to the North Lyell Fault. There have also been suggestions that similar features are present on the south boundary faults of the Linda Graben. Together with the Gold Creek alluvials, there appears to be a gold source in the area related to the North Lyell Fault Zone, which is an important aspect of the metallogenic history of the area.

5.6 Lyell Pioneer

Until a recent discovery of old reports, virtually nothing was known about this prospect. It's location was known from Figure 24, but it seemed to be more an alluvial gold working than anything else. The report by Cundy (1901a) included a very useful map which has been redrawn as Figure 23. Cundy explained that there was no major mineralisation in either the main or in Calligan's Tunnel. However, he did report quartz veins with traces of gold near the contact of the Owen Conglomerate with the Lyell Schists.

Also Cundy reported a gossan zone near the entrance to the main Tunnel (Figure 23) which he compared with the mineralisation at the Lyell Blocks. A shaft had been sunk to 100 feet on the eastern part of the property through surface clay, pug and loose boulders, but the ground proved too heavy for the timbering and collapsed.

In this area, the clays have been washed off the Spurs (Wade, 1957a) and the flanking valley to the north which may turn out to be as prospective as that to the south (which hosts the King Lyell deposit). The Lyell Pioneer area is an exploration target, particularly along the Zeplin Fault zone as shown in Figure 1.

5.7 Mt Lyell Extended

This area has been named as a separate prospect, mainly by virtue of it's tenure history, being in James Crotty's stable at the turn of the century (Russell, 1898). The property contained two prospecting tunnels. Location is shown on Figure 23, but no detailed map of the workings has yet been found.

Cundy (1901b) mentions that in the 183m long lower tunnel, minor chalcopyrite in schist and a 2 foot wide vein with galena were located but did not persist more than a few feet. A gossan is present at the entrance to the lower tunnel containing shows of native copper but the workings had collapsed by 1901. PMIT makes mention of Lyell Extended in 1901, 1908 and 1909, but apart from suggestions of appreciable native copper, no production is known.

6. GEOLOGY

Some previous geological descriptions have suffered from being a mixture of fact and theory. The most useful descriptive accounts found are by Thureau (1886), Power (1891), Batchelor (1902; 1904), Gregory (1905), Wade (1975b), Solomon (1969) and Reid (1970). It is thought vital to distinguish between observation and deduction so sections 6.1 on host rocks and 6.2 on mineralisation are as factual as possible and any deductions are made in section 6.3 on ore genesis. The writer has studied any information available, visited the deposits, looked at drill core and produced summary drill logs and drill sections for most of the holes drilled. In this section, the Copper Clays are treated as a group as similar features have been observed at many deposits and the report would otherwise be too repetitive.

6.1 Host Rocks

Regional

The known Copper Clay deposits are restricted to altitudes of between 200 and 600m at the Western end of the Linda Valley or against the eastern flank of the Lyell Saddle. They are located in the Western Linda Graben, east of the Great Lyell Fault, south of the North Lyell Fault and north of the Owen Spur Fault. The deposits are located under water courses in valleys coincident with synclines and WNW trending Tabberabberan faults.

Ore Host

The immediate host to ore is clay, thought to be remnant clay after the dissolution of the calcium carbonate component of the Gordon Limestone. To test this assumption a series of XRD mineral determinations were made on samples of mineralised clay and on a residue after weak sulphuric acid leaching of Halls Creek Limestone. This was

prepared by Rob Gregory at GMA's Perth laboratory from samples collected by Morrison (1995). Mineralised clays were found to be mineralogically similar to the Gordon Limestone residue being composed of quartz, muscovite and kaolinite (Appendix 8) as was also found by Solomon (1969, p. 44).

There are references to mineralisation in the Gordon Limestone itself, but these do not form a large part of the mineralisation in sight. There is a tendency, best seen in the long sections of Figures 14 and 19, for the better grade clays and undissolved Gordon Limestone to be incompatible.

Gordon Limestone

Regionally, the Gordon Limestone commonly contains a sequence of basal interbedded limestone and sandstone followed by 300-500m of micritic impure limestone (Banks and Baillie, 1989; Calver, 1995). Recent work by Morrison (1995) at the Halls Creek Quarry has shown that the limestone typically consists of about 40-70% calcium carbonate and 25-50% clay. The formation is not typically pyritic. Also, the sequence is mainly massive with few shaley interbeds. A middle Ordovician age has recently been established for the Pioneer Sandstone (Calver, pers. comm., 1995) which is similar to the Gordon Limestone. This suggests a time gap of some 10-20 Ma associated with the Haulage unconformity.

Ore Footwall

Assays of interest normally cut out in the Pioneer Sandstone which acts as a footwall to copper clay mineralisation. Often, the sands are poorly consolidated. Anomalous chromium to 0.43% has been recorded, for instance, in LB09 at the Consols. In the vicinity of the Copper Clays, the Pioneer Sandstone is generally only 10-15m thick with a basal conglomerate. This passes unconformably into either conglomerates or sandstones of the Owen Conglomerate which are usually strongly hematitic.

Ore Hanging Wall

In the mineralised positions, upper parts of the Gordon Limestone are not seen. If present, the overlying stratigraphic unit is the Crotty Quartzite of the Silurian Eldon Group, but this has not been mapped in the Linda Valley.

There is often no hanging wall or only recent fluvioglacial sediments present. This makes it possible that part of the copper clay mineralisation was eroded during Pleistocene glaciation.

6.2 Mineralisation

Ore Minerals

From the drill logs and mining history, it is concluded that there is one dominant ore mineral, namely native copper and two minor ore minerals, namely cuprite and chalcocite. No thorough work has been undertaken to quantify the various minerals present and this will be a difficult process as the ore is not homogenous. Although the

sulphides covellite, bornite and chalcopyrite are present, they are often contained in variably sized sub-angular to sub-rounded clasts suggesting they represent detrital material from the orebodies up slope.

Markham (1966; 1968) made heavy mineral concentrates from drillholes LB13, 26, 28 and 31 as it was impossible to prepare polished sections. The concentrates included the following minerals:

Pyrite was the most abundant sulphide mineral occurring as discrete euhedral crystals averaging 0.03mm in size. They may show partial replacement by chalcocite, digenite or covellite. The textural features suggest a process whereby copper in solution is precipitated by reaction with sulphides. Markham thought the pyrite was authigenic (grew in situ).

Chalcocite occurs as marginal replacements to pyrite and as framboidal aggregates of spherical, oval and crescent-shaped grains. Digenite, covellite and bornite are also present in the framboidal aggregates.

Native Copper and Cuprite are claimed by Markham to be more typically observed in the near surface portions of the deposits and grade into sulphides at depth. This was also suggested by Power (1891). However, this is neither true from the drill logs nor from the Lyell Blocks production concentrate grades. In his own samples from depth, Markham observed native copper and cuprite and assumed that they have formed by oxidation of primary copper sulphides.

Sphalerite and Galena; a few isolated grains occur which are not associated with copper sulphides. The sphalerite is typically a honey-yellow colour. Wade (1957b) also noted black clays derived from the Gordon Limestone containing fine, widely disseminated grains of galena.

Other mineralogical descriptions by Edwards (1939, 1958) are also useful. During a field visit in March 1958, Edwards collected some limonite nodules (geode like) from King Lyell with internal cavities up to 2cm across encrusted with interlayered seams of native copper and cuprite up to 0.5mm wide. Open or filled cracks lined with these minerals lead into the cavities. There are also sheets of native copper filmed by malachite. Another group of specimens from the Blocks consisted of porous sheet-like slightly colloform bands of cuprite studded with minute inclusions of native copper.

Other relevant previously-mentioned descriptions include native copper occurring as films infilling joints and cleavages, including joints in the footwall Pioneer Sandstone and as chalcocite encrusted dendritic masses. Massive nuggets of native copper of up to 75kg were encountered at the Blocks (Blainey, 1993, p. 187). In his drill logs, Batchelor (1902) records pebbles of native copper and in hole ML12, the only mineralised Gordon Limestone interval with no associated clay mineralisation, fine pyrite-chalcopyrite and native copper are disseminated in silicified limestone.

Determination of the relative quantities of each mineral is difficult as they are not spread homogenously through the deposits. The best evidence is from the deposit's

metallurgical history. To be able to produce concentrates averaging 69% Cu with 72% recovery, the minerals involved must be high copper species.

Since chalcocite is very fine and hard to recover, this indicates that native copper and cuprite are the main minerals present which agrees with the drill logs. An educated guess at the overall proportions would be 70-90% native copper, 5-20% cuprite and 5-10% chalcocite.

Associated Lithologies

Not all of the Copper Clays consist of clay. There are two other major components which both tend to be unmineralised. These are limestone and detrital sediments. The limestone may be massive or vughy limestone or limestone gravels. Dolomite has been logged, but this may include ironstained limestone. Various unconsolidated detrital sediments are mixed with the clays. They vary from silt to sand, grit, pebbles, cobbles and boulders. Examples of all major local lithologies have been seen, including sub-angular to rounded clasts of cleaved and mineralised Lyell Schists, hematite, quartzite, quartz, conglomerate, sandstone, bornite and gossan. These lithologies may occur as wide intervals, eg: up to 100m of hematitic sand in LB37 or as isolated boulders in clay.

Where limestone is present, there are no descriptions of any typical skarn mineral assemblages or any clay mineral hydrothermal alteration. Occasional quartz veins containing sulphides are present under the Gordon Limestone. For instance, in hole NL1102, in a zone of altered Owen Conglomerate under Lyell Blocks, an interval of 46m at 30% $BaSO_4$ contains narrow quartz-barite-galena veins assaying up to 0.37% Pb and 9 g/t Ag over 2m. These assays are diluted by non-vein material in the assay interval. Separate galena and chalcopyrite bearing quartz veins are present in Owen Conglomerate at Mt Lyell Extended. Gold is present in quartz veins at Lyell Pioneer.

There is insufficient Gordon Limestone core available to be sure if the fresh limestone normally carries sulphide minerals. There are several holes with long lengths of unmineralised limestone, eg: LB6, 7, 32, 33, 34, 35 and NL1101.

Of the 48 holes drilled at the Blocks, Consols and King Lyell only 16 have intersected limestone and clay. Of these, only 5 have some mineralisation in limestone, whereas 15 contain mineralised clay. Only one hole, ML12 (Batchelor, 1902), near the eastern margin of the King Lyell deposit, had mineralised limestone with no associated mineralised clay (Figure 19).

Relationship to Structure and Metamorphism

The deposits are hosted predominantly by poorly lithified to plastic clay (cf. history of ground support collapse). No descriptions of any effects of metamorphism such as schistocity or metamorphic mineral development in the deposits themselves have been sighted.

The limestone precursors were present in synclines with common axial planar faults which have similar orientations to the S_2 schistocity in Lyell Schists and Owen

Conglomerate; where it can be determined (Figure 1). These synclines have acted as hosts for topographic lows which have allowed limestone and clay to be preserved.

It is thought, for instance by Wade (1957b), that former limestone on the Whaleback, Linda, Pioneer and Gormanston Spurs has been converted to clay by carbonate dissolution and then either scraped off by ice or washed off by the heavy rainfall. This process has left unusual exposed fold surfaces as if they had been carefully excavated.

6.3 Ore Genesis

Review of Previous Ideas

Various theories have been advanced to account for the copper clay mineralisation. Gregory (1905, p. 140) pointed out that the ore deposition at the Blocks was subsequent to the main transverse faulting. At the turn of the century, Power, Batchelor and Cundy promoted the view that the deposits were products of in situ decomposition. There was a long gap before anyone even commented on the Copper Clays until Alexander (1953) claimed that their relationship to structure indicated a hypogene origin. Wade (1957a) and Wade and Solomon (1958) thought that the deposits occurred in shales of the Gordon Formation that had been mineralised by hypogene hydrothermal activity and then oxidised. However, Wade (1957b) noted that native copper is forming at the present time, particularly after rain by adsorption of copper on the clay from acid solution.

Markham (1966) suggested that the textural features strongly suggested the precipitation of iron and copper sulphides in a low temperature environment. He said that in view of their proximity to major sulphide deposits, it was reasonable to assume that they were the source of the iron and copper. The primary orebodies could have contributed metals to the Ordovician clay sediments by processes of leaching and transportation as soluble iron and copper sulphates. Markham thought it was difficult to decide when the process occurred, either syngenetically in the Ordovician or epigenetically in present day climatic and topographic conditions. A syngenetic Ordovician genesis was his preferred model.

Solomon (1967) suggested that the Iron Blow and associated hematite were gossans formed by oxidation during the early Ordovician. His descriptive paper on the Copper Clays (Solomon, 1969) is, in the writer's opinion, the best available summary; though Solomon seems unaware of the results of the 1963-65 drilling of holes LB1-37. Solomon's genetic views were multifaceted and included an enrichment of copper and iron sulphides during sedimentation. He thought a Tabberabberan age via hydrothermal remobilisation could not be ruled out. Solomon also considered that modern low-temperature precipitation and exchange with iron and/or limestone from copper in groundwaters derived from the deposits uphill was possible. In summary, he concluded that "the Copper Clay deposits are a combination of indigenous and transported gossans."

Reid (1970; 1975) followed the views of Markham (1968) and Solomon (1969).

In the early 1980's, a group of geologists from the new mine owner, RGC, soon dispensed with many of the views established after years of careful mapping, research and debate.

Bird (1982a; b), Brook (1984), Sillitoe (1984; 1985) and Arnold (1985) resurrected many dormant Tabberabberan epigenetic ideas and applied them to most of the orebodies on the field; including the Copper Clays. Sillitoe asserted that much of the mineralisation was structurally controlled by faults and that the Lyell Blocks mine exploited a steep 10-15m wide structure. If the mineralisation was structurally controlled, then Markham's (1968) syngenetic limestone-shale concentration model was unlikely. Also, any Tertiary supergene addition presupposed pre-existing sulphides. Sillitoe concluded that the Copper Clay deposits support hydrothermal sulphide mineralisation at Mt Lyell in post Gordon Limestone, probably Tabberabberan, times.

Bird (1984) stated that recent work at Mt Lyell had shown that volcanogenic models were erroneous and that the mineralisation is related to hydrothermal events that post-date the Gordon Limestone and Crotty Quartzite and is therefore of probable Devonian age. The two main styles of deposit likely in the Gordon Limestone were vein or replacement mineralisation. Vein deposits were likely to be of native copper-silver type with 2-4% Cu, 30-100 g/t Ag and possibly 1-2 g/t Au. Potential for up to 20 million tonnes of this style of material was recognised. Bird tested his model by drilling diamond holes NL1101 and 1102, totalling 1,024 metres, under Lyell Blocks. Apart from retesting the zone where the Lyell Blocks main shaft had terminated in 1915, the holes intersected very little mineralisation. The Copper Clays sections were not assayed and the best results were from the 46m of 30% barite section reported above.

Arnold (1985) recognised that the puggy ex-Gordon Limestone clays are clearly much modified from Paleozoic precursor limestones. He recognised, as did Sillitoe (1985), that pre-glacial deep weathering and periglacial and fluvioglacial processes have caused additional complications in the area. Arnold commented that the 'black pug' showed evidence of slumping, mixing and resedimentation with the glacial sediments. He provides a concise review of various models, but, apart from supporting Bird's ideas, he makes no firm conclusions.

Conclusions from Previous Ideas

Previous workers' ideas can be grouped into the four most popular models:

1. Decomposition
2. Syngenetic
3. Epigenetic - Hydrothermal
4. Natural Hydrometallurgy

1. In Situ Decomposition of Lyell Schists

The decomposition model was accepted by most of the early workers, including Power (1891), Cundy (1901) and Batchelor (1904). The model involved in situ decomposition of schist to pug and near surface oxidation of chalcopyrite to native copper. This process is untenable because Lyell Schists do not

decompose; stable detrital schist clasts are present in the copper clays. Pug is now widely accepted as a residue after weak acid dissolution of Gordon Limestone calcite.

2. Ordovician Syngenetic Deposition with Gordon Limestone

Markham (1966; 1968) and Reid (1970; 1975) were the main protagonists of this model which was also considered possible by Solomon (1969). The syngenetic model involved sulphide deposition during sedimentation in a shaley or mudstone horizon at the base of the Gordon Limestone. Metal-bearing solutions are in part derived by low-temperature leaching from the adjacent large deposits (Blow, West Lyell, Corridor). Flitcroft and McKeown (1992, section 6.4) wanted to carry out lead isotopic work to determine if the ores were Cambrian in age, ie: older than their host rocks.

This model is based on several misconceptions on the nature of the deposits:

(a) that the surface native copper grades into primary sulphide mineralisation at depth (drilling and mining have shown that it does not);

(b) that there was a shaley zone at the base of the Gordon Limestone. This is unsupported by regional stratigraphic evidence and the 'clays' are thought to be residues from mass formational dissolution; and

(c) the ore has formed in a zone which is of post landscape, probably Cainozoic, age.

3. Devonian Epigenetic Hydrothermal Models

The epigenetic model was mentioned by Alexander (1953), Wade (1957a; b), Wade and Solomon (1958), but principally developed by Bird (1982a; b; 1984), Brook (1984) Angus (1984) and Sillitoe (1984; 1985). The model proposes primary epigenetic vein-type mineralisation associated with strong barite-hematite metasomatism during late stages of the Devonian Tabberabberan Orogeny. The model also predicts vein mineralisation in structural zones cutting the Owen Conglomerate. Later, in situ oxidation has produced native copper at surface.

This model also fails to take account of the drill log and mining evidence of predominant native copper. Furthermore, there is little evidence for the associated veins. Practically all the Gordon Limestone drilled is unmineralised and the barite-galena veins discovered under the Blocks and Mt Lyell Extended can hardly be responsible for copper mineralisation.

With the process suggested, either some skarn or clay mineral hydrothermal alteration would be expected, but has not yet been identified. Also, Cainozoic mass limestone dissolution is not compatible with the Tabberabberan timing of this model. Markham's (1966) recognition of low temperature framboidal textures is another problem. In fact, there is so much conflicting evidence that

this model is regarded as unacceptable. If the model is invalid, ore potential in the Owen Conglomerate is also downgraded.

4. Natural Cainozoic Hydrometallurgy

Elements of this model have been suggested as a possibility by Wade (1957b), Markham (1966), Solomon (1969), Arnold (1985) and Hills (1990), though none of these authors chose recent concentration as their preferred option. The model proposed here involves a source of copper and acid solutions from oxidation of primary deposits such as the Blow, West Lyell and the Corridor on the Lyell Saddle. The solutions are then transported downhill until they meet carbonate from the Gordon Limestone which neutralises the solutions. This makes the dissolved copper unstable and it is readily precipitated by reducing agents in the clay such as carbon, kerogen, bacteria, iron oxides or humic acid.

Objections to this model have been based on presumed structural control, the presence of sulphides at depth and difficulties with low temperature copper precipitation mechanisms.

Evidence for the hydrometallurgical model is the distribution of native copper throughout the clay formed by Cainozoic mass dissolution of Gordon Limestone. This process chemically aids copper precipitation at low temperatures, producing minerals and textures consistent with the model and observations. Precious metal contents in the Copper Clays are generally low (see Tables 4 and 5) which would be expected in the low temperature model, but not in the higher temperature syngenetic or epigenetic models. The lack of associated mineralised limestone, veins and metamorphic features are all also consistent with the natural hydrometallurgical model which is considered the most acceptable current theory.

Several chemists, geochemists and metallurgists have been asked if they can suggest a likely precipitation mechanism consistent with the geological evidence. At one stage it was thought that the cation exchange capacity of montmorillonite clays (Bush and others, 1993) could explain the mechanism. However, XRD work on samples of mineralised clay (Appendix 8) found no smectite group clays, so a cation exchange process is unlikely. Current thoughts on possible mechanisms include the reaction between humic acid and copper sulphate precipitating copper metal (Baker, pers. comm, 1995), or a type of natural electrowinning process where copper is forced out of dilute turbulent solution by weak natural electric charges. The subject warrants further research.

Timing of Mineralisation

The sheets of copper in joints, dendritic copper nuggets and pebbles of native copper all suggest copper deposition in a vadose zone post landscape formation. The mass dissolution of calcium carbonate of the Gordon Limestone is also likely to have occurred after the formation of the landscape. The process would be much slower, if at all possible, if the limestone was in the phreatic zone below a poorly oxygenated fresh water table.

The abundant oxidising sulphides on the Lyell Saddle are naturally producing both surface and groundwater solution flow which is acid, oxygenated and cupriferous. The reaction of pyrite, chalcopyrite, water and oxygen produce goethite (gossan with residual precious metals), sulphuric acid, copper sulphate and iron sulphate. The recent CMT environmental management report showed that a daily mean of over two tonnes of copper is being transported downstream from Mt Lyell (Thompson and Brett, 1994, Figure 4.22). Mining has caused accelerated copper transport rates, but even at 0.1 tonnes per day and only a 5% of total copper precipitation rate, it would only take some 27,000 years to accumulate the 50,000 tonnes of copper identified in the three main deposits to date (Appendix 3).

Since glacial deposits of up to 0.8 Ma age overlie but do not contain interesting copper intersections (Appendix 1), the mineralising process is thought to be mainly of pre-Quaternary age. As the current landscape is thought to have developed during the Cainozoic, it is likely that the Gordon Limestone dissolution and copper deposition took place during this period, and is probably currently continuing.

Limestone Dissolution Process

While trying to understand the dissolution process, the writer contacted Dr Armstrong Osborne of Sydney University who is an expert on limestone caves. He is currently working on a model for major cave system formation which suggests the rate of dissolution is exponentially increased if oxidising sulphides (generally pyrite) are present in the groundwater catchment (Osborne, 1994). The mass dissolution of Gordon Limestone at Mt Lyell is clearly an extreme example of this process.

According to Osborne (1984; 1986) cave sediments are very common, and although an areally minor depositional environment, they are among the most complex in terms of range of processes and geometry of products. Cave sediments in Eastern Australia are generally of Cainozoic Age with sediments generally derived from outside the cave. A number of unusual sedimentary features can develop such as:

- high initial dips
- diamictites with angular and rounded clasts
- results of floor and/or roof collapse
- lateral facies changes
- circular and vertical unconformities
- stratigraphic reversals
- non metamorphosed sediments unconformably underlying older bedrock

Such features help explain the complex sequence of interbedded clay, limestone and sediments seen in copper clay diamond drill core.

In fact, it is surprising there is any limestone left in the Linda Valley. This can be explained by the channelling effect of groundwater flow through established open space. It is highly likely that during the dissolution process a complex and constantly-evolving cave system developed which included major collapse and breccia formation. Another common feature of cave systems is their sediment content, including boulder beds if a

source and access are present. During Pleistocene glaciation, it is even likely that diamictic sediments were squeezed through subterranean cave systems under pressure (K Kiernan, pers. comm., 1995).

The gradual dissolution of limestone can be likened to the gradual melting of ice during a glacial retreat. Examples of the complex types of sedimentary accumulations developed are illustrated by Fitzsimmons and others (1993, Figure 32).

Implications of the Model

As the deposits occur in faulted synclines, some may claim they exhibit structural control, though this is an inherited control on valley position. The important control on mineralisation is thought to be by natural hydrometallurgy, ie: the dissolution, transport and precipitation mechanisms that have allowed the economic concentrations to be formed.

The model has a predictive capacity for orebody location, shape and grade variation. The deposits will have an overall synclinal chute-like geometry with major variations in grade caused by the copper nugget effect and paleofluid channelling zones. The highest grades may be expected near source with a gradual tapering off in grade with depth. Zones of potential mineralisation are present all along the western limit of clays as shown on Figure 1.

The deposits will be very complex in detail having evolved through a cave stage where irregular channels and collapse followed by dissolution have occurred. Before and during glacial stages, rounded sediments of varying grainsize will have been transported through caves and mixed up with angular collapsing limestone debris.

Ore fluid channelling will have led to adjacent high and low grade zones. Correlation between adjacent drillholes will be very difficult and numerous voids are still likely to be present. Isolated boulders in clay and porous gravel zones will present severe drilling difficulties.

6.4 Comparisons With Other Native Copper Deposits

Although copper is one of the commonest native elements, it generally occurs as a supergene oxidation product rather than as the major mineral in ore deposits. In order to gain helpful insights from other native copper deposits, a search was made on the main international computer data bases at the AMF.

The most common deposits of native copper are associated with low-grade metamorphism of basic volcanics. The best example is the Michigan Native Copper District on the Keweenaw Peninsula, USA (Lindegren, 1913). Since 1845, over 5 million tonnes of copper have been produced from the 1095 Ma Portage Volcanics and interbedded sediments. Hosts for this production are approximately 58% volcanics, 40 conglomerates and 2% veins with 96% coming from a 45km long, 3km wide zone. Ore is principally mined underground, historically at grades of up to 4% copper, but more recently at grades of between 0.5 and 1.0% copper.

Most authors have concluded that the deposits formed at about 1050 Ma during prehnite-pumpellyite regional metamorphism at 250°C and depths up to 3.5km. Copper has been leached from basic volcanics containing about 70 ppm Cu during dehydration reactions and transported as chloride complexes containing 1500 to 2000 ppm Cu. Solutions have percolated upwards towards lower temperature levels where hydration was dominant. Open space in both amygdaloidal basalts and porous conglomerates has been filled to produce 'pseudo' stratiform 'manto' (blanket) type orebodies. The deposits are silver rich and contain some native silver. Where constrictions have concentrated fluid flow through, grades are higher.

Other native copper locations with geological similarities to the Keweenawan deposits are:

- Victorian district of Zimbabwe (King, 1989)
- Carboniferous Clyde Plateau lavas of the Midland Valley of Scotland (King, op. cit.)
- Seachondong basalt flows of southern Korea (Lee and Kim, 1970).

Additional occurrences of native copper are known in serpentinised peridotite from central Japan (Kahehira and others, 1975) and also from central Sweden and at the Franklin Mine, New Jersey, USA (Burke and Dunn, 1988). A paragenesis of low temperature (300°C) alkaline hydrothermal alteration under very low sulphur fugacity conditions has been suggested to control precipitation.

Neither the Keweenawan deposits, nor any others for which descriptions have been located, seem similar to the Mt Lyell Copper Clays. They are therefore thought to be a very rare to unique type of ore deposit.

7. RESOURCE ESTIMATES

It is surprising that with 48 diamond drillholes completed, and many significant intersections (Table 3), that nobody appears to have yet carried out a resource estimate. In the most recent detailed Copper Clays MLMRC report, Reid (1970, p.7) claims that as the data is unrepresentative and contains errors, it should not be used for evaluation purposes. While there are certainly problems with the data quality, this is the case with most resource estimations. The writer believes that geologists have a responsibility to make the most of the information available while reporting concerns caused by deficiencies in the data. Since 1970, the AIMM resource reporting code has been developed and this gives guidelines for dealing with Copper Clay type situations.

The procedure adopted here is to analyse resource assessment criteria with a view to suggesting improvements. Detailed results for each main deposit are then reported in sections 7.1 to 7.3.

Estimation Technique and Interpretation

A total of 48 diamond drillholes for 4,900 metres have been completed between September 1902 and July 1984. Drill logs are stored in the Lyell Blocks blue book in the

geology office safe. Summary logs have been produced for this work (Appendix 1) and lists of intersections compiled as Table 3 and Appendix 2.

These logs have been plotted on a series of 13 sections at the most appropriate intervals to reduce the length of projection onto each section. Intersections have been bulked as much as possible and a low cut off grade of 0.1% Cu has been used to simplify the interpretation. Cross sections have been interpreted and areas of mineralisation on each section measured by planimeter. Average grade assigned is a length weighted average of the intersections inside each perimeter. A tonnage factor of 2.1 tonnes/m^3 has been used to account for the low density clays and voids. Tonnages for each section have then been added. Resources obtained are detailed in Appendix 3 and summarised on Table 7.

The orebodies have been defined by clustering of holes with significant drill intersections and orebody perimeters drawn to divide ore bearing holes from barren holes. All holes within the mineralised perimeter for each orebody have been included in the estimate.

If it were not for the problems of core recovery, the resource might be able to be classified as indicated. However, the recovery is so poor that classification as an inferred resource is preferred. In the writer's experience, poor core recovery seldom 'makes' or 'breaks' an orebody. In detail, the recovery is quite variable and available estimates for each intersection are given in Appendix 2.

Drill Hole Density

Currently, a total of 36 holes and various underground workings define the three resources. Spacing is somewhat irregular, possibly due to difficulties of site access, but from the sections and workings, a high confidence in ore continuity is achieved. This is also because there are no examples of unmineralised holes within ore perimeters. Holes KL16 at King Lyell and LB37 at the Blocks are closest to being negative. There are sufficient holes and continuity for an inferred resource.

Hole Location Accuracy

For all but the old ML holes, locations have been surveyed. The five Mt Lyell holes drilled in 1902 are probably only accurate to ±5m, but this will not affect the current conclusions.

Drilling Techniques and Recovery

All holes, except the five 1902 holes, have been drilled since 1963 with modern types of diamond drill equipment. There is probably no ideal rig to drill the difficult mixture of limestone, clay, running sand, gravel and boulder clay that is present. In observing the core that has been recovered, it is surprising to see how much unconsolidated sand, gravel and some cored boulders has been recovered. Overall recoveries over mineralised intervals vary from 20 to 100% with an average from Appendix 2 of 56%. A void space of 20% has been assumed to reduce the tonnage factors so that the real recovery may be 56/80, ie: up to about 70%.

Such a recovery could still give a good representation of what is in the ground, depending on whether the 30% lost core was of lower, similar or higher grade than the material recovered. In the general absence of sludge assays, there is no way of knowing whether the lost core is of consistently higher or lower grade.

The only sludge assays located were from the 1902 drilling of hole ML09; these are reported in Table 4. Surprisingly, these sludge assays are significantly higher grade than the recovered core. However, it is doubtful if the sludge assays are representative – so they do not really help.

The writer believes the most likely lost material is unconsolidated sediment; which is generally unmineralised. This would have the effect of increasing the average grade of material recovered. Because of the difficulties of drilling copper clay type mineralisation, it is unlikely the problem will be eradicated. However, it should be possible to do better and quantify the errors so that risk can be reduced.

Sampling Techniques

These are not well documented. From drill core remaining, it can be seen that it was not always cut in half, even if it was possible to do so. As much of the core is unconsolidated, it can only be assumed that the samplers attempted to select representative material for analysis.

Assay Techniques

These were similar to others in use at Mt Lyell. It is clear from the logs that if native copper is present, it can often be visually seen down to very low levels of 0.1% Cu, though for some intervals of plus 0.5% Cu, no copper minerals are mentioned. This could be caused by the problem of identifying chalcocite which is hard to distinguish from black clay, carbon and manganese staining.

The main problem, illustrated on Table 4, is reproduceability due to difficulties in preparing a homogenous pulp from malleable native copper. The only data available are from the 1970 King Lyell drilling. They show poor comparability, so it is concluded that there is an assay reliability problem. It is not known whether this would lead to generally higher or lower results. This would depend on how the assay charge was taken. It is thought most likely that with unpulverised native copper present, the charges would be generally of lower grade than the bulk pulp.

Tonnage Factor

Tonnage factors will vary from 0 for voids to 8.9 for native copper. The SG of the clay is probably about 2.6, but with the possibility of up to 20% voids an SG of 80% of 2.6, that is 2.1 tonnes/m^3 has been used in the estimation. Wade (1957b) reports work by Batchelor in 1902 who measured the SG of King Lyell clay at 2.08. However, Wade used a figure of 2.4 in his estimations. If he was right, the current estimate is up to 15% or 1.2 million tonnes low on tonnes.

Table 4

Details of Sludge Assays and Re-Assays

Sludges

From 1902 drilling recorded in drill logs for Hole ML09

Interval (m)			Core Assay	Sludge Assay	Difference
From	To	Length	% Cu	% Cu	%
19.8	21.3	1.5	1.9	8.7	+ 358
21.3	22.9	1.5	2.4	15.0	+ 525
22.9	24.4	1.5	2.9	12.9	+ 345

No details on core recovery available.
No details of sludge collection method available.

Note That from 13.7 to 16.8m in Batchelor's Prospecting shaft located 15m to the northeast of ML09, a gossan zone was recorded and was estimated to contain 20% copper!

Re-Assays

Only carried out on 1970/71 King Lyell drilling

Hole No	Interval (m)			Original Cu Assays	Re-Assays	Difference
	From	To	Length	% Cu	% Cu	%
KL01	30.5	38.1	7.6	1.69	0.85	- 50
KL02	74.5	78.5	4.0	0.86	1.60	+ 86
KL16	19.8	42.7	22.9	0.049	0.042	- 14

Note These are bulked intervals. Individual assays may show more variation.

SG Weighted Assay Averaging

Another factor which can have a major influence on contained copper is whether composite assays are SG and length weighted rather than the normal length only weighting which has been used here. For instance, one meter of native copper and 1 meter of barren clay average 77% (8.9/8.9+2.6)not 50% Cu.

A more realistic example is to average 4m of clay at 0.1% Cu with an SG of 2.6 and 1m of mineralisation at 10% Cu with an SG of 3.3. Normal length weighted averaging gives 5m at 2.08% Cu, but the correct average is 5m at 2.48% Cu; which is a 16% increase in grade. It is thought likely that the average grade would be increased from 10-15% if SG weighted averaging were used.

Nugget Effect

Another effect which is difficult to evaluate is the nugget effect. It is likely that, as in gold deposits, a large proportion of contained copper is present in a small proportion of the volume. To properly evaluate this would require geostatistical knowledge of the range and nugget effect from a much higher drill density. This is not likely to be available. The factor is important if there is a desire to maximise copper production in the short term.

From the model developed, it is anticipated that high-grade fluid channelling zones of plus 3% copper are likely to be present. Deposits with high nugget effects are difficult to evaluate and some cutting of high grades may be appropriate.

Cut-Off Grade

A cut-off grade of 0.1% copper was initially chosen arbitrarily. The decision can be justified as follows: At a revenue of $35 per % Cu recovered, if treatment cost is $1.0 to $1.5 per tonne, then in an open cut situation, recovered copper needs to be over 0.03 to 0.04% to justify treatment as ore rather than waste. At this grade, recovery can be expected to be only 30 to 50%, so a cut-off head grade of 0.08 to 0.1% Cu is appropriate.

Quality of Data Description

Unfortunately, this leaves a lot to be desired as hardly any information or reports describing what has been done are available. The blue drill log books are vital, and do systematically record the most important data. However, logging has been carried out by many geologists and a degree of inconsistency is inevitable.

On check logging, the only problem found was an interval in the most recent King Lyell holes logged as quartzite which was found to be silicified vughy limestone with a fizzy reaction to 10% HCL. It is hoped that by simplification to only a few lithologies, the summary logs are as reliable as possible.

As no reports on drilling of 46 of the 48 holes were produced (or have not been referred to or located), it is thought that the MLMRC were not impressed by the results. When

they did carry out metallurgical testing, it is suspected they were considering treating the copper clays through their normal flotation process. As the flotation test results were disappointing, it is easy to see how they became disenchanted with the copper clays and did not regard a resource assessment as necessary.

7.1 Lyell Blocks

Blocks mining history is reasonably documented, but more detailed period reports are probably available through State Archives. Previous stoping history suggests a zone of higher grade mineralisation parallel to and under the gossan zone which was exposed in Copper Creek (Figure 26). Assuming mineralisation between the Blocks and Balance Shaft area is really continuous, the Blocks orebody is up to 400m long, 140m wide and 160m deep. Strike is NW which is not parallel to either the North Lyell Fault or the set of WNW trending Tabberabberan D_2 faults to the south. The body appears to be controlled by the topography and the natural hydrometallurgy model previously described.

Nine of the 16 holes drilled (for 2569m) outline the orebody. Of the 7 remaining, 2 were abandoned (LB3 and 36), 2 were not assayed (NL1101 and 1102) and 3 are some distance to the southeast of the orebody (LB1, 2 and 4). Of the 9 holes defining the orebody, only one, LB28, was drilled in the central higher grade zone. It achieved an intersection of 165m at 0.6% Cu which is the highest m% accumulation in all clays holes. The other 8 holes intersected significantly less mineralisation (Appendix 2.1).

There is good evidence for strong mineralisation on the old long projections (Figures 28 and 29) and from the complexity of the underground workings (Figure 27). It is anticipated that despite the ore extracted, after more holes are drilled, the average grade will be increased by relatively more strong intersections from the central core.

The Blocks resource of 4.7 mt at 0.5% Cu contains 23,700 tonnes of copper which is 48% of the inferred resource recognised in this report. Without the benefit of the 1960's drilling, Wade (1957b) estimated probable reserves of 1.63 mt at 1.75% Cu containing 28,500 tonnes of copper. He also recognised potential for an additional six million tonnes of ore above the Crown Lyell Quarry. Since this is largely outside the current inferred resource mineralisation, total resource potential of 10 mt at 1% Cu is recognised.

The few mildly anomalous silver and gold assays from copper clays are mainly from Lyell Blocks (Tables 5 and 6). There is a suggestion, at least with the anomalous Ag, that it correlates with higher sulphur assays. This implies that detrital clasts of bornite and chalcopyrite ore from North Lyell form a part of the Blocks mineralisation as recognised by Batchelor (1904).

Table 5

Gold and Silver Contents of Copper Clays ore from Production Records

Ore Source	Period	Tonnes	% Cu	g/t Au	g/t Ag	Note
North Lyell	Pre 1903	101,978	7.8	0.32	85	1
Lyell Blocks	1900/01	18,537	7.6	0.29	47	2
King Lyell	1903	21	40.42	bld	19	
	1904	2.0	1.09	Tr	3.4	
	1906	3.6	1.68	bld	Tr	
North Lyell	1904	59.1	0.50	bld	13.8	3
	1906	34.6	0.34	bld	11.0	3

Notes:

Tr = Trace.
bld = below limit of analytical detection.
1 Bornite-rich ore mined by the North Lyell Company.
2 Bornite-rich ore mined from Lyell Blocks - see Blainey (1993, p. 143).
3 Probably Blocks ore sold to MLMRC, possibly from the Copper Clays north of Lyell Blocks.

Table 6

Copper Clays Mineralisation, Copper, Pyrite, Gold and Silver Contents from Drill Intersections

Deposit	Hole No	Intersection (m)			Cu %	FeS_2 %	Au g/t	Ag g/t	Description of Interval from drill logs
		From	To	Interval					
Blocks	LB28	16.5	30.8	14.3	0.15	1.07	0.15	1.4	sand, clay, py cubes
Blocks	LB28	30.8	45.7	14.9	0.13	1.14	0.30	2.6	clay
Blocks	LB28	45.7	65.2	19.5	0.21	1.36	0.30	4.3	hematitic schist
Blocks	LB28	65.2	72.9	7.6	0.11	0.23	Tr	0.8	decomposed schist
Blocks	LB28	72.9	76.7	3.8	0.59	0.51	0.15	1.4	clay
Blocks	LB28	76.7	85.5	8.7	2.94	0.64	Tr	1.2	clay, fine diss NCu
Blocks	LB28	85.5	100.6	15.4	0.27	1.41	Tr	2.1	clay, limestone, py cubes
Blocks	LB28	100.6	112.8	12.2	1.09	1.17	Tr	1.2	clay, NCu
Blocks	LB28	112.8	121.9	9.1	2.37	1.63	Tr	12.4	clay, NCu
Blocks	LB28	121.9	129.5	7.6	1.06	0.78	0.15	27.7	coloured clay, NCu
Blocks	LB28	129.5	139.0	9.5	1.81	1.56	Tr	2.9	grey clay fine NCu
Blocks	LB28	139.0	152.4	13.4	0.35	2.05	Tr	3.5	grey clay NCu & py
Blocks	LB28	152.4	167.3	14.9	0.15	1.33	Tr	1.2	(clay with fine py in darker
Blocks	LB28	167.3	181.1	13.7	0.10	1.74	Tr	Tr	(layers
Blocks	LB30	9.0	21.3	12.3	0.21	0.20	Tr	2.9	clay
Blocks	LB30	21.3	35.1	13.8	0.34	0.45	Tr	3.1	clay with NCu
Blocks	LB30	35.1	45.7	10.6	0.45	2.67	Tr	3.4	clay with NCu
Blocks	LB30	45.7	54.5	8.8	0.30	3.58	Tr	2.8	clay with py
Blocks	LB32	12.5	24.4	11.9	0.16	0.01	Tr	1.2	clay with NCu
Consols	LB13	5.5	17.4	11.9	0.10	2.46	Tr	2.5	coloured clays, diss py
Consols	LB13	58.5	75.9	17.4	0.68	bld	0.15	3.1	clay, limestone, py, NCu
Consols	LB18	27.1	29.9	2.8	10.5	bld	Tr	Tr	grey clay, NCu
Consols	LB23	53.3	59.4	6.1	0.27	4.0	Tr	0.9	clay with fine diss NCu
Consols	LB23	59.4	67.1	7.7	1.54	bld	bld	Tr	clay with fine NCu

Abbreviations:

Tr = Trace
bld = below limit of detection
py = pyrite
diss = disseminated
NCu = Native Copper
FeS_2 is as recorded on logs and probably an under-estimate as some of the determined sulphur has been alloted to chalcopyrite.

The main problem with the Blocks, particularly during the exploration stage, is the presence of waste dumps from the various Corridor pits. These comprise Crown 3 and North Lyell waste dumps which together cover the northwestern part of the orebody.

7.2 Lyell Consols

Mining history is not well documented, but ore was developed on several levels and over a considerable strike length (Figure 24). In 1907, reserves were estimated at 95,627 tonnes at 3% copper (Wade, 1957b). The orebody trends approximately east-west at a similar orientation to the Consols and Whaleback Faults. It is up to 400m long, 160m wide and 70m deep and opportunities for immediate extensions are not obvious.

Lyell Consols is the most densely drilled of the three deposits, with 23 holes for 1646m (Appendix 2.2). Again, one hole, LB18, had by far the best intersection of 31m at 1.56% Cu and the mineralised body appears to be strongly zoned around a high-grade core. The total resource of 1.9 mt at 0.5% Cu contains 8,700 tonnes of copper which is 18% of the global resource. It is again predicted that after a more representative drilling pattern is established, a higher grade though lower tonnage resource of about 1.5 mt at 1% Cu will be established.

The long section view of Lyell Consols (Figure 14) is remarkably similar to that at King Lyell (Figure 19). It is expected that the overall geology is also very similar. The higher grade predicted for King Lyell may be a function of hole location to date, or if it is real, it is probably caused by King Lyell's closer proximity to a source of high grade solutions derived from the Iron Blow.

There are no known mine dumps overlying the Consols deposit which, apart from general site access difficulties, should be the easiest of the three main deposits to develop.

7.3 King Lyell

Although the mining history of King Lyell is different, with all previous production coming from sluicing operations, there are still sufficient shafts and tunnels to again be confident of ore continuity. These workings are shown on a beautiful plan from about 1903 (Figure 30). In plan view, King Lyell is the most erratic of the three deposits and although it is the smallest, it is also the highest grade containing 16,500 tonnes of copper or 34% of the current resource. The orebody as defined is up to 300m long, 180m wide and 45m deep, containing 1.5 mt of 1.1% Cu. There is a good chance that mineralisation will be found over an area of up to 400m long, 260m wide and 45m deep. This could lead to a doubling of the current resource to about 3 mt of 1.0% Cu.

The King Lyell resource is defined by only 7 holes of the total of 9 drilled. Of the other two, KL1 was not a good intersection, but it was not assayed over the whole hole and ML12 achieved an intersection at 62.5m depth of 49m at 0.52% Cu. This hole is near the Lyell Highway (Figure 19) and has been previously mentioned as the only hole to have a significant intersection in Gordon Limestone and not in clay. Further drilling in this area is recommended.

The 9 holes drilled are not well sighted to outline the entire deposit. Reid (1970) proposed a more extensive programme, but for some unknown reason, the 4 holes completed only achieved an average 44% recovery and Reid may have given up in disgust. Despite the poor recovery and poor assay reproduceability (Table 4), hole KL13 located 75m from Batchelor's Shaft achieved a good intersection of 27m at 1.10% Cu.

The other holes were disappointing, but the logger did not recognise some of the copper and the holes were not assayed over their entire length. Core condition has deteriorated since they were drilled and many core trays are missing, so it is not thought possible to correct this problem.

Strong zones of native copper rich mineralisation were intersected in the 1898/9 adit (55m at 4.6% Cu), in hole ML09 (40m at 2.2% Cu) and in Batchelor's Shaft (45m at 1.1% Cu; "excluding a zone between 45 feet and 50 feet which probably went 20% Cu"). At the base of Batchelor's Shaft, cross cuts were driven. The north cross cut passed through 0.2-0.3% copper before reaching the conglomerate contact and the south cross cut averaged 1.98% Cu over 26m (Reid, 1970).

King Lyell has a problem from two of the Iron Blow waste dumps (Figure 15) though these are only thought to be up to about 10m thick and may contain grades of interest. King Lyell is likely to be the second easiest mining proposition once the current aggregate operation is terminated. It contains a large area of prospective untested clay on its southern side between Cooney's Creek and the Gormanston Fault (Figure 15).

8. OPEN CUT POTENTIAL

The aim of this section is to take a preliminary look at economic factors to make sure that an exploration programme can be justified. This also encourages foresight during the exploration programme so that logging and assaying is suitable for future needs, and samples can be collected for metallurgical assessment. As the clays have been classified as an inferred resource, there is an "implication that there are reasonable prospects for eventual economic exploitation."

8.1 Waste to Ore Ratios

An exercise was undertaken where simple pit outlines were drawn on photocopies of each resource cross section. That is on Figures 3-6, 9-13 and 16-19. To avoid an extra 13 figures in this report, these are kept on file at Mt Lyell if required. In these simplistic pits, walls were drawn at 45° if in clay and at 50° if in conglomerate. Floors are not at the same RL, as all pits slope downhill parallel to the topography, but they are geometrically sensible. Deep or isolated ore was excluded to give figures for in pit resources. No adjustment has been made for end effects, so the volume of waste is probably slightly high. No dilution has been applied.

Results are summarised in Table 7 and detailed in Appendix 4. Waste to ore ratios on individual sections vary widely from 0.61 to 1 to 5.5 to 1, but overall ratios from each

orebody are more consistent in varying from 1.9 to 1 to 2.8 to 1, with an overall average of 2.3 to 1. The waste was higher than expected due to deposit location in valleys. In Appendix 5, the proportion of copper in resources recovered is listed. On individual sections, this varies from 66 to 100% and averages 83%.

Since more control is possible via detailed pit planning, the degree of waste removal is not thought to be a major hurdle to development. In fact, it presents an opportunity to develop a considerable production of construction aggregate materials for on mine use and off mine sale.

8.2 Metallurgy

Since the copper clays have been previously profitably treated, there is good reason to believe that as long as a substantial resource is outlined, it should be economically viable to treat. Grayson's (1903) description of the Blocks plant is given in section 5.1. There is sufficient information on mineral distribution from the existing 48 drill holes to believe that a gravity concentration plant would again be most appropriate. However, all options need to be considered during any feasibility study as follows:

1. Gravity concentration
2. Flotation
3. Solvent extraction and electrowinning

Previous gravity treatment at the Blocks and Consols gave recoveries of 72% with concentrate grades of 69% copper. This should be able to be improved with modern experience and technology. As the ore contains some gossan and some mineralised clasts, crushing and grinding for the estimated 20-25% that is not clay would be necessary. The Blocks also produced a secondary iron-rich product averaging about 20% copper.

Some metallurgical testwork was carried out by MLMRC from the 1964/65 Blocks and Consols drilling. As only the work sheets could be located, John Geoghegan, MLMRC's final Mill Superintendent, kindly provided a commentary on the work which is given in Appendix 6. He concluded that high slime contents would probably preclude flotation treatment and that fine chalcocite is likely to be the most troublesome component.

If a drilling technique such as air core/hammer be utilised, then for the mineralised sections, gravity recoverable copper as well as total chemical copper should be determined. Such work could be carried out in a diamond indicator mineral laboratory. It is thought that this type of analysis would be of great value in assessing the deposit. Five meter, ten meter or longer composites could be treated so the metallurgical response of the whole deposit can be determined.

Table 7

Summary of Copper Clays Resources

Deposit	Global Resource			In Pit Resource		
	Mt	% Cu	Tonnes Cu	Mt	% Cu	Tonnes Cu
Lyell Blocks	4.71	0.50	23,757	4.18	0.49	20,590
Lyell Consols	1.86	0.47	8,698	1.69	0.48	8,084
King Lyell	1.47	1.13	16,570	1.05	1.12	11,810
Totals	**8.04**	**0.61**	**49,025**	**6.92**	**0.59**	**40,484**

Deposit	Waste Mt	In Pit Resource Mt	Waste to Ore Ratio	Recovered % of	
				Tonnes	Cu
Lyell Blocks	7.92	4.18	1.89	89	87
Lyell Consols	4.77	1.69	2.82	91	93
King Lyell	3.00	1.05	2.86	71	71
Totals	**15.69**	**6.92**	**2.27**	**86**	**83**

Collection of metallurgical samples by near surface sampling is not recommended as it would be very hard to ensure representativeness. Regular size fraction analysis of uncrushed materials would also be necessary to determine the proportion of oversize that would need crushing. A small heavy-media plant could also be considered as this would effectively separate mineralised oversize and create a useful aggregate by-product.

8.3 Preliminary Economics

A brief and simplistic look at the economics of a copper clay operation is considered necessary to allow debate on the project's relative importance. Ball park cost figures have been obtained after discussions with Richard Winby and Hamish Bohannan. If contract earthmoving and an alluvial type concentration are utilised, capital cost is not expected to be a major hurdle. The type of operating cost scenario envisaged is as follows:

	per tonne ore $
• Mining (earthmoving) at $2.25/tonne and 2.3:1 waste to ore (3.3 tonnes per tonne ore)	7.43
• Cu recovery by gravity, including comminution	1.50
• Cu transport and recovery	2.10
• Administration	0.25
Total	**$11.28**

Thus, to break even, recovered grade at a revenue of $35 per % Cu would be 0.32% Cu. This equates to a head grade of 0.45% Cu at a recovery of 72%. For current resources, with an average grade of 0.61% Cu, an operating surplus of about $4 per tonne could be expected. However, the current grades at the Blocks and Consols would be marginal. To have operating costs as 50% of revenue with a 2.3 to 1 waste to ore ratio, a recovered grade of 0.64% Cu and head grade of 0.9% Cu would be required. This is thought to be possible by modification of pit designs.

For the King Lyell orebody (1.5 mt at 1.13% Cu) and a 2.9 to 1 waste to ore ratio, the figures are:

	per tonne ore $
• Mining per tonne of ore treated (3.9 x 2.25)	8.78
• Concentrate production	1.50
• Cu recovery and transport	3.95
• Administration	0.25
Total	**$14.48**
Revenue from 1.13% head grade is 0.81% recovered grade at $35 per % Cu	28.48
Surplus	**$14.00**

or for 1.5 mt, a total surplus of $21.0m

From these figures, it is concluded that a copper clays operation has the potential to be highly profitable. An exploration programme designed to increase confidence in resource knowledge is therefore justified.

9. EXPLORATION PROPOSALS

In order to enable a pre-feasibility study to be carried out, it is necessary to upgrade the resource classification to an indicated resource. Due to the unconsolidated nature of the Copper Clays, the resource estimation has similar problems to those evaluating alluvials and it may never be possible to estimate a measured resource. Any development may have to proceed on the basis of an indicated resource and probable reserve. This is possible with a two-stage drilling programme, the first stage of which is proposed here.

The main improvements to resource assessment factors which are necessary are:

1. Greater drillhole density.
2. Significantly improved recovery.
3. Minor improvements in assays, tonnage factor, size fraction analysis, quantitative mineralogy and documentation.
4. Commencement of systematic metallurgical investigations.
5. Commencement of mining investigations.

1. Greater Drillhole Density

A 64 hole, 5590m drilling programme is proposed as illustrated in Figures 20, 21 and 22 and tabulated in Appendix 7. A nominal spacing of 40m along lines 80m apart is proposed. This is slightly more dense than the 122 x 61m pattern proposed by Reid (1970) for an imperial grid. Holes average 87m in length which is similar to the historical average for the 46 holes to 1971 for 3876m at 84m per hole.

Hole depths have been predicted by reference to the cross section interpretations and are expected to be reasonably reliable. Only 8 holes (7 at the Blocks, 1 at King Lyell) are expected to be between 120 and 200m in length so their may have to be some compromise on these by leaving all difficult holes for a second phase.

It is important to complete holes to the Pioneer-Owen contact if possible as the Pioneer Sandstone is generally only about 10m thick and often carries good native copper grades.

Some holes will have to be drilled through Blow and North Lyell waste dumps, but this is unlikely to be more difficult than drilling through unconsolidated boulder beds. For best cross section interpretation, hole collars should be reasonably close to those planned.

This will involve considerable time and earthmoving cost for site preparation.

The grids at King Lyell and the Consols are the mine grid, whereas at the Blocks, a new local grid may be necessary. Holes on each traverse have been sighted so that at least one hole is outside the presumed orebody. If follow-up drilling is carried out it may be necessary to vary the hole spacing so there are more holes in the higher grade-width zones.

If drilling funds are scarce, holes could be drilled on the current cross sections. However, it would be necessary to redrill most holes for better recovery so the new pattern might as well be used. Another alternative is that one deposit, such as King Lyell, could be drilled first to check that objectives are being met.

2. Significantly Improved Recovery

Improved recovery is so important that a drill contractor and drillers with patience to experiment and a desire to improve will be necessary. It is likely that no one technique will be suitable for every situation and a combination of air core for clay and sand and a face sampling hammer for waste dumps, gossan, boulders and limestone may be necessary. A technique with casing advancing behind the bit may also be worth trialing.

It may be worth bringing in one or more experienced drilling consultants, showing them a selection of available clays core, and seeing what they recommend.

The air core technique is likely to create a big improvement in recovery in all unconsolidated zones with pebble size or less. A large diameter system may recover cobble size materials too and would create larger samples for metallurgical testing. It should be possible to retain 1 or 2m samples in large porous bags so that they can be drained and weighed. This will enable more reliable estimates of recovery and tonnage factor to be derived.

3. Minor Improvements

Assays: more duplicate work and screening of pulps to see if there is a copper smearing problem should be carried out.

Tonnage factor: weighing dry materials recovered from each meter should enable better data to be obtained. Clay samples are unlikely to dry and would have to be weighed damp and a water content estimated.

Size fraction analysis: this is necessary to make an estimate of the proportion of oversize that will be generated and hence crushed to liberate its copper.

Mineralogy: more quantitative estimates of the proportions of each copper mineral are necessary. A combination of whole rock and XRD analysis of concentrates on a suite of representative composites

should enable a reliable modal analysis. The previous practice of estimating pyrite contents from sulphur assays when the main mineral is native copper, not chalcopyrite, needs to be discontinued.

Drill logging: drill logging by a competent mineral observer with a binocular microscope on panned concentrates is thought to be desirable for semi-quantitative determinations.

Documentation:

it is taken for granted that CMT would not recreate the problems MLMRC have by lack of documentation of work for 25 years.

4. Metallurgical Investigations

As long as samples are kept in porous bags, loose water will drain out and the samples can be retained until assays are available. Composites for metallurgical or any other analysis can then be produced. In the next exploration stage, an attempt to determine total copper and gravity recoverable copper should be made. Some leaching and flotation tests can also be carried out so that a likely treatment route is known. This will influence later exploration.

5. Mining Investigations

There are not thought to be great problems here, but the proportion of the resource which can be freely dug or scraped compared to that requiring drilling and blasting needs to be known for an improvement in the quality of mining cost estimates. If a driller, an engineer and a geologist discuss the problem, it should be possible to devise a method of recording the ground type during drilling. For instance, air core drillable could indicate free digging and hammer drillable could indicate a need for drill and blast. Some work on suitable pit wall angles also needs to be carried out.

The Copper Clays evaluation is a challenging task on the boundary between alluvial, soft and hard rock evaluations. There will be many opportunities to improve evaluation techniques.

10. EXPLORATION POTENTIAL

In this section, the exploration potential of the whole Copper Clay's area is considered. This includes the head of the Linda Valley to AMG easterly co-ordinate 384000mE which is the eastern boundary of 1M/95 and ELA5/95. It can be sub-divided into four main possibilities:

1. Copper Clays and extensions.
2. Tabberabberan quartz vein gold.
3. Carbonate hosted lead-zinc.
4. Volcanic hosted copper-gold-silver.

10.1 Copper Clays and Extensions

The three deposits identified are not closed off by drilling and are likely to have extensions. Such opportunities have been termed Mpc or mineralisation-potential copper clays on Figure 1.

This includes the potential identified by the author of Figure 24 (probably Wade) in 1957. These areas are listed from north to south below with estimates of resource potential (rp) or inferred resource (ir) mineralisation at an average grade of between 0.6 and 1.2%, say 0.8% copper.

Area	Category	Resource Mt
North of Blocks (Wade, 1957b)	rp	6.0
Lyell Blocks (this work)	ir	4.7
Gap between Blocks and Balance Shaft	rp	1.0
North of Consols (Fig 24)	rp	0.5
West of Consols (Fig 24)	rp	0.5
Lyell Consols (this work)	ir	1.9
Lyell Extended/Pioneer (Fig 24)	rp	1.5
King Lyell (this work)	ir	1.5
South of and surrounding King Lyell	rp	1.5
Total	**rp**	**19.1**

If 83% of this potential is converted to a reserve and 72% of this is recovered, then potential copper production is about 91,000 tonnes.

10.2 Tabberabberan Quartz Vein Gold

The gold potential of the Linda Valley area has been addressed by Sillitoe (1984; 1985), Angus (1984) and Jones (1984; 1985). Evidence has been given in section 5.5 of gold-bearing quartz vein stockworks at McDowels, parallel to the North Lyell Fault. There are four areas currently known and possibly worthy of evaluation. These are:

1. McDowell's PA.
2. North flank of Mt Owen, Moore's and Gormanston Gold Mines.
3. Lyell Pioneer Gold workings.
4. Source of Jimmy Watson's old creek alluvials.

There is thus sufficient evidence to believe that Tabberabberan quartz vein stockworks, and associated processes, represent an important part of the metallogenic history of the area. However, to date, none have been large enough to warrant long-term exploitation. If larger deposits were present near the surface, we could expect to see some evidence in the distribution of alluvial workings.

Geoff Cordery (pers. comm., 1995) has suggested that these systems may develop into silica or sulphide-rich gold systems, such as the Henty deposit, at depth. This is a much more attractive target. As Tabberabberan D_2 veins show evidence of local derivation (Cox, 1979, p. 63), the most likely location for pyritic veins is in the Lyell Schists at depth and for siliceous veins adjacent to the Owen Conglomerate.

10.3 Carbonate Hosted Lead-Zinc

Markham (1966) recovered small isolated grains of honey-yellow sphalerite and galena from his copper clays drill composites. The minerals were not associated with the copper sulphides. Since the Owen Conglomerate hosted galena-barite veins reported by Bird (1984) do not contain anomalous zinc, the Markham Pb/Zn is thought to be from carbonate hosted mineralisation developed in the Gordon Limestone. A reward claim for zinc was granted over the Gormanston Cemetery area in 1902.

Banks and Baillie (1989) and Calver (1995) both report common secondary dolomitisation processes in Gordon Limestone Micrites. Therefore, given the regional linears indicating long-term deep fractures, an anomalous metal basement and the observed galena and sphalerite, it is likely that carbonate hosted Pb-Zn mineralisation of unknown dimensions and grade is developed.

Discovery of such deposits will be difficult. If a geophysical signature is present, it could be targeted. This is unlikely and mapping and sampling followed by reconnaissance drilling would be a favoured exploration strategy. As sulphides are stable in pug derived from Gordon Limestone, a search for galena and sphalerite in heavy mineral concentrates from shallow pug could be a very sensitive technique.

10.4 Volcanic Hosted Copper-Gold-Silver

The gold mineralisation mentioned above may be a leakage indication from a much greater prize; namely an extension of the Mt Lyell Field on the east side of the Great Lyell Fault (GLF). This idea will be more fully developed in a later report. The GLF is thought to be a Tabberabberan D_1 reverse fault generated at a depositional/lithological/rheological boundary due to overturned folding generated by the structurally weak hydrothermal alteration zone in the Central Volcanic Complex.

Orebodies such as the Blow, A Lens, Prince Lyell, Lyell Tharsis and North Lyell appear to be truncated by the GLF. Due to the reverse movement on the GLF, the extensions of these orebodies can be expected on the east side of the GLF under the Lyell Saddle - Western Linda Valley area. Unfortunately, depths are likely to be in the -1000 to -1500 RL range, so any orebodies here are probably propositions for the second half of the 21st Century. However, if their existence was confirmed, it would surely have a positive long-term effect on the Company's market profile.

The time to execute such a programme would be when Prince Lyell development is down to the 90 or 100 series. A 1km easterly cross-cut at say, -600 RL and a drilling development drive parallel to and approximately 1.5 km beneath the Copper Clays may be necessary.

11. KEYWORDS

Literature Review; Mine History; Open Cut Potential; Native Copper; Clay; Limestone; Ordovician; Devonian; Cainozoic; Glaciation; Ore Fluids; Mineralisation; Diamond Drilling; Mineralogy; Metallurgy; Mining; Economic Evaluation; Resources; Exploration Potential.

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APPENDIX 1

Copper Clays Summary Diamond Drill Logs

1.1 Lyell Blocks Holes

	Drilled		
LB 1- 4	From 16 October 1963	to	17 July 1964
LB 28-37	From 26 October 1965	to	10 June 1966
NL 1101-2	From 10 March 1984	to	18 July 1984

1.2 Lyell Consols Holes

LB 5-27	From 21 July 1964	to	24 September 1965

1.3 King Lyell Holes

ML 8-12	From 17 September 1902	to	28 November 1902
KL 1-2, 13 & 16	From 21 October 1970	to	25 November 1970

Total: 40 holes

1.1 Lyell Blocks Summary Drill Logs

Hole No	From (m)	To (m)	Abr	Summary Log
NL1101	0	39.1	QM	Clay, sand, pebbles and boulders – moraine.
	39.1	73.5	Ogc	Clay, sand and pebbles, ex-limestone.
	73.5	151.8	Ogl	Limestone and clay, some 'rubble' zones.
	151.8	211.2	Ogc	Clay, sandy clay, concretions, generally soft.
	211.2	226.3	Ogs	Sandstone, probably Pioneer Sandstone.
	226.3	554.4	Ous	Altered sandstone, brecciated, variably siliceous and/or hematitic, siderite veined, siltstone, conglomerate; quartz-carbonate-chlorite-barite siderite-galena veins over interval 246-451m. Pyrophyllite with veins and galena. Mylonitic zones over interval 482-546: Altered Owen Conglomerate. EOH.
Cu				No assays.
NL1102	0	2.0	-	Waste dump.
	2.0	75.2	-	Zone of high silica alteration of Owen Conglomerate.
	75.2	176.3	Ous	Quartz sandstone and conglomerate, fractured throughout with quartz-hematite-barite-sericite-pyrophyllite alteration: Altered Owen Conglomerate.
	176.3	195.5	Ogs	Green-pink quartz conglomerate, sandstone and siltstone, weak schistocity.
	195.5	217.5	Ogc	Grey and green sandstone and clay, ex Gordon Limestone.
	217.5	221.3	Ogs	Similar to 176.3 - 195.5m.
	221.3	269.4	Ous	Hematite-barite altered quartz sandstone.
	269.4	300.5	Ogs	Grey-green quartz sandstone.
	300.5	470.0	Ous	Purple hematitic sandstone and conglomerate, chert, jasper; altered Owen Conglomerate. EOH.
Cu				Cu assays 333-353m all <0.02%, anomalous BaSO4, 333-379m average 46m at 30% BaSO4. Anomalous Pb & Ag 333-335, 2m at 0.37% Pb, 9ppm Ag, 0.01% Cu, <0.05 g/t Au, 33% BaSO4.
LB1	0	63.1	Qm	Clay, sand, pebbles and boulders, unconsolidated moraine.
	63.1	71.0	Ogc	Sandy clay.
	71.0	95.1	Ogl	Limestone grey and brown with grit, limonite cubes, cavities.
	95.1	101.7	Ogs	Chromite-bearing quartzite (0.27% Cr): Pioneer Sandstone assays low 63-95m all Cu <0.1%, Au:Tr, Ag <2ppm.
Cu	85.0	95.0	Mcc	10m at 0.1% Cu.
LB2	0	64.2	Qm	Pebbles, sand, clay; probably moraine or scree.
	64.2	70.3	Ogc	Clay.
	70.3	101.2	Ogl	Grey limestone and clay.
	101.2	117.8	Ogc	Variegated clays.
	117.8	122.1	Ogs	Cr anomalous (0.27%) conglomerate: Pioneer edit.
Cu	106.4	117.8	Mcc	11.4m at 0.13% Cu.
LB3				Abandoned.
LB4	0	31.4	Qm	Boulders and clay: moraine
	31.4	110.2	Ogl	Limestone and fawn limestone, clay, some limonite and manganese staining, native Cu at 99m.
	110.2	147.7	Ogc	Variegated clays very weakly mineralised, small flakes of native Cu at 120m. Can see native Cu at 0.15% Cu.
	147.7	158.8	Ogs	Apple green quartzite (0.24% Cr): Pioneer Sandstone.
	158.8	161.2	Ous	Purple hematitic conglomerate-sandstone; Owen.
Cu	99.1	101.2	Mcc	2.1m at 0.26% Cu, spotty weak mineralisation 95-148m.
LB28	0	16.5	Qm	Boulders, clay, schist, wood – moraine.
	16.5	74.8	Ogc	Weakly mineralised variegated clay with sand, pyrite cubes, decomposed schist 58.3m Av 0.16% Cu.
	74.8	143.3	Ogc	Variegated mineralised clay, core/bedding 50°, pyritic, native Cu with red clay.
	143.3	193.5	Ogc	Dark grey clay, some sand, pyrite layers & veins parallel to bedding.

LB (2)

Hole No	From (m)	To (m)	Abr	Summary Log
LB28 cont	193.5	201.2	Ogs	Pebbly quartzite. Pioneer Sandstone EOH.
Cu	74.8	143.3	Mcc	1.33% Cu for 68.5m. 113-122m, 9m for metallurgy averages 2.4% Cu. Comment: A very good hole, most of Cu seems to be native Cu associated with red clay - NB steep layering. NB: 16.5 - 74.8m = 58.3m of 0.16% Cu and 143.3 - 181.4m = 38.1m of 0.12% Cu ie: 16.5 to 181.4 = 165m at 0.64% Cu.
LB29	0	9.1	Qm	Sand and boulders.
	9.1	73.0	Ogc	Variegated clays weakly mineralised to 73m then moderately mineralised.
	73.0	92.7	Ogc	Mineralised clays with traces of native Cu (BCA-90%)
	92.7	103.3	Ogs	Green tinged quartzite and sandstone: Pioneer EOH.
Cu	73.0	92.7	Mcc	0.25% Cu over 19.7m.
LB30	0	57.6	Ogc	Mineralised clays from surface with native Cu in first 2m. Bedding 90° to core axis. Office tunnel workings filled with white clay from 54.9 to 57.6m.
	57.6	71.6	Ogs	Pale green quartzite then pink conglomerate BCA 45° contrast to clays - could be a faulted contact. EOH.
Cu	0	48.8	Mcc	48.8m of 0.37% Cu - long mineralised interval.
LB31	0	4.7	Qm	Sand and boulders.
	4.7	27.7	Ogc	Variegated clays with sand, flat bedded.
	27.7	38.1	Ogs	Chromite-bearing pebbly quartzite; BCA 35°; Pioneer EOH.
Cu	5.0	25.0	Mcc	20m of 0.44% Cu.
LB32	0	12.5	Qm	Clay and boulders; boulder clay – moraine.
	12.5	22.3	Ogc	Weakly mineralised clay.
	22.3	32.9	Ogl	Sand, limestone; some with magnesite-lined vughs.
	32.9	64.0	Ogc	Clay, sand, pyritic clay.
	64.0	81.1	Ogs	Sand then quartzite and pebbly quartzite; Pioneer EOH.
Cu	14.0	22.0	Mcc	8.0m of 0.2% Cu - disappointing weak mineralisation.
LB33	0	12.2	Qm	Gravel, clay, pebbles, sand: moraine-outwash.
	12.2	49.1	Ogc	Variegated clays, flecks of native Cu but only up to 0.2% Cu - bedding 50-90° to core axis. Av 70°. 46.6 to 48.8 logged as a drive but not near workings!
	49.1	86.3	Ogl	Fawn limestone and clay.
	86.3	121.0	Ogc	Light grey and white clays only traces of mineralisation. 2m cavity at 118m.
	121.0	128.3	Ogs	Grey-green pebbly quartzite; Pioneer ODH.
Cu	38.1	45.7	Mcc	7.6m of 0.30% Cu.
LB34	0	24.4	Qm	Sand and gravel.
	24.4	48.8	Ogc	Clay, minor gravel, pyritic schist.
	48.8	56.4	Ogl	Clay with limestone fragments.
	56.4	86.6	Ogc	Grey and yellow clays some sandy limonite.
	86.6	105.5	Ogs	Chromite-bearing quartzite and conglomerate: Pioneer EOH.
Cu	24.4	39.6	Mcc	15.1m of 0.16% Cu.
LB35	0	12.5	Qm	Clay, sand and pebbles.
	12.5	54.9	Ogc	Yellow sandy clay with limonite.
	54.9	92.0	Ogl	Limestone and clay.
	92.0	162.5	Ogc	Brown to khaki clay with laminae of pyritic clay BCA 90° throughout.
	162.5	178.9	Ogs	Pebbly quartzite; Pioneer Sandstone EOH.
Cu	13.8	25.8	Mcc	12.0m of 0.20% Cu.
LB36				Abandoned.
LB37	0	44.2	Qm	Unconsolidated quartz sand with pebbles & boulders.
	44.2	140.2	Ogc	Hematitic sands.
	140.2	182.9	Ogc	Variegated clays red-orange-grey-black-yellow.
	182.9	193.9	Ogs	Pebbly chromititic quartzite EOH.
Cu	149.4	182.9	Mcc	33.5m of 0.2% Cu - also 23.2 - 85.3, 12.1m at 0.42% Cu.

1.2 Lyell Consols Summary Drill Logs

Hole No	From (m)	To (m)	Abr	Summary Log
LB5	0	2.0	Qm	Moraine.
	2.0	84.1	Ogc	Clay and sand.
	84.1	95.0	Ogs	Quartzite and conglomerate to EOH.
Cu	76.5	92.7	Mcc	16.2m at 0.32% Cu
LB6	0	17.7	Qm	Clay-moraine.
	17.7	33.0	Ogc	Clay - pug.
	33.0	75.3	Ogl	Limestone, sand and clay.
	75.3	147.5	Ogc	Black puggy clay and cavities.
	147.5	157	Ogs	Quartzite and conglomerate with chromite to EOH.
Cu				All assays less than 0.1% Cu.
LB7	0	78.6	Qm	Pebbles and sand, probably from moraine.
	78.6	82.4	Ogc	Black clay with minor fine pyrite.
	82.4	141.7	Ogl	Grey limestone with black shaley bands, limestone pebbles, calcite veins parallel to bedding.
	141.7	157.0	Ogs	Quartzite and conglomerate, chromite stain. EOH.
Cu				All assays less than 0.1% Cu.
LB8	0	8.5	Qm	Moraine with sand and pebbles.
	8.5	22.9	Ogc	Carbonaceous clays with some pyrite layers.
	22.9	32.0	Ogl	Dolomite, sand and clay.
	32.0	34.4	Mw	Timber and cavity - Consols workings?
	34.4	47.2	Ogc	Coloured clay with native Cu, sand.
	47.2	54.6	Ogs	Chromitic quartzite, some native Cu on joints, EOH.
Cu	30.8	48.5	Mcc	17.7m at 0.63% Cu.
LB9	0	3.0	Qm	Clay, sand and pebbles, probably moraine.
	3.0	27.4	Ogc/Mw	Clay and sand, bedded. (7-6-9.1 Consols Drive).
	27.4	40.2	Ogl	Mixture of clay, limestone, pyritic shale, limonite and goethite.
	40.2	53.0	Ogc	Clay, sand, limonite and goethite, quartzite, unconsolidated sand and conglomerate.
	53.0	64.9	Ogs	Native Cu on joint planes, green colouration after chromite. Up to 0.43% Cr.
Cu	18.3	57.2	Mcc	38.9m at 0.23% Cu.
LB10	0	4.3	Qm	Unconsolidated gravel and clay; moraine.
	4.3	14.9	Ogc	Clay, limonite, goethite and sand.
	14.9	46.2	Ogl	Dolomite, sand and clay, dolomitic remnants.
	46.2	58.5	Ogs	Quartzite and conglomerate with chromite. EOH.
Cu	3.2	50	Mcc	18m at 0.30%.
LB11	0	7.3	Qm	Boulders, pebbles, sand and clay; moraine.
	7.3	35.1	Ogc	Variegated clays, limonite, sand.
	35.1	42.4	Ogs	Chromititic conglomerate, quartzite and sand, EOH.
Cu	27.6	35.1	Mcc	7.5m at 0.22% Cu.
LB12	0	3.4	Qm	Clay with pebbles; moraine.
	3.4	19.2	Ogc	Clay, sand, limonite.
	19.2	44.2	Ogl	Mixture of clay, fragments of dolomite and sand.
	44.2	46.3	Mw	Timber and cavity - Consols No. 1 level drive.
	46.3	56.2	Ogc	Clay, sand, limonite with fine native copper.
	56.2	64.6	Ogs	Quartzite and conglomerate, EOH.
Cu	42.1	56.5	Mcc	14.4m at 0.46% Cu.
LB13	0	5.5	Qm	Brown sand and pebbles - moraine.
	5.5	20.1	Ogc	Clay with fragments of quartz, fine disseminated euhedral pyrite in black carbonaceous layers, sand.
	20.1	35.7	Ogl	Mixture of dark clays, sand and dolomite mostly less than 0.1% Cu.
	35.7	74.4	Ogc/Mw	Variegated clays with limonite-hematite and native Cu. Slivers of timber at 43.6m. Angular fragments of quartzite at 50m.
	74.4	93.0	Ogs	Brown unconsolidated and then pink quartzite with native Cu on joint and cleavage planes. EOH.
Cu	34.1	75.9	Mcc	42m at 0.71% Cu.

LC (2)

Hole No	From (m)	To (m)	Abr	Summary Log
LB14	0	7.0	Qm	Pebbles in clay – moraine.
	7.0	19.5	Ogl	Clay and sand with dolomite.
	19.5	41.1	Ogc	Clay with sandy bands and fine grained native Cu.
	41.1	52.1	Ogs	Pink quartzite with small-scale faulting. EOH.
Cu	15.2	45.7	Mcc	30.5m of 0.30% Cu.
LB15	0	10.4	Qm	White/grey sandy clay with pebbles – moraine.
	10.4	21.9	Ogc	Grey to black clay with disseminated euhedral pyrite.
	21.9	25.0	Ogl	Dolomite (muddy) with magnesite-lined cavities.
	25.0	36.0	Ogc	Clay and sand.
	36.0	38.7	Ogs	Pink chromitic quartzite. EOH (Pioneer Sandstone).
Cu	13.4	31.1	Mcc	18m of 0.2% Cu.
LB16	0	2.4	Qm	Clay, gravel, sand; probably moraine.
	2.4	57.0	Ogc	Variegated clay, hematitic and limonitic sand, gravelly clay towards base of unit.
	57.0	77.1	Ogs	Sand then conglomerate and quartzite with chromite-Pioneer Sandstone. EOH.
Cu	40.5	58.2	Mcc	17.7m at 0.12%, very weak.
LB17	0	6.1	Qm	Sand and pebbles – moraine.
	6.1	39.0	Ogc	Variegated clay with hematite and veins of native Cu. Native Cu finely disseminated from 8m to EOH.
	39.0	55.8	Ous	Pink quartzite and conglomerate, no chromite mentioned, probably Owen Conglomerate. EOH.
Cu	9.4	40.5	Mcc	31.1m of 0.26% Cu.
LB18	0	21.3	Ogc	Clay with cellular limonite and goethite.
	21.3	36.9	Ogc	Cellular limonite to gossan with pyrolusite and patchy native Cu (2.7m of 10.5% Cu from 27.1m).
	36.9	48.2	Ogc	Grey sandy clay.
	48.2	76.2	Ogs	Pink to purple sandstone and conglomerate.
	76.2	93.3	Ous	Native Cu on joints to 93m.
Cu	10.7	41.7	Mcc	31.0m at 1.56% Cu (29.9 to 36.0 sent for metallurgy).
LB19	0	5.0	Qm	Pale grey sandy clay with pebbles and pyrite pebbles – moraine.
	5.0	32.6	Ogc	Variegated clay, sand (some unconsolidated).
	32.6	47.2	Ogs	Quartzite with pebbles and chromite, EOH.
Cu	19.8	37.5	Mcc	17.7m at 0.1% Cu (adjacent hole 18 and very weak).
LB20	0	4.5	Qm	Sand, clay, pebbles – moraine.
	4.5	25.0	Ogc	Variegated clay, with quartzite boulders (ex Owen), unconsolidated sand, limonite gravel, cuprite at 22m and native Cu 22-25m.
	25.0	49.8	Ogs	Sand, quartzite and conglomerate with chromite layers.
Cu	18.1	23.8	Mcc	5.7m of 0.31% Cu.
LB21	0	3.7	Qm	Sand and gravel: moraine.
	3.7	28.0	Ogc	Variegated clays with limonite, goethite, hematite and pyrolusite. Unconsolidated cross-bedded sand. Native Cu 24.1 - 24.4m.
	28.0	40.1	Ogs	Conglomerate and quartzite. EOH.
Cu	18.3	28.0	Mcc	9.7m of 0.43% Cu.
LB22	0	8.8	Qm	Grey and black clay.
	8.8	18.6	Ogc	Variegated clay with limonite-goethite gravel.
	18.6	29.3	Ogl	Caverns, dolomite and grey clay.
	29.3	31.1	Ogc	Limonitic yellow clay with native copper.
	31.1	44.5	Ogs	Quartzite and conglomerate with chromite EOH.
Cu	23.2	36.0	Mcc	12.8m at 0.35% Cu.
LB23	0	9.8	Qm	Pale grey clay, sand and pebbles; moraine.
	9.8	54.7	Ogc	Sand, quartzite, boulders.
	54.7	90.2	Ogc	Brown and black clay with fine native copper visible after panning. (59.4-65.5 suitable for metallurgical testing).
	90.2	100.3	Ogs	Sand, quartzite then chromititic conglomerate, joint smears of native Cu at 94m EOH.
Cu	55.0	90.0	Mcc	35.0m at 0.62% Cu.

Hole No	From (m)	To (m)	Abr	Summary Log
LB24	0	4.6	Qm	Sand and pebbles: moraine.
	4.6	14.3	Ogc	Clay and sand.
	14.3	17.2	Mw	Timber and clay : collapsed stope.
	17.2	38.1	Ogc	Clay and sand.
	38.1	46.3	Ogs	Quartz, conglomerate, dolomite; chromite-bearing. EOH.
Cu	22.6	39.6	Mcc	17.0m of 0.2% Cu - very weak.
LB25	0	6.7	Qm	Sand and pebbles - moraine.
	6.7	43.7	Ogc	Variegated clay with sand, cellular limonite and goethite.
	43.7	50.3	Ogs	Conglomerate-pink, Pioneer sandstone type. EOH.
Cu				All <0.1% Cu.
LB26	0	6.1	Qm	Clays and cobbles; moraine.
	6.1	37.2	Ogc	Variegated clays and pyrite crystals up to 2mm. Fine native copper throughout visible after panning.
	37.2	37.8	Ogl	Dolomite (cavernous with magnesite-lined cavities) and limonitic clay.
	37.8	67.4	Ogc	Variegated clay, sand, mud.
	67.4	78.3	Ogs	Sand, quartzite and conglomerate. EOH.
Cu	33.5	70.7	Mc	37.2m 0.31% Cu.
LB27	0	4.0	Qm	Sand and gravel; moraine.
	4.0	77.1	Ogc	Variegated clays with decomposed hematite, sand, cobbles of black hematite.
	77.1	84.7	Ogs	Sand and conglomerate - Pioneer Sandstone EOH.
Cu	15.2	79.2	Mcc	64m at 0.26% Cu.

1.3 King Lyell Summary Drill Logs

Hole No	From (m)	To (m)	Abr	Summary Log
ML8	0	21.3	Lss	Weathered schist, pyritic, some quartz sand; could be glacial.
	21.3	76.2	Ogc	Variegated clay, ironstone pebbles, gossanous clay, blue pug with pyrite nodules.
	76.2	86.9	Ogs	Quartzite then conglomerate.
Cu	24.4	38.1	Mcc	13.7m of 0.3% Cu
ML9	0	42.7	Ogc	Variegated clays with quartz, ironstone and pyrite pebbles. Strong native Cu over whole hole.
	42.7	44.8	Ogs	Conglomerate impregnated with native Cu. NB: Three sludge assays are higher than core assays:
				From / To / Sludge Assay / Core Assay 65' / 70' / 8.7 / 1.9 (60' to 70') 70' / 75' / 15.0 /) 2.4 75' / 80' / 12.9 /)
Cu	3.0	43.0	Mcc	40m of 2.16% Cu (Best intersection in Cu clays).
ML10	0	21.3	Ogc	Black pug with pebbles of quartz, pyrite, conglomerate and schist, small cubes of pyrite, siderite pebbles, occasional grains of native Cu, sand.
	21.3	42.7	Ogc	Mainly sand with pebbles as above and native Cu.
	42.7	48.5	Ogc	Brown, blue and grey clays.
	48.5	51.2	Ogs	Conglomerate.
Cu	30.5	38.1	Mcc	7.6m of 0.44% Cu - Native Cu in sand.
ML11	0	13.7	Qm	Blue-black clay with quartz and pyrite pebbles, wood at 9m, grit, probably moraine – fluvoglacial.
	13.7	45.4	Ogl	Limestone, ironstained, not mineralised.
	45.4	83.8	Ogc	Yellow clay intermixed with 'black and blue' clay, limestone remnants. Native Cu from 56-85m, including octahedral crystals of Cu at 64m, also brown clay. Clay colour not related to native Cu content. Native Cu with yellow (64m), blue (69m), brown (82m) and black (83m) clays.
	83.8	87.8	Ogs	Conglomerate. EOH.
Cu	48.8	85.3	Mcc	36.5m of 0.67% Cu.
ML12	0	27.4	Qm	Black clay, gravel and sand, minor limestone, pyrite and native Cu, gradual increase in limestone content.
	27.4	114.9	Ogl	Gradational contact with limestone and quartz sand. Siliceous limestone, clay, quartz sand, pyrite, chalcopyrite and native Cu. Not clear from Batchelor's 1902 log if the Cu minerals are disseminated in silicified limestone or are detrital particles associated with the quartz sand. I think the py-cpy-Cu is disseminated in some silicified limestone. Cf comment at 84m - limestone showing specks of native Cu (Assay 0.5% Cu) from 86-93 6m of full core of solid limestone. No Cu is mentioned and assay is <0.1% Cu. From 94.5 to 102.1 7.6m averages 1.3% Cu and is logged as yellow sand with more clay and native Cu, pyrite and chalcopyrite.
	114.9	128.0	Ogc	Clay rich zone but still limestone remnants, also pyrite, native Cu and chalcopyrite.
	128.0	134.7	Ogs	Sand then conglomerate. EOH.
Cu	62.5	111.3	Mcc	48.8m of 0.52% Cu - mainly in limestone zone.

KL (2)

Hole No	From (m)	To (m)	Abr	Summary Log
KL1	0	2.4	Qm	Sand and clay.
	2.4	40.4	Ogc	Clay and sand, minor limestone pebbles. 0-19.8 not assayed but hole 9 only 10m away - had 2.4 to 21.3m, 18.9m averaged 3.05% Cu. Also from 32.0 to 38.6 is described as very little visible Cu mineralisation and no visible mineralisation apart from rare 1mm crystals of pyrite. This section assays 30.5 to 38.1m 7.6m at 1.7% Cu. Seems like poor logging and insufficient assays in KL1 (P G Burgdorf). Repeat assays do not compare very well either.
	40.4	43.1	Ogs	Conglomerate. EOH.
Cu	30.5	38.1	Mcc	7.6m of 1.69% Cu.
KL2	0	16.2	Qm	Sandy clay with schist and quartz fragments – moraine.
	16.2	30.3	Ogc	Black 'puggy' clay with 10% (by assay) fine disseminated pyrite, no copper.
	30.3	73.0	Ogl	Clay and sand with limestone remnants.
	73.0	98.5	Ogc	Dark grey and black clay with no visible copper except minor native Cu at 78.5m. EOH. Do not know if hole hit Pioneer Sandstone. 0-13, 34-73, 78.5-79.2, 86.1-92.7m, all not assayed and Cu would be expected. Last assay has 0.28% Cu. Adjacent hole ML11 intersected 36m of 0.67% Cu from 49-85m!
Cu	74.7	78.5	Mcc	3.8m average 0.86% (but 34-73m not assayed).
KL13	0	6.1	Qm	Sand and clay, probable moraine.
	6.1	18.9	Ogc	Grey; brown and black 'pug'.
	18.9	40.7	Ogl	Limestone with sand and disseminated pyrite interval not assayed but was mineralised in holes ML11 and ML12.
	40.7	70.7	Ogc	Brown pug, sandy in part, fine native Cu from 56.4 to 64.0 and pebbles of native Cu from 68.6 to 70.7m.
	70.7	73.2	Ogs	White to purple sandstone with numerous quartz veins. EOH.
Cu	42.7	70.1	Mcc	27.4m (open at both ends) averages 1.08% Cu.
KL16	0	44.7	Ogc	Red brown clay with gossanous siliceous ironstone fragments, quartz fragments. Visible native Cu from 25.8m to 29.7m but not reflected in Cu assays.
	44.7	58.2	Ogs	Pink and brown sandstone and conglomerate. EOH.
Cu	38.1	39.6	Mcc	1.5m of 0.13% Cu - does not match the other holes in the area or the description in the log, may have been a sample prep or assay mix-up.

APPENDIX 2

List of Drill Intersections In Each Hole

DRILL INTERSECTIONS

2.1 Lyell Blocks

Hole No	Year Drilled	Hole Depth (m)	Clay Length (m)	Best Intersections In Clays				Estimated Recovery %
				From (m)	To (m)	Length (m)	Cu %	
LB01	1963	101.7	32	85.0	95.0	10.0	0.10	40
LB02	1963/4	122.1	53	106.4	117.8	11.4	0.13	40
LB03	1964	16.5	-	-	-	-	-	Abandoned
LB04	1964	161.2	119	99.1	101.2	2.1	0.27	50
LB28	1965	201.2	128	16.5	181.4	164.9	0.64	50
Including		-		74.8	143.3	68.5	1.33	50
Including		-		77.0	85.5	8.5	2.20	67
Including		-		114.0	137.0	23.0	1.80	75
LB29	1965/6	103.3	58	73.0	92.5	19.5	0.25	50
LB30	1966	71.6	57	0.0	48.8	48.8	0.37	50
LB31	1966	38.1	23	5.0	25.0	20.0	0.44	50
LB32	1966	81.1	53	14.0	22.0	8.0	0.20	75
LB33	1966	128.3	97	38.1	45.7	7.6	0.30	100
LB34	1966	105.5	56	24.4	39.6	15.1	0.16	50
LB35	1966	178.9	160	13.8	25.8	12.0	0.20	77
LB36	1966	41.2	-					Abandoned
LB37	1966	193.9	52	149.4	182.9	33.5	0.20	50
Total m		1,544.6						

DRILL INTERSECTIONS

2.2 Lyell Consols

Hole No	Year Drilled	Hole Depth (m)	Clay Length (m)	Best Intersections In Clays				Estimated Recovery %
				From (m)	To (m)	Length (m)	Cu %	
LB05	1964	95.0	84	76.5	92.7	16.2	0.32	60
LB06	1964	157.0	148			all	<0.1	30
LB07	1964	157.0	63			all	<0.1	70
LB08	1964	54.6	39	30.8	48.5	17.7	0.63	60
Including				30.8	31.1	0.3	5.60	30
LB09	1964	64.9	56	18.3	57.2	38.9	0.23	65
LB10	1964	58.5	20	32.0	50.0	18.0	0.30	60
LB11	1964	42.4	28	27.6	35.1	7.5	0.22	22
LB12	1964	64.6	45	42.1	56.5	14.4	0.46	75
LB13	1965	93.0	70	34.1	75.9	41.8	0.71	50
LB14	1965	52.1	34	15.2	45.7	30.5	0.30	50
LB15	1965	38.7	33	13.4	31.1	17.7	0.25	55
LB16	1965	77.1	55	40.5	58.2	17.7	0.12	60
LB17	1965	55.8	31	9.4	40.5	31.1	0.26	80
LB18	1965	93.3	48	10.7	41.7	31.0	1.56	60
Including		-		27.1	29.9	2.8	10.50	84
LB19	1965	47.2	33	19.8	37.5	17.7	0.10	75
LB20	1965	49.8	24	18.1	23.8	5.7	0.31	65
LB21	1965	40.1	30	18.3	28.0	9.7	0.43	70
LB22	1965	44.5	39.	23.2	36.0	12.8	0.35	50
LB23	1965	100.3	90	55.0	90.0	45.0	0.62	80
Including		-		59.4	70.1	10.7	1.38	91
LB24	1965	46.3	34	22.6	39.6	17.0	0.20	20
LB25	1965	50.3	34			all	<0.1	50
LB26	1965	78.3	67	33.5	70.7	37.2	0.30	70
LB27	1965	84.7	73	15.2	79.2	64.0	0.26	80
Including		-		53.3	59.4	6.1	0.78	100
Total m		1,645.5						

DRILL INTERSECTIONS

2.3 King Lyell

Hole No	Year Drilled	Hole Depth (m)	Clay Length (m)	Best Intersections In Clays				Estimated Recovery %
				From (m)	To (m)	Length (m)	Cu %	
ML08	1902	86.9	53	24.4	38.1	13.7	0.3	poor
ML09	1902	44.8	43	2.4	43.0	40.6	2.16	low
Including				2.4	6.1	3.7	9.5	low
ML10	1902	51.2	47	30.5	38.1	7.6	0.44	low
ML11	1902	87.8	85	48.8	85.3	36.5	0.67	poor
ML12	1902	134.7	127	62.5	111.3	48.8	0.52	poor
Including				94.5	106.7	12.2	1.05	poor
*B.Shaft	1902/3	-	45	0.0	45.1	45.1	1.06	good
KL01	1970	43.1	40	30.5	38.1	7.6	1.69	37
KL02	1970/1	98.5	98	74.7	78.5	3.8	0.86	44
KL13	1971	73.2	70	42.7	70.1	27.4	1.08	46
KL16	1970	58.2	43	38.1	39.6	1.5	0.13	50
Total m		**685.6**						

* Batchelor's Prospecting Shaft November 1902-1903.
Excluding a gossan zone between 13.7 and 16.8m estimated at 20% Cu.
If included, the whole 45.1m averages 2.41% Cu.

Cumulative metres

Lyell Blocks	1,544.6
Lyell Consols	1,645.5
King Lyell	685.6
	3,876m
NL 1101 & 1102	025
Total:	**490/m**

APPENDIX 3

Global Resource Estimations

Appendix 3

LIST OF GLOBAL RESOURCES

3.1 LYELL BLOCKS

Section No	Area m²	Projection m	Tonnes (SG=2.1)	Grade % Cu	Contained Cu T
IN	8750	100	1,837,500	0.65	11,944
IS	8750	39	716,625	0.65	4,658
2N	4520	39	370,188	0.26	962
2S	4520	42.5	403,410	0.26	1,049
3N	4850	42.5	432,863	0.37	1,602
3S	4850	65	662,025	0.37	2,450
4N	2280	40	191,520	0.38	728
4S	2280	20	95,760	0.38	364
Totals		**388**	**4,709,891**	**0.50**	**23,757**

3.2 LYELL CONSOLS

Section No	Area m²	Projection m	Tonnes (SG=2.1)	Grade % Cu	Contained Cu T
1	1810	64	243,264	0.26	633
2	2340	100	491,400	0.43	2,113
3N	1525	64	204,960	1.37	3,808
3S	1410	80	236,880	0.34	805
4	2070	49	213,000	0.52	1,108
5	3005	75	473,288	0.26	1,231
Totals		**432**	**1,862,792**	**0.47**	**8,698**

3.3 KING LYELL

Section No	Area m²	Projection m	Tonnes (SG=2.1)	Grade % Cu	Contained Cu T
1	4815	80	808,920	1.37	11,082
2	2415	74	375,291	0.94	3,528
3	1630	83	284,109	0.69	1,960
Totals		**237**	**1,468,320**	**1.13**	**16,570**

Global Total

Deposit	Tonnes SG=2.1	Grade % Cu	Contained Cu T
Lyell Blocks	4,709,891	0.50	23,757
Lyell Consols	1,862,792	0.47	8,698
King Lyell	1,468,320	1.13	16,570
Totals	**8,041,003**	**0.61**	**49,025**

APPENDIX 4

In Pit Resources and Waste to Ore Ratios

Appendix 4

In Pit Resources and Waste to Ore Ratios

Section	Areas m²		Projection Length (m)	M. Tonnes		Waste Ore Ratio	Grade % Cu	Tonnes Cu
	Waste	Ore		Waste	Ore			
LYELL BLOCKS				***2.2**	**2.1**			
1	15,755	7,235	139	4.82	2.11	2.28	0.65	13,727
2	12,890	4,530	81.5	2.32	0.76	3.05	0.26	2,016
3	2,545	4,600	107.5	0.60	1.04	0.58	0.37	3,822
4	1,385	2,140	60	0.18	0.27	0.67	0.38	1,025
Totals				**7.92**	**4.18**	**1.89**	**0.49**	**20,590**
LYELL CONSOLS				**2.4 / 2.2**	**2.1**			
1	3,550	1,205	64	0.55	0.16	3.44	0.26	421
2	3,440	2,265	100	0.83	0.48	1.73	0.43	2,045
3N	1,535	1,475	64	0.34	0.20	1.70	1.37	2,716
3S	3,160	1,165	80	0.61	0.20	3.05	0.34	665
4	8,090	2,045	49	0.83	0.21	3.95	0.52	1,094
5	10,240	2,790	75	1.61	0.44	3.66	0.26	1,143
Totals				**4.77**	**1.69**	**2.82**	**0.48**	**8,084**
KING LYELL				**2.2**	**2.1**			
1	3,905	3,445	80	0.66	0.58	1.14	1.37	7,929
2	7,030	1,620	74	1.14	0.25	4.34	0.94	2,366
3	6,890	1,260	83	1.20	0.22	5.47	0.69	1,515
Totals				**3.00**	**1.05**	**2.86**	**1.12**	**11,810**
GRAND TOTALS				**15.69**	**6.92**	**2.27**	**0.59**	**40,484**

* These numbers are the Tonnage Factors utilised.

Appendix 5

Proportions of Resources Recoverable

Section	In Pit Resource	Grade	Global Res.	Grade	Recovered %	
	MT	% Cu	MT	% Cu	Tonnes	Cu
B1	2.11	0.65	2.56	0.65	83	83
B2	0.76	0.26	0.77	0.26	99	99
B3	1.04	0.37	1.09	0.37	95	95
B4	0.27	0.38	0.29	0.38	93	93
Totals	**4.18**	**0.49**	**4.71**	**0.50**	**89**	**87**
C1	0.16	0.26	0.24	0.26	67	67
C2	0.48	0.43	0.49	0.43	98	98
C3N	0.20	1.37	0.20	1.37	100	100
C3S	0.20	0.34	0.24	0.34	83	83
C4	0.21	0.52	0.21	0.52	100	100
C5	0.44	0.26	0.48	0.26	94	94
Totals	**1.69**	**0.48**	**1.86**	**0.47**	**91**	**93**
K1	0.58	1.37	0.81	1.37	72	72
K2	0.25	0.94	0.38	0.94	66	66
K3	0.22	0.69	0.28	0.69	79	79
Totals	**1.05**	**1.12**	**1.47**	**1.13**	**71**	**71**
GRAND TOTAL	**6.92**	**0.58**	**8.04**	**0.61**	**86**	**83**

APPENDIX 6

1966 Metallurgical Testwork

MEMORANDUM

To: Kevin Wills
From: John Geoghegan
Subject: Copper Clays testwork
Date: 24th. May, 1995

The testwork carried out in 1966 was apparently aimed at evaluating the possibility of including the clays in the mill feed of the day, hence the limited scope and lack of perserverance when the results were not encouraging.

Five samples were subjected to laboratory flotation and, as there is no information regarding pre-treatment or sieve analyses,I suspect that the samples were simply repulped.The fraction referred to as "slime" probably consists of decanted colloidal material which would have been excluded from flotation because of the problems such material usually causes. The slime flotation test on sample #2 shows typical lack of response, 50% recovery with a concentrate grade similar to the feed grade. On the other hand, flotation of the "sand" fraction generally gave good recoveries (up to 90%) and reasonable concentration ratios.

As you point out poor core recovery may have resulted in non-representative samples, but in all samples tested the slime fraction represents a large percentage of the total, and only in sample #1 is the slime assay significantly lower than the head. This condition would probably preclude flotation as a treatment option but some further testwork may be warranted once the mineralogy is known.

Conventional treatment of native copper bearing material would comprise comminution, if necessary, followed by gravity treatment to recover the coarse copper and flotation to recover the fine material.
The developmemt of high "g" force gravity separators may provide an alternative to flotation for the fine fraction treatment of materials such as this.The Kelsey Centrifugal Jig and the Moseley Multigravity Separator are examples of these.

If a drilling program is undertaken, the mineralogy and physical variations can be assessed and an appropriate metallurgical evaluation program can be designed. Native copper and un-oxidised sulphides will be recovered in the existing circuit, fine secondary chalcocite is likely to be the most troublesome component.

THE MOUNT LYELL MINING AND RAILWAY CO. LTD.

21/1/66

Form 423

METALLURGICAL RESEARCH DEPARTMENT

TEST NO ...1.... NAME Native Copper Clays

PRODUCT	Weight	% Weight	ASSAY Cu	ASSAY S	Cu Units Cu	Cu Units	Cu Units S	DISTRIBUTION Cu		DISTRIBUTION S
Conc	181	6.03	26.81	6.92	162		42	71.0	65	71.1
Tail	615	20.5	0.73	0.15	15		3	6.6	6	5.1
Sand Float	796	26.53	6.67	1.7	177		45	77.6	71	76.2
Slimes	2204	73.47	0.70	(1.0) 0.19	51	73.4	14	22.4	29	23.8
Head	3000	100.0	2.28	0.59	228	250	59	100.0		100.0

LBS PER TON

OPERATION	GRINDING Mesh	Product %	%
TIME	+ 10		
	+ 14		
	+ 20		
	+ 28		
	+ 35		
	+ 48		
	+ 65		
	+ 100		
	+ 150		
MACHINE	+ 200		
R. P. M.	+ 325		
pH	- 325		
% SOLIDS			
TEMPERATURE			

REMARKS : T.[illegible]

6:30

:5

91. [illegible]

THE MOUNT LYELL MINING AND RAILWAY CO. LTD.

23/4/66 Form 423

METALLURGICAL RESEARCH DEPARTMENT

Cu Blocks Floor II

~~TEST NO~~ . . . . . NAME . . . . . .

PRODUCT	Weight	% Weight	ASSAYS Cu	ASSAYS Fe	ASSAYS	Cu Units	Cu DISTRIBUTION
Sand Con	847	3·9	0·86	13·9	2·45	3·35	30·2
Mid	42·3	2·0	·073	13·4	0·12	·15	1·4
Tail	957·0	44·4	·064	9·4	·07	2·84	25·8
Slime	500	23·2	·102	10·0	·28	2·36	21·8
Sludge Con	213·0	9·9	·128	9·3	·26	1·26	11·3
Mid	113·2	5·2	·088	9·7	·15	·46	4·2
Tail	245·2	11·4	·058	8·4	·22	·66	6·0
Total Weight =	2156·4	100·0	0·11			11·02	100·0
Sand Head		50·3	0·126			6·34	57·3
Sludge Head		26·5	0·090			2·38	21·5
Total Head		100·0	0·11			11·02	

LBS PER TON

OPERATION	GRINDING	
TIME	Product	
	Mesh	%
	+ 10	
	+ 14	
	+ 20	
	+ 28	
	+ 35	
	+ 48	
	+ 65	
	+ 100	
	+ 150	
MACHINE	+ 200	
R. P. M.	+ 325	
pH	− 325	
% SOLIDS		
TEMPERATURE		

REMARKS :

Slime at grade similar to head and saved grade amounts to 50% by weight, and little beneficiation is possible by flotation.

Sulphidized mid gives little extra recovery.

THE MOUNT LYELL MINING AND RAILWAY CO. LTD.

29/4/66. Form 423

METALLURGICAL RESEARCH DEPARTMENT

Cu Clays

TEST NO Head No III. NAME Research.....

PRODUCT	Weight	% Weight	~~Cu~~ ASSAY Cu	S	Cu Units	Cu DISTRIBUTION
Slimes	1409	47·3	~~·055~~ ·053	·07	2·51	43·5
Con	70~~2~~	2·4	·55	2·27	1·32	23·0
Mid	135·	4·5	·036	·21	·16	2·8
Tail	13663	45·8	·039	—	1·78	30·7
Total Weight	2980·5	100·0	·058		5·77	100·0
Sand Head.		54·2	·060		~~3·29~~ 3·26	56·5

LBS PER TON

OPERATION	GRINDING		
TIME		Product	
	Mesh	%	%
	~~+~~ 10		
	+ 14		
	+ 20		
	+ 28		
	+ 35		
	+ 48		
	+ 65		
	+ 100		
	+ 150		
MACHINE	+ 200		
R. P. M.	+ 325		
pH	— 325		
% SOLIDS			
TEMPERATURE			

REMARKS : Decant.

THE MOUNT LYELL MINING AND RAILWAY CO. LTD.

Form 423

METALLURGICAL RESEARCH DEPARTMENT

TEST NO Cu Blocks IV NAME . Consols No. 3.

PRODUCT	Weight	% Weight	ASSAYS Cu Units				DISTRIBUTION Cu	
Con	57	2·1	17·3	36·3	11·13	23·4	41·6	40·4
Tail	601	20·6	0·204	4·2	0·14	2·9	49	49
Slime	2246	77·3	0·605	40·7	041	31·7	53·5	54·7
Total	2904	100·0		87·2		58·9	100·0	100·0
Tail.								
+ 100		0·8						
+ 150		8·2	0·196	1·96			10·0	
+ 200		10·3	0·122	1·26			6·6	
+ 325		20·4	0·182	3·72			19·0	
- 325		60·3	0·209	12·60			64·4	
		100·0		19·54			100·0	

LBS PER TON

OPERATION

TIME

MACHINE

R. P. M.

pH

% SOLIDS

TEMPERATURE

GRINDING

Mesh	Product %	%
+ 10		
+ 14		
+ 20		
+ 28		
+ 35		
+ 48		
+ 65		
+ 100		
+ 150		
+ 200		
+ 325		
- 325		

REMARKS : Calculated Head 0·87

Overall Recovery 41·6%

Calculated Tail 0·195

Assayed " 0·204

89% rec. on sands.

THE MOUNT LYELL MINING AND RAILWAY CO. LTD.

Form 423

METALLURGICAL RESEARCH DEPARTMENT

TEST NO Cu Blocks V NAME .....

PRODUCT	Weight	% Weight	ASSAYS Cu	Cu Units	ASSAYS S	S Units	DISTRIBUTION Cu	DISTRIBUTION S
Con	57	1.9	10.48	19.9	1.93	3.7	77.4	33.3
Tail	1919	62.9	0.030	1.9	0.04	2.5	7.4	22.6
Slime	1074	35.2	0.112	3.9	0.14	4.9	15.2	44.1
				25.7		11.1	100.0	100.0
Total	3050	100.0						

Tail			Cu Units		Cu
+ 100	1.6		0.082	0.13	3.5
+ 150	20.0	0.194	0.027	0.54	23.5
+ 200	22.4	0.082	0.031	0.69	16.2
+ 325	30.0	0.182	0.046	1.38	36.6
- 325	26.0	0.209	0.038	0.99	30.4
	100.0	17.86		3.73	100.0

LBS PER TON

OPERATION	Conditt	Rough.
TIME	18	5
Amyl Xanthate	0.14	
208	0.07	
404	0.3	
Lime.	1.5	
MACHINE	2000 gm.	
R. P. M.	1000	
pH		11.0
% SOLIDS	26	
TEMPERATURE		

GRINDING

Mesh	Product %	%
+ 10		
+ 14		
+ 20		
+ 28		
+ 35		
+ 48		
+ 65		
+ 100		
+ 150		
+ 200		
+ 325		
- 325		

REMARKS :

Calculated Head 0.25

Overall Recovery 77.4%

Calculated Tail 0.037

Assayed Tail 0.030

91% rec on sands.

APPENDIX 7

List of Proposed Drillholes

Appendix 7

7.1 Blocks Proposed Holes

Abbreviated Name	Line No	Position On Line	Predicted Depth (m)	Line Total
B1	080N	080W	50	
B2	080N	040W	200	
B3	080N	000	180	
				430
B4	000	080W	50	
B5	000	040W	200	
B6	000	000	200	
B7	000	040E	180	
				630
B8	080S	080W	30	
B9	080S	040W	120	
B10	080S	000	150	
B11	080S	040E	130	
				430
B12	160S	040W	30	
B13	160S	000	80	
B14	160S	040E	90	
				200
B15	240S	000	40	
B16	240S	040E	60	
B17	240S	080E	80	
				180
B18	320S	080W	30	
B19	320S	040W	70	
B20	320S	000	90	
B21	320S	040E	100	
B22	320S	080E	120	
				410
GRAND TOTAL				**2,280**

7.2 Consols Proposed Holes

Abbreviated Name	Line No	Position On Line	Predicted Depth (m)	Line Total
C1	6220E	5280N	10	
C2	6220E	5320N	80	
C3	6220E	5360N	80	170
C4	6300E	5280N	10	
C5	6300E	5320N	60	
C6	6300E	5360N	60	130
C7	6380E	5280N	10	
C8	6380E	5320N	80	
C9	6380E	5360N	70	
C10	6380E	5400N	30	
C11	6380E	5440N	70	
C12	6380E	5480N	40	300
C13	6460E	5320N	60	
C14	6460E	5360N	60	
C15	6460E	5400N	70	
C16	6460E	5440N	100	290
C17	6540E	5320N	80	
C18	6540E	5360N	90	
C19	6540E	5400N	110	
C20	6540E	5440N	100	380
GRAND TOTAL				**1,270**

7.3 King Lyell Proposed Holes

Abbreviated Name	Line No	Position On Line	Predicted Depth (m)	Line Total
K1	6260E	4660N	80	
				80
K2	6340E	4500N	150	
K3	6340E	4540N	120	
K4	6340E	4580N	80	
K5	6340E	4620N	70	
K6	6340E	4660N	60	
K7	6340E	4700N	40	
K8	6340E	4740N	70	
				590
K9	6420E	4540N	100	
K10	6420E	4580N	100	
K11	6420E	4620N	80	
K12	6420E	4660N	60	
K13	6420E	4700N	70	
K14	6420E	4740N	80	
				490
K15	6500E	4580N	120	
K16	6500E	4620N	120	
K17	6500E	4660N	110	
K18	6500E	4700N	90	
K19	6500E	4740N	60	
				500
K20	6580E	4620N	120	
K21	6580E	4660N	140	
K22	6580E	4700N	120	
				380
GRAND TOTAL				**2,040**

Summary:

Deposit	No. of Holes	No. of Metres
Blocks	22	2,280
Consols	20	1,270
King Lyell	22	2,040
Total	64	5,590

ROGER TOWNEND AND ASSOCIATES
CONSULTING MINERALOGISTS
ACN 069 920 476

Principal: Dr. Roger Townend

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KEVIN WILLS,

COPPER MINES OF TASMANIA,

BOX 84 ,

QUEENSTOWN,

TASMANIA

21-7-95

OUR REF 95037

YOUR REF c 0223

XRD EXAMINATION OF FIVE DRILL CORE "CLAY" SAMPLES AND ONE POWDER.

R TOWNEND

INTRODUCTION

Five drill cores and one powder were submitted for XRD etc with reference to their copper bearing clay component.

Part of the core was lightly ground , and as initial binocular examination suggested significant non clay material, a representative portion was wet sieved through a 38 micro screen.It was dried at 65oC.The black powder FS G 5034 was analysed as received.

RESULTS

1. SCREENING.

	-38 MICRONS WT%
FS G 5029	77.3
FS G 5030	41.5
FS G 5031	71.2
FS G 5032	62.7
FS G 5033	58.1

2. XRD ANALYSES.

The five -38 micron fractions and one powder all gave similar results. Only sample FS G 5033 had significant amounts of other phases.

The major minerals in all samples were QUARTZ and MICA (? MUSCOVITE). KAOLIN was also present , major in 5029 and 5033, trace in 5034 and accessory to minor in the others. SIDERITE was significant in 5033, and there was a major peak at 9.12 in the same sample,which has not been assigned.

The 14-15 angstrom smectite peaks were absent or of trace presence only. These results appear to confirm the work of Solomon(1969, Proc. Aus. IMM.)

Discount rate 15%
Cu price $1.00/lb
use 7%

Positive NPV 1.15 over $1.50/lb Cu.

Discussion

The King Lyell copper clay resource is in the Linda valley between the Iron Blow waste dumps and the Lyell Highway. The site is currently occupied by the Batchelor gravel stockpiles, crushing and screening equipment.

This is a quick look at the King Lyell copper clays following on from the resource definition work by K. Morrison and J. Knight in 1997. The main aim of this work was to put some mining, milling and economic parameters around the resource and recommend the direction for further work on the copper clays.

A plan and representative cross sections are included in this memo. Otherwise it is assume that readers have access to the report by Morrison and Knight. The lower grade whole body was evaluated and no work attempted in the west only higher grade section. The size of the deposit made it unlikely that the higher grade section would be financially attractive.

Resource and Reserve

The mineralisation in the copper clays is a combination of native copper, cuprite (Cu_2O_4), and digenite (Cu_9S_5). The modelling on the resource that had been done by Morrison and Knight was based on simple geometry that assumed a 60 degree overall pit slope. This was reduced to 45 degrees for the following reasons -

- there will probably be a large amount of wet clay in the pit walls
- the deepest section of the conceptual King Lyell pit is 100 metres deep
- provision to be made for haul roads

A comparison of 45 and 60 degree pit slopes was made on representative sections which are attached. This was used to estimate a factor for the additional waste at 45 degrees.

The ore resource and waste estimate (at 45 degree pit slopes) is as follows

Resource	1.2 Mt @ 1.4 % Cu
Waste	4.2 Mt (3.5:1 strip ratio)

For the purposes of this exercise it has been assumed that there is no dilution, and that the reserve is equal to the resource. The details are shown in the attached table of resource, reserve and waste estimates.

After first pass modelling of a 250,000 tpa case using the King Lyell resource, in consultation with Nick Clarke, a conceptual resource of 7.5 million tonnes was assumed at the same head grade as above, with a 750,000 tpa production rate giving a 10 project life.

Mining

Mining for the King Lyell would be by contractor in an open cut up to 100 metres deep. The ore and waste are assumed to be free digging, with an excavator being used. The ore would be hauled up the pit and across surface for 500 metres to a stockpile adjacent

to the mill. The waste would be hauled to waste dumps which would be managed to allow for rehabilitation. The existing waste dumps may also have to be moved in places.

The mining rate would be 250,000 tpa of ore and 960,000 tpa of waste. A detailed pit design was not within the scope of this study. Mining of the conceptual 750,000 tpa operation would be similar, probably from a number of pits simultaneously.

Mining costs were derived from those for open pit oxide ore and waste mining at Reedy. Details of the derivation of costs are attached. These were increased by 50 % to take account of smaller scale, variable digging conditions, and generally poorer pit conditions because of the water and clay.

The mining physicals and costs are shown in the attached spreadsheets for the 250,000 tpa and conceptual 750,000 tpa cases.

Milling

A conceptual gravity circuit was laid out in consultation with Nick Clarke. A semi-autogenous (SAG) mill was included to allow some size reduction in the feed to the gravity circuit. This would be followed by jigs and spirals to produce a 45 % Cu concentrate at a recovery of 65 %. This concentrate would be dried and trucked to the concentrate shed in Queenstown: The gravity concentrate would then be mixed with the chalcopyrite concentrate for transportation.

Physicals and costs are shown on the attached spreadsheets. For the 250,000 tpa option 2,300 tonnes of copper would be produced in 5,100 tonnes of concentrate per year. These would increase to 6,800 tonnes and 15,200 tonnes respectively for the conceptual 750,000 tpa option.

Tailings would be pumped to a new tailings dam close by with all tailings from a 250,000 tpa option being contained in the tailings dam. For the conceptual 750,000 tpa option, an initial tailings dam would be followed by disposal in separate copper clays open pits.

Mill operating costs were generated from rough estimates of manning, consumable and power usage. The mill capital costs were scaled down from a similar operation in the AusIMM Cost Estimation Handbook.

Smelting and Refining

The concentrate would be transported to smelters mixed with the chalcopyrite concentrate and treated and refined on similar terms. Because of a higher concentrate grade the cost per unit of copper would be less than that for the current chalcopyrite concentrates. This realisation plan would need to be examined in more detail to ensure that smelters would accept this type of concentrates.

Environmental

The area of the King Lyell and other prospective copper clay deposits is made up of steep hillsides, narrow valleys, old workings (of the copper clays) extensive waste dumps and close contact with the public on the Lyell Highway. Currently CMT has no

responsibility for rehabilitation of this area. Any new workings could open up extensive commitments, particularly in relation to old waste rock dumps. There was no provision made in this estimate for cost of rehabilitation.

Any proposals need to carefully consider the operational and environmental strategies. A small resource could not be extracted without consideration of adjacent resources. The economics of a small operation do not look feasible, and by the time waste removal and tailings disposal had been taken care of other resources may have been sterilised.

Project Economics

June '06 Cu at $US 1.60/lb gives + 10M NPV for both 250,000 & 750,000 tpa scenarios.

The net present values (NPV) before tax for the 250,000 and conceptual 750,000 tpa projects are shown on the relevant spreadsheets. A discount rate of 15 % was used which may be too high considering the existing inflation rate. However it is probably still appropriate considering the risk of such projects.

The 250,000 tpa option came in with an NPV of -$3.6 million, mainly due to short project life (5 years) and relatively high mill capital and operating costs. There was also a large component for a tailings dam. The waste mining and disposal was a major cost accounting for 55 % of annual operating site expenditure. Even the reduction of the strip ratio from 3.5 to 2.3 (originally estimated) still resulted in a negative NPV.

Sensitivities on site operating costs, head grade, stripping ratio and copper price were calculated on 15 % increments up to 45 %. A break even NPV was achieved at 20 to 40 % favourable movements in each of these parameters. This option does not look feasible, particularly considering the risks involved in a small scale one deposit project.

The conceptual 750,000 tpa option has an NPV of $10.2 million, helped by economies of scale in the mill capital and operating costs. No economies of scale were assumed for mining, as this production rate would be achieved with a number of smaller sized deposits. The waste removal accounted for 67 % of annual site operating costs.

Despite the positive NPV, the 750,000 tpa option was not particularly durable, with sensitivity analysis showing a zero NPV with 10 to 30 % unfavourable movements in site operating costs, head grade, stripping ratio and copper price.

Recommendations or higher Cu price! - Recall this was written in '97 when Copper was less $1.00/lb!!

A larger resource appears to be necessary to make the King Lyell and the copper clays in general more economically attractive. A review of potential copper clay resources in the area should be initiated to identify a resource of at least 5 to 10 million tonnes. In addition more detailed designs should be done on the resources to more accurately quantify the waste tonnage.

KING LYELL COPPER CLAYS
RESOURCE, RESERVE AND WASTE ESTIMATES

By: Tony Weston
Date: 31/07/97
File: g:\mining\engineer\tony\excel\klcc.xls

Up to 20% in Cu clay samples.

Estimated King Lyell Resource

Resource

	(m^3)	(SG)	(t)	(% Cu)
West	240,000	2.6	612,000	2.0
Whole	475,000	2.5	1,211,000	1.4

Assume the reserve is the same as the resource

Waste (60 degree pit slopes)

	(m^3)	(SG)	(t)	Strip Ratio
West	363,000	2.2	800,000	1.3
Whole	1,243,000	2.2	2,735,000	2.3

The waste quantities have been estimated at a pit slope of 60 degrees, which is optimistic for a wet, easy digging material with clay. Assume that this angle is 45 degrees, including allowances for haul roads. The geometry is shown in an attached diagram with an estimate of the increase in area (and hence volume) for the lesser pit slope.

Waste (45 degree pit slopes)

Assume 45 degree pit slopes, as shown on the attached representative sections.
The additional crossectional area is shown in the table below.

	Area at 60 Degrees (m^2)	Additional Area at 45 Degrees (m^2)
Cross-section 1	900	45
Cross-section 2	4,400	2,250
Cross-section 3	6,200	2,300
Cross-section 4	7,300	5,700
Total	18,800	10,295

Adjustment factor for the strip ratio 55%

	(m^3)	(SG)	(t)	Strip Ratio
West	561,781	2.2	1,238,085	2.0
Whole	1,923,675	2.2	4,232,703	3.5

Conceptual Resource

Assume a production rate of 750,000 tonnes per annum and a 10 year life for the orebody at the same grade as the whole of the King Lyell orebody

	(m^3)	(SG)	(t)	(% Cu)	Strip Ratio
Resource	2,942,000	2.5	7,500,000	1.4	-
Waste (45 deg)	11,914,000	2.2	26,214,000	-	3.5

KING LYELL COPPER CLAYS
250,000 TONNE PER YEAR PRODUCTION

By:	Tony Weston	
Date:	31/07/97	
File:	g:\mining\engineer\tony\excel\klcc.xls	

This is is based on the existing resource/reserve at the King Lyell

Mining Physicals

Mining - ore	250,000	tpa
- waste	874,000	tpa
Stripping ratio	3.5	
Milling	250,000	tpa

Assume average haul for ore and waste from top of pit ramp to ore or waste stockpiles as follows. Also average haul depth in pit at average gardient

Haul distance away from pit	500	m
Average depth in pit	50	m
Pit haul road gradient 1 in	10	
Haul distance	1,000	m

Mining Costs

The mining would be done by a contractor who would deliver ore to a stockpile at the mill bin and waste to an adjacent waste stockpile. No allowance has been made for increasing haul distance at depth (say 100 m), so this average cost over estimates the cost at project end. The waste cost includes waste dump management

Mining costs for oxide ore at Reedy for following 1996/97 production

Ore production	456,000	t
Waste	2,865,000	bcm
	2.0	t/m^3
	5,730,000	t
Loading	$ 0.25	/t
Hauling	$ 0.89	/t.km

Increase these costs to take account of more difficult digging, haul road conditions and scale of production. Allow extra for waste dump management

Scale down factor	1.5	
Ore - loading	$ 0.38	/t
- hauling	$ 1.78	/t
Total ore	$ 2.16	/t
Waste - haul (as for ore)	$ 2.16	/t
- waste dump	$ 0.50	/t
Total waste	$ 2.66	/t

Miling Physicals

Milling	250,000	tpa
Recovery	65%	
Concentrate grade	45%	

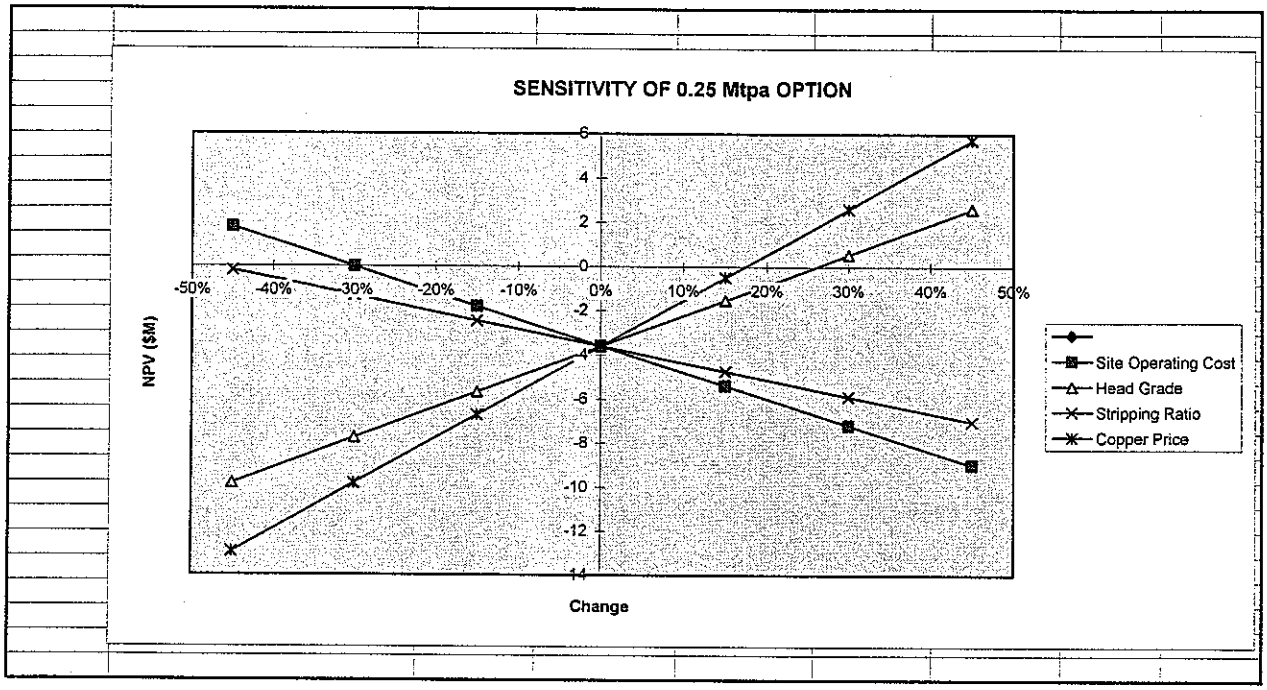
Mill Operating Costs

The mill operating costs were developed by Nick Clarke for a flow sheet shown elsewhere and cover the costs of picking up the ore at the stockpile, through treatment to be ready for haulage to the concentrates shed. These have been developed on an annual basis.

Item	Activity		Rate		Total ($)	Rate ($/t)
Front end loader	(hrs)	7,300	($/hr)	35	255,500	1.02
Operators	(ea)	6	($/year)	50,000	300,000	1.20
Supervisor	(ea)	1	($/year)	60,000	60,000	0.24
Metallurgist	(ea)	0.5	($/year)	70,000	35,000	0.14

On Costs		30%			118,500	0.47
Power					70,000	0.28
Consumables					40,000	0.16
Maintenance Spares					200,000	0.80
Maintenance manpower	(hrs)	4,380	($/hr)	35	153,300	0.61
Flocculant					15,000	0.06
Miscellaneous					30,000	0.12
Management Overhead		10%			39,500	0.16
Total					1,316,800	5.27
Realisation Costs		Activity	Rate ($US)	Rate ($A)		
Exchange Rate	0.75					
Haulage of cons to cons shed	(t)	Concentrates	-	2.00		
Truck, rail haulage and wharfage	(t)	Concentrates	-	25.60		
Shipping	(t)	Concentrates	28.00	37.33		
Smelting charges	(t)	Concentrates	90.00	120.00		
Refining charge	(lb)	Copper	0.09	0.12		
Mill Capital Costs						
Mill throughput rate	250,000	tpa				
Mill availablity	85%					
Mill throughput rate	34	tph				
Scaled down from a larger mill in the Australian Costs Estimation Handbook						
Plant throughput rate	150	tph				
Case 10A		kW	Steel (t)	Mechanical (t)		
Feed Preparation		360	120	200		
Nuggets		90	50	60		
Fine		320	90	60		
Total		770	260	320		
Scale power pro-rata on throughput rate, other items pro-rata to the power of 0.6						
as per discussions with Nick Clarke						
King Lyell Plant		kW	Steel (t)	Mechanical (t)		
Feed Preparation		81	49	81		
Nuggets		20	20	24		
Fine		72	37	24		
Total		172	106	130		
Factor	($)	375	4,000	11,000		
Cost	($)	64,632	423,632	1,433,831		
Total Cost	($)	1,922,000				
Factor		2.5				
Total	($)	4,805,000				
Tails Dam	($)	1,000,000				
Total Mill Capital	($)	5,805,000				
Schedule						
Years		1 to 4	5			
Mining - ore	(t)	250,000	211,000			
Mining - waste	(t)	874,000	737,000			
Milling	(t)	250,000	211,000			
Grade	(%Cu)	1.4	1.4			
Copper in feed	(t)	3,500	2,954			
Recovery	(%)	65	65			

Copper in concentrates		(t)	2,275	1,920				
Concentrate grade		(%)	45	45				
Concentrates		(t)	5,056	4,267				
Smelter copper recovery		(%)	95	95				
Payable copper		(t)	2,161	1,824				
Payable copper		(lb)	4,763,395	4,020,305				
Costs								
Years		Rate	1 to 4	5				
		($/unit)	($)	($)				
Mining - ore	$	2.16	538,750	454,705				
Mining - waste	$	2.66	2,320,470	1,956,735				
Milling	$	5.27	1,316,800	1,111,379				
Haulage of cons to cons shed	$	2.00	10,111	8,534				
Total Site Operating Cost			4,186,131	3,531,353				
Truck and rail	$	25.60	129,422	109,232				
Shipping	$	37.33	188,741	159,297				
Smelting charge	$	120.00	606,667	512,027				
Refining Charge	$	0.12	571,607	482,437				
Total			5,682,568	4,794,346				
Revenue								
Copper price		1.00	US$/lb					
		1.00	US$/lb					
		2,204	US$/t					
Exchange Rate		0.75						
Copper Price		2,940	A$/t					
Years		Rate	1 to 4	5				
		($/unit)	($)	($)				
Revenue		2,940	6,354,075	5,362,839				
Discounted Cash Flow								
Discount Rate			15%					
Year	0		$ (5,805,000)					
	1		$ 583,919					
	2		$ 507,756					
	3		$ 441,527					
	4		$ 383,936					
	5		$ 282,642					
NPV			$ (3,605,221)					
NPV Sensitivities							20%	
		-45%	-30%	-15%	0%	15%	30%	45%
Site Operating Cost		1.8	(0.0)	(1.8)	(3.6)	(5.4)	(7.2)	(9.0)
Head Grade		(9.8)	(7.8)	(5.7)	(3.6)	(1.5)	0.5	2.6
Stripping Ratio		(0.2)	(1.3)	(2.5)	(3.6)	(4.7)	(5.9)	(7.0)
Copper Price		(12.9)	(9.8)	(6.7)	(3.6)	(0.5)	2.6	5.7


SENSITIVITY OF 0.25 Mtpa OPTION
NPV ($M)
6
4
2
0
-2
-4
-6
-8
-10
-12
-14
-50%
-40%
-30%
-20%
-10%
0%
10%
20%
30%
40%
50%
Change
Site Operating Cost
Head Grade
Stripping Ratio
Copper Price

KING LYELL COPPER CLAYS
750,000 TONNE PER YEAR PRODUCTION

By:	Tony Weston	
Date:	31/07/97	
File:	g:\mining\engineer\tony\excel\klcc.xls	

This is is based on the existing resource/reserve at the King Lyell

Mining Physicals

Mining - ore	750,000	tpa
- waste	2,621,000	tpa
Stripping ratio	3.5	
Milling	750,000	tpa

Assume average haul for ore and waste from top of pit ramp to ore or waste stockpiles as follows. Also average haul depth in pit at average gardient

Haul distance away from pit	500	m
Average depth in pit	50	m
Pit haul road gradient 1 in	10	
Haul distance	1,000	m

Mining Costs

The mining would be done by a contractor who would deliver ore to a stockpile at the mill bin and waste to an adjacent waste stockpile. No allowance has been made for increasing haul distance at depth (say 100 m), so this average cost over estimates the cost at project end. The waste cost includes waste dump management

Mining costs for oxide ore at Reedy for following 1996/97 production

Ore production	456,000	t
Waste	2,865,000	bcm
	2.0	t/m^3
	5,730,000	t
Loading	$ 0.25	/t
Hauling	$ 0.89	/t.km

Increase these costs to take account of more difficult digging, haul road conditions and scale of production. Allow extra for waste dump management

Scale down factor	1.5	
Ore - loading	$ 0.38	/t
- hauling	$ 1.78	/t
Total ore	$ 2.16	/t
Waste - haul (as for ore)	$ 2.16	/t
- waste dump	$ 0.50	/t
Total waste	$ 2.66	/t

Milling Physicals

Milling	750,000	tpa
Recovery	65%	
Concentrate grade	45%	

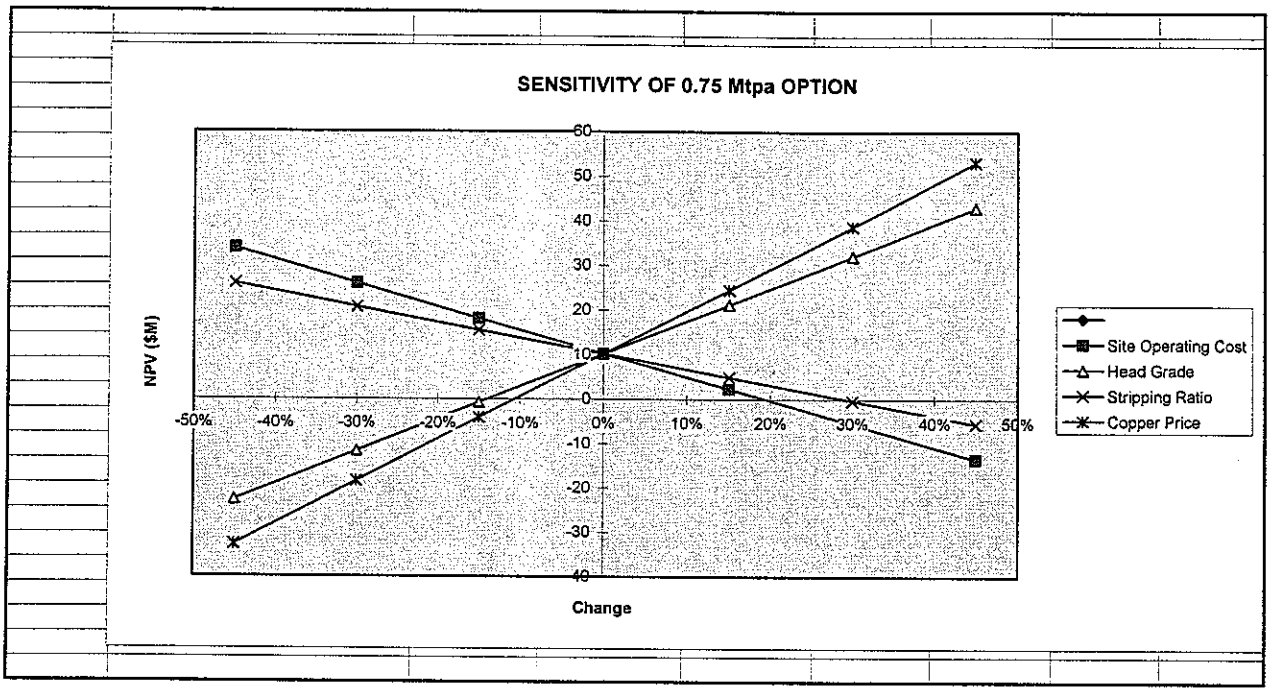
Mill Operating Costs

The mill operating costs were developed by Nick Clarke for a flow sheet shown elsewhere and cover the costs of picking up the ore at the stockpile, through treatment to be ready for haulage to the concentrates shed. These have been developed on an annual basis.

Item	Activity		Rate		Total ($)	Rate ($/t)
Front end loader	(hrs)	7,300	($/hr)	35	255,500	0.34
Operators	(ea)	6	($/year)	50,000	300,000	0.40
Supervisor	(ea)	1	($/year)	60,000	60,000	0.08
Metallurgist	(ea)	0.5	($/year)	70,000	35,000	0.05

On Costs		30%			118,500	0.16
Power					140,000	0.19
Consumables					120,000	0.16
Maintenance Spares					400,000	0.53
Maintenance manpower	(hrs)	8,760	($/hr)	35	306,600	0.41
Flocculant					45,000	0.06
Miscellaneous					30,000	0.04
Management Overhead		10%			39,500	0.05
Total					1,850,100	2.47
Realisation Costs		Activity	Rate ($US)	Rate ($A)		
Exchange Rate	0.75					
Haulage of cons to cons shed	(t)	Concentrates	-	2.00		
Truck, rail haulage and wharfage	(t)	Concentrates	-	25.60		
Shipping	(t)	Concentrates	28.00	37.33		
Smelting charges	(t)	Concentrates	90.00	120.00		
Refining charge	(lb)	Copper	0.09	0.12		
Mill Capital Costs						
Mill throughput rate	750,000	tpa				
Mill availablity	85%					
Mill throughput rate	101	tph				
Scaled down from a larger mill in the Australian Costs Estimation Handbook						
Plant throughput rate	150	tph				
Case 10A		kW	Steel (t)	Mechanical (t)		
Feed Preparation		360	120	200		
Nuggets		90	50	60		
Fine		320	90	60		
Total		770	260	320		
Scale power pro-rata on throughput rate, other items pro-rata to the power of 0.6						
as per discussions with Nick Clarke						
King Lyell Plant		kW	Steel (t)	Mechanical (t)		
Feed Preparation		242	94	157		
Nuggets		60	39	47		
Fine		215	71	47		
Total		517	205	252		
Factor	($)	375	4,000	11,000		
Cost	($)	193,896	818,958	2,771,857		
Total Cost	($)	3,785,000				
Factor		2.5				
Total	($)	9,463,000				
Tails Dam	($)	1,000,000				
Total Mill Capital	($)	10,463,000				
Production						
Years		1 to 10				
Mining - ore	(t)	750,000				
Mining - waste	(t)	2,621,000				
Milling	(t)	750,000				
Grade	(%Cu)	1.4				
Copper in feed	(t)	10,500				
Recovery	(%)	65				

Copper in concentrates		(t)	6,825					
Concentrate grade		(%)	45					
Concentrates		(t)	15,167					
Smelter copper recovery		(%)	95					
Payable copper		(t)	6,484					
Payable copper		(lb)	14,290,185					
Costs								
Years		Rate	1 to 10					
		($/t)	($)					
Mining - ore	$	2.16	1,616,250					
Mining - waste	$	2.66	6,958,755					
Milling	$	2.47	1,850,100					
Haulage of cons to cons shed	$	2.00	30,333					
Total Site Operating Cost			10,455,438					
Truck and rail	$	25.60	388,267					
Shipping	$	37.33	566,222					
Smelting charge	$	120.00	1,820,000					
Refining Charge	$	0.12	1,714,822					
Total			14,944,749					
Revenue								
Copper price		1.00	US$/lb					
		1.00	US$/lb					
		2,204	US$/t					
Exchange Rate		0.75						
Copper Price		2,940	A$/t					
Years		Rate	1 to 10					
		($/t)	($)					
Revenue		2,940	19,062,225					
Discounted Cash Flow								
Discount Rate			15%					
Year	0		$(10,463,000)					
	1		$ 3,580,414					
	2		$ 3,113,403					
	3		$ 2,707,307					
	4		$ 2,354,180					
	5		$ 2,047,113					
	6		$ 1,780,098					
	7		$ 1,547,912					
	8		$ 1,346,010					
	9		$ 1,170,444					
	10		$ 1,017,777					
NPV			$ 10,201,657					
NPV Sensitivities								
		-45%	-30%	-15%	0%	15%	30%	45%
Site Operating Cost		33.8	25.9	18.1	10.2	2.3	(5.5)	(13.4)
Head Grade		(22.6)	(11.7)	(0.7)	10.2	21.1	32.1	43.0
Stripping Ratio		25.9	20.7	15.4	10.2	5.0	(0.3)	(5.5)
Copper Price		(32.8)	(18.4)	(4.1)	10.2	24.5	38.8	53.2


SENSITIVITY OF 0.75 Mtpa OPTION
NPV ($M)
60
50
40
30
20
10
0
-10
-20
-30
-40
-50%
-40%
-30%
-20%
-10%
0%
10%
20%
30%
40%
50%
Change
Site Operating Cost
Head Grade
Stripping Ratio
Copper Price

REEDY EXCAVATION AND HAULING COSTS

By:	Tony Weston
Date:	30/07/97
File:	g:\mining\engineer\tony\excel\reeexc.xls

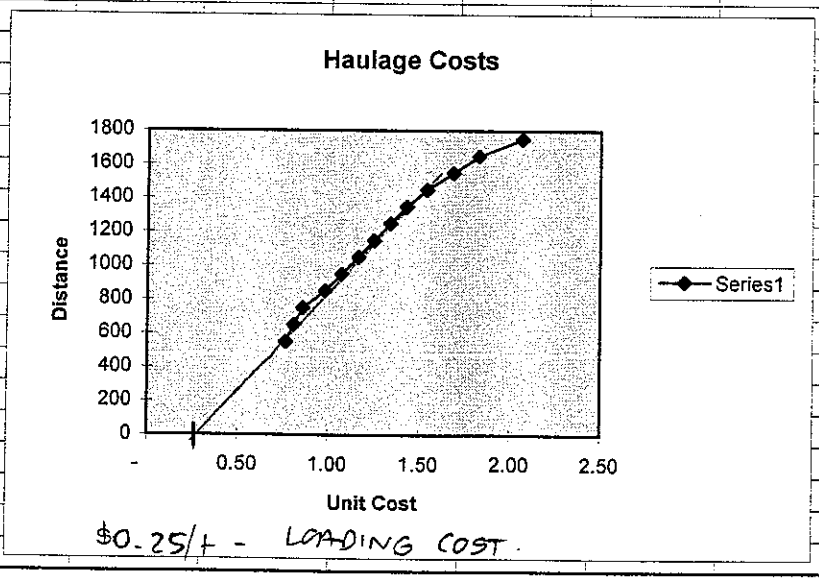
Assume

Haul road	1 in	10	
Haul to stockpile		500	m
Density of ore		2.0	t/m^3

Rates include digging, loading and haulage

In situ elevation	Rate ($/BCM)	Rate ($/t)	Haul Distance (m)	Rate Haulage Only ($/t)	Rate Haulage Only ($/t.km)
-5	1.52	0.76	550	0.51	0.93
-15	1.61	0.81	650	0.56	0.85
-25	1.71	0.86	750	0.61	0.81
-35	1.96	0.98	850	0.73	0.86
-45	2.14	1.07	950	0.82	0.86
-55	2.33	1.17	1050	0.92	0.87
-65	2.50	1.25	1150	1.00	0.87
-75	2.68	1.34	1250	1.09	0.87
-85	2.86	1.43	1350	1.18	0.87
-95	3.08	1.54	1450	1.29	0.89
-105	3.37	1.69	1550	1.44	0.93
-115	3.65	1.83	1650	1.58	0.95
-125	4.13	2.07	1750	1.82	1.04
Average					0.89

Loading Cost (from Graph) $ 0.25 /t



$0.25/t - LOADING COST.

MTECH PTY LTD
CONSULTANTS IN MINERAL PROCESSING
P.O. Box 403 Kalgoorlie Western Australia 6430 Telephone/Fax: (090) 91 1815 A.C.N. 056 594 364
P.O. Box 710 Applecross Western Australia 6153 Telephone/Fax: (09) 472 12

Job:
Job No. Date: 23/7/97
Prepared: NCC Sheet: 1

TITLE: Copper Clays – 250000 tpa plant

Based on Australian Cost Estimation Handbook

Case 10A	kW	Steel, t	Mechanical, t
Feed Preparation	360	120	200
Oversize Inspection	0	0	0
Nuggets	90	50	60
Coarse	0	0	0
Fine	320	90	60
TOTAL	770	260	320

Basis: 150 tph capacity SCALING $(\frac{35}{150})^{0.6}$

Scale to 35 tph capacity (250 000 tpa @ 85% Avail)

Feed Preparation	84	60	80
Nuggets	40	20	24
Fine	130	40	24
TOTAL	254	120	128
$ Factor	375	4000	11000
COST	95250	480,000	1,408,000

Total Equipment $1,983,250

Factor Up ×2.5 $4,958,125

Add Tails Dam $1,000,000

TOTAL CAPITAL COST
PLANT + TAILS DAM = $6,000,000

Operating Cost

	$ pa
Front end Loader @ $35/h Dry Hire	260,000
Operators (2 positions, shift)	400,000
Supervisor (1)	85,000
Metallurgist (1/2)	45,000
Power	70,000
Consumables (screens, balls, cyclones, liners)	40,000
Maintenance spares	200,000
Maintenance manpower	150,000
Flocculant	15,000
Miscellaneous	30,000
Management Overhead @ 10% Manpower	50,000
TOTAL	1,345,000

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P.O. Box 710 Applecross
Western Australia 6153
Telephone/Fax: (09) 472 123

Job:
Job No. Date: 23/7/97
Prepared: NCC Sheet: (2)

TITLE: Copper Clays – 750000 tpa plant

Capital Cost Plant	$10,000,000	
Tailings Dam	$2000000	
Operating Cost		**$ pa**
Front end loader @$40/h		300,000
Operators		400,000
Supervisor (1)		85,000
Metallurgist (1)		90,000
Power		140,000
Consumables		120,000
Maintenance spares		400,000
Maintenance manpower		300,000
Flocculant		45,000
Miscellaneous		30,000
Management Overhead		50,000
TOTAL		1,960,000

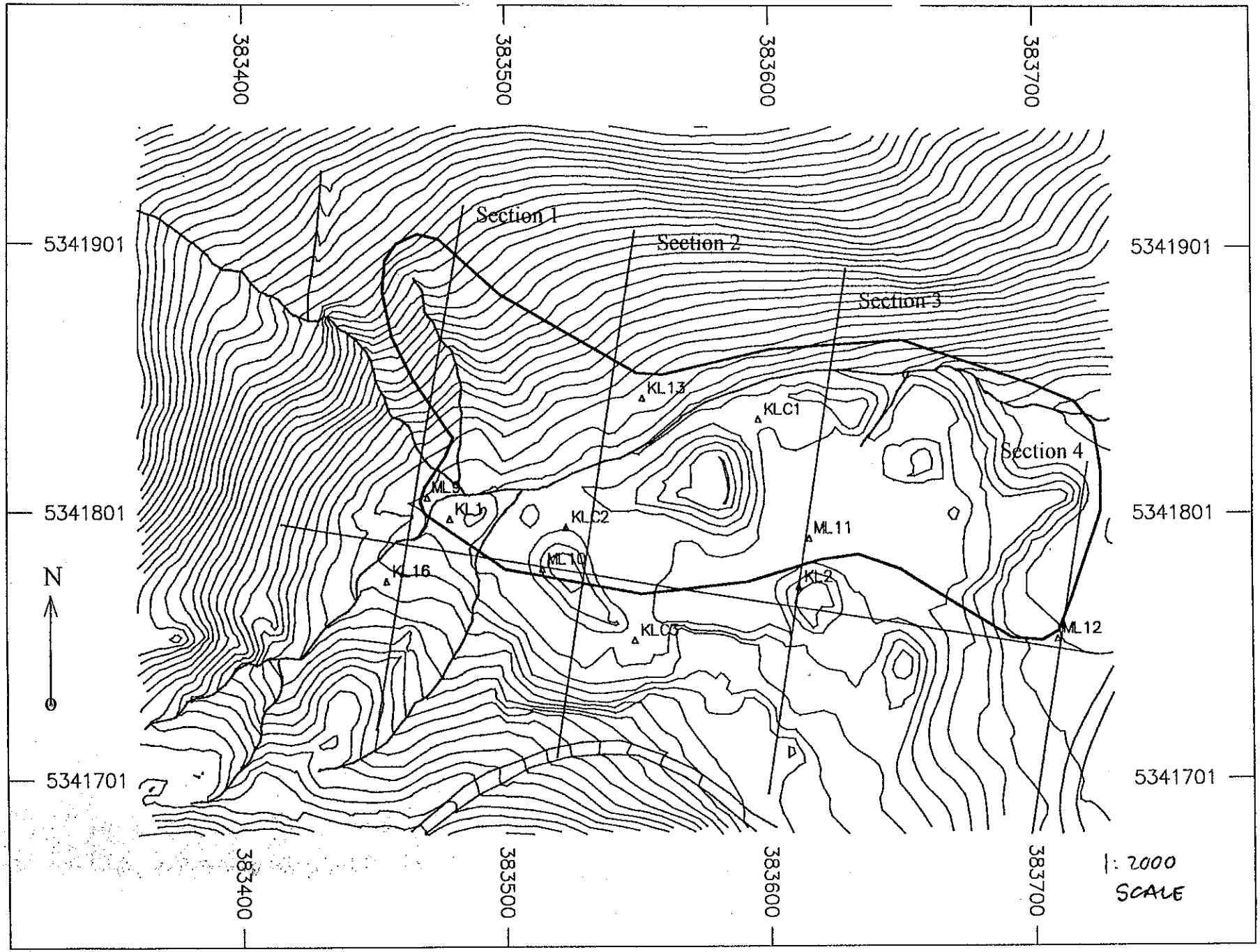
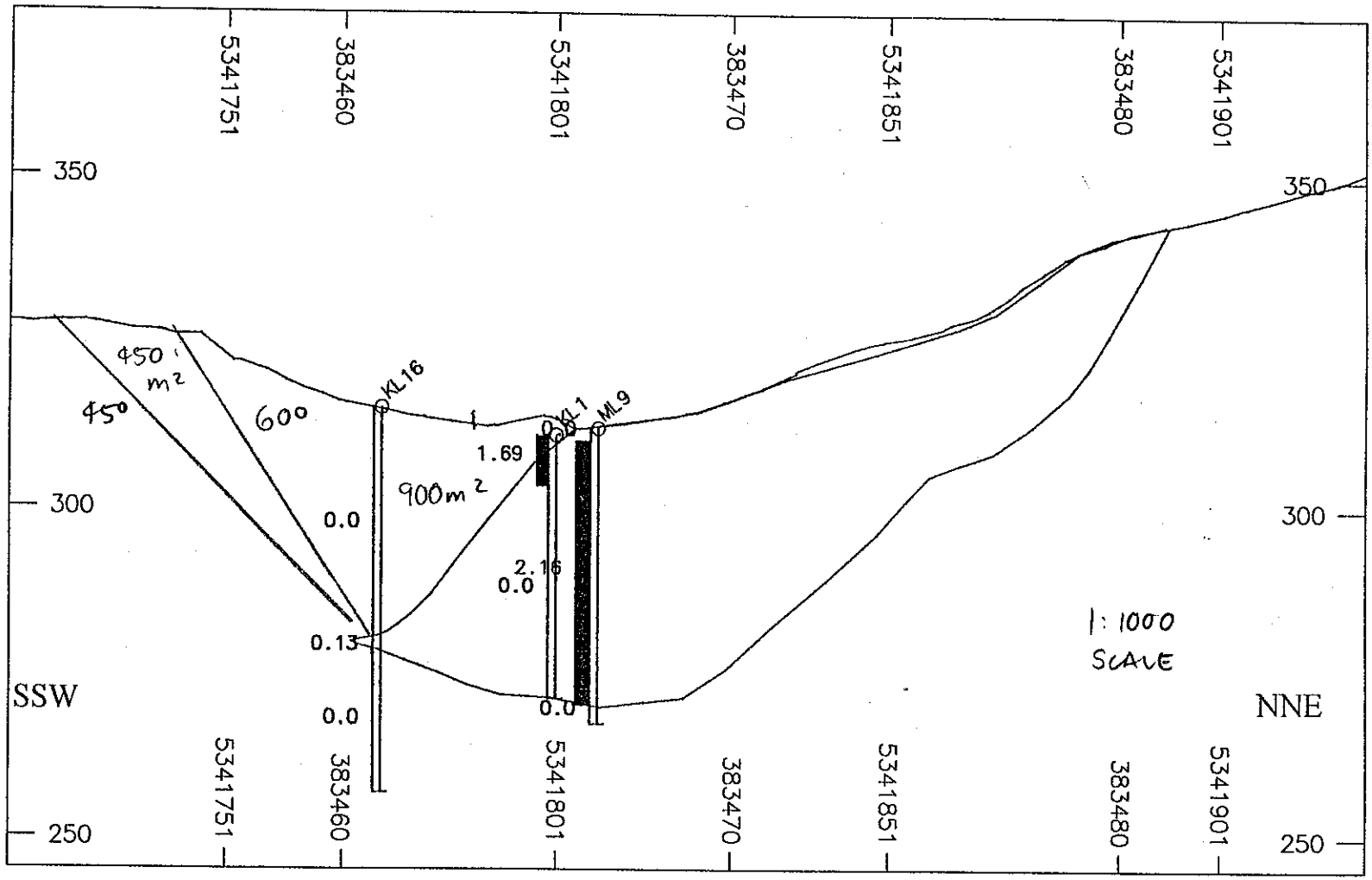


Figure 1. Base map, scale 1:2000, showing topography, creeks, Lyell Highway (red), limit to mineralisation (magenta) drill hole locations (AMG coordinates), and section locations (green)



AJW/31-7-97

Figure 2. Cross Section 1, showing intersection grades in % Cu.

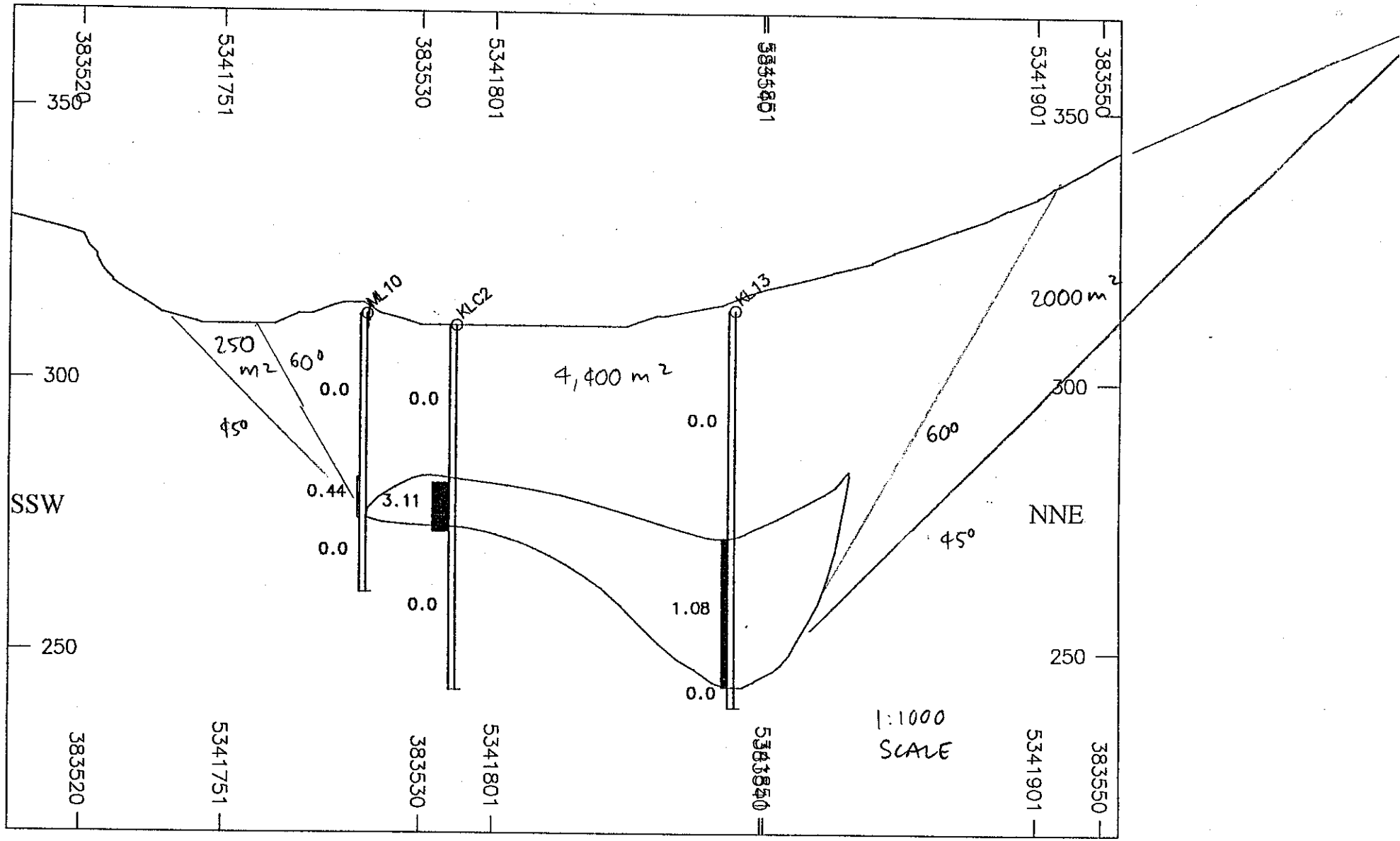


Figure 3. Cross Section 2, showing intersection grades in % Cu.

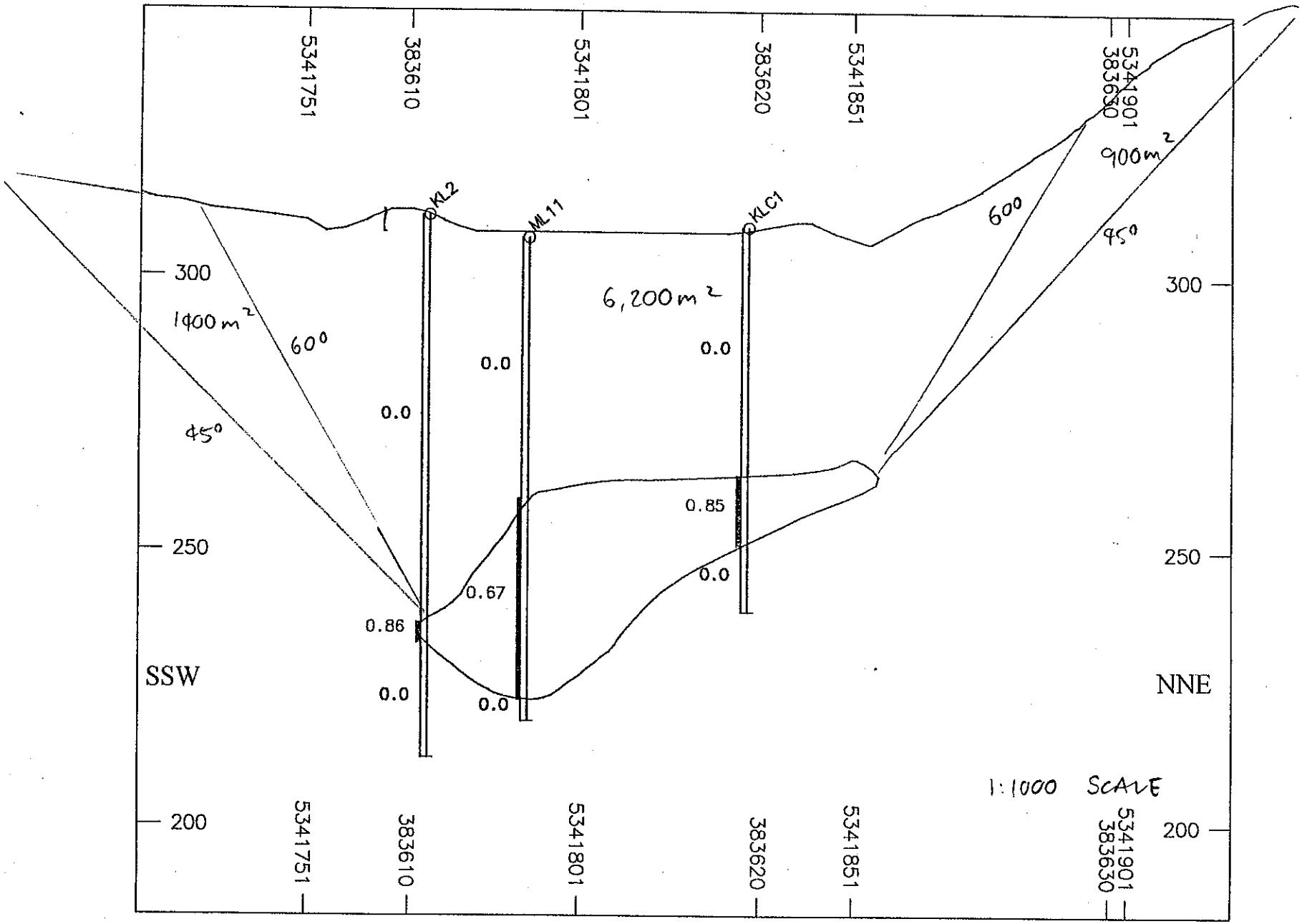


Figure 4. Cross Section 3, showing intersection grades in % Cu.

AOW 31.7.97

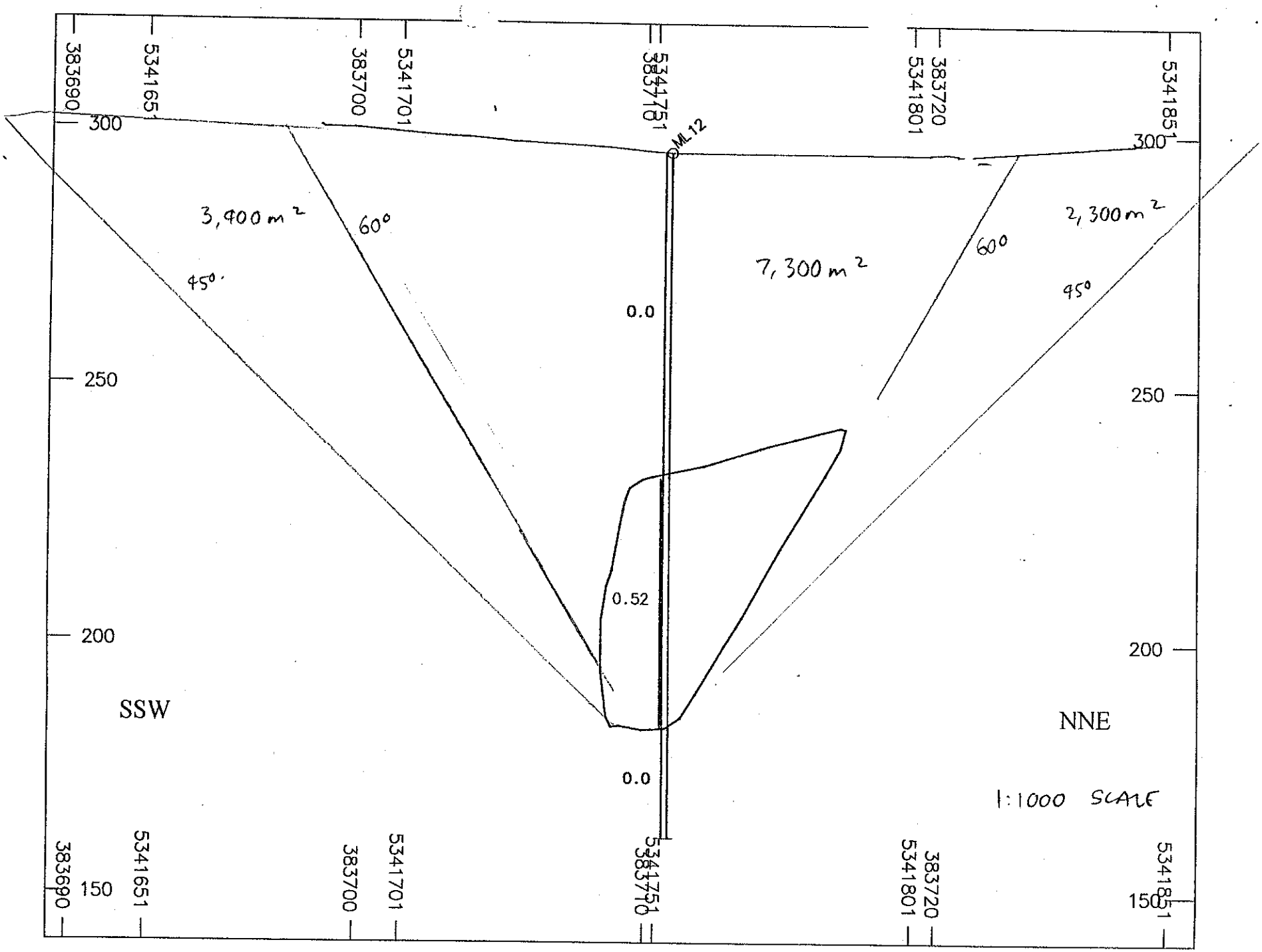
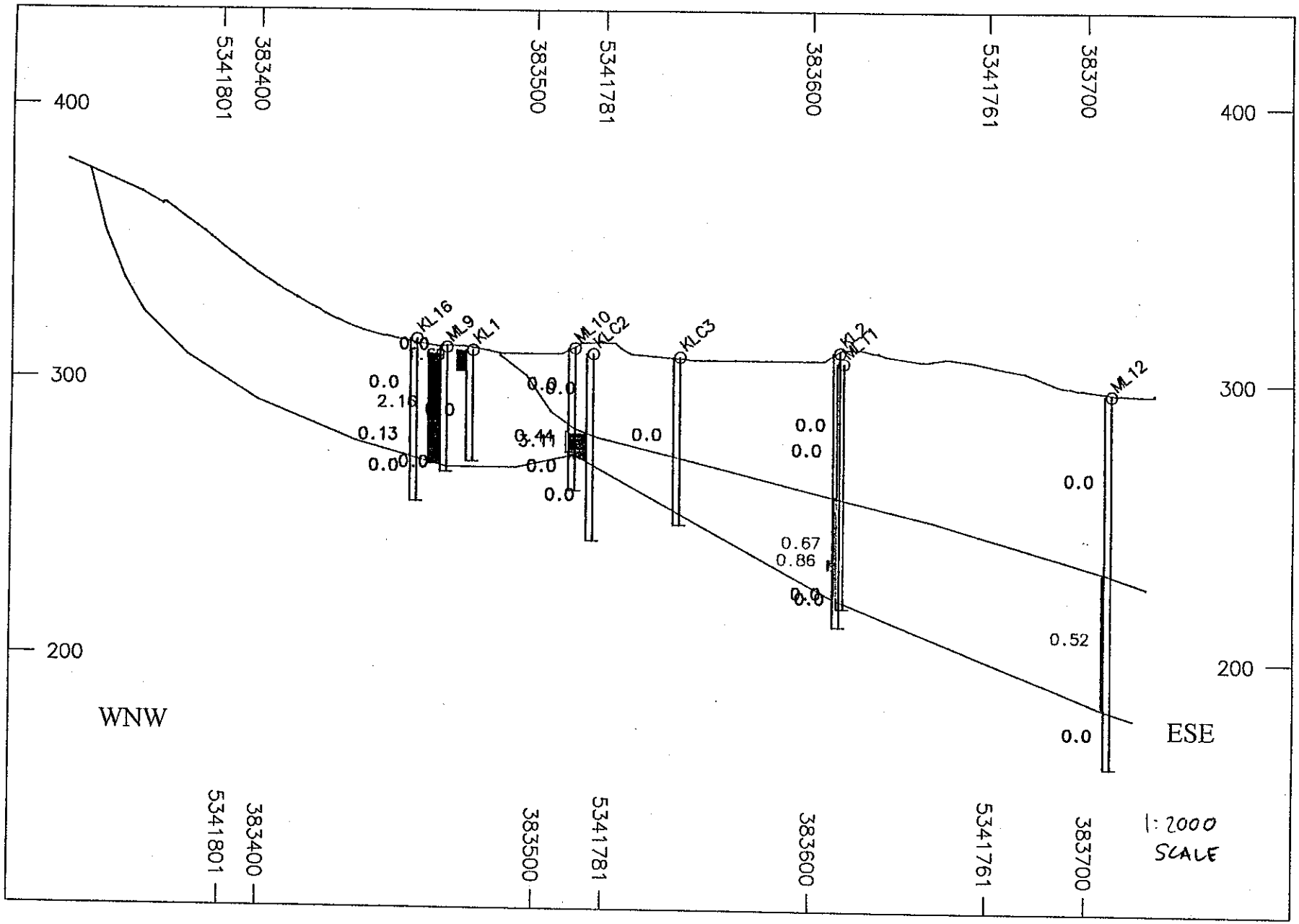


Figure 5. Cross Section 4 through drill hole ML12, showing intersection grades in % Cu.

AW/31-7-97



AJW| 31.7.97

Figure 6. Longitudinal Projection through drill hole ML12, showing intersection grades in % Cu.

APPENDIX C

DRILL HOLES PROVIDED

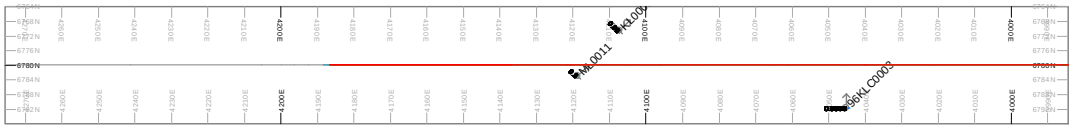
Appendix x1

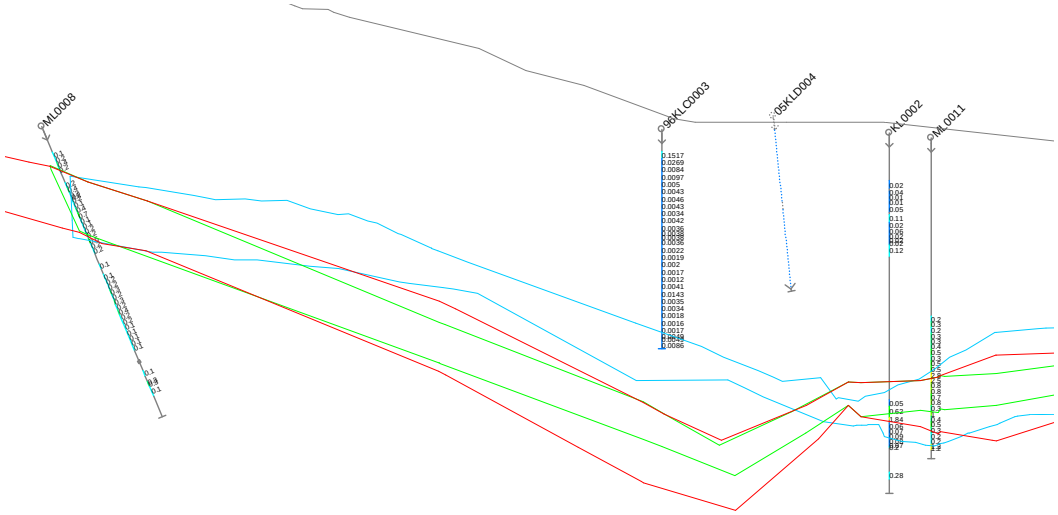
List of holes in drill database

KL0001
KL0002
KL0013
KL0016
ML0008
ML0009
ML0010
ML0011
ML0012
05KLD001
05KLD002
05KLD003
05KLD004
05KLD006
96KLC0001
96KLC0002
96KLC0003

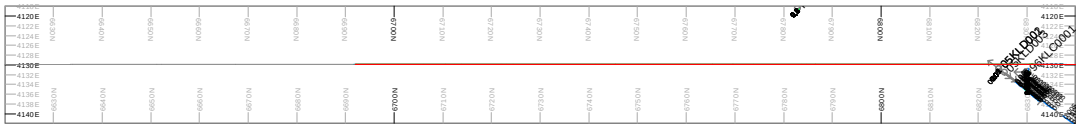
APPENDIX D

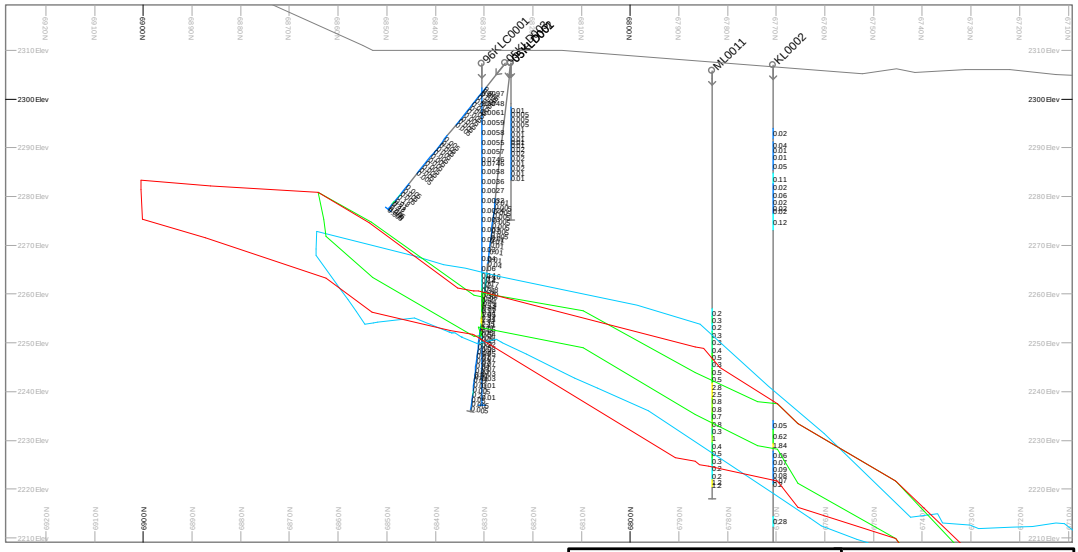
PLOTS OF WIREFRAMES



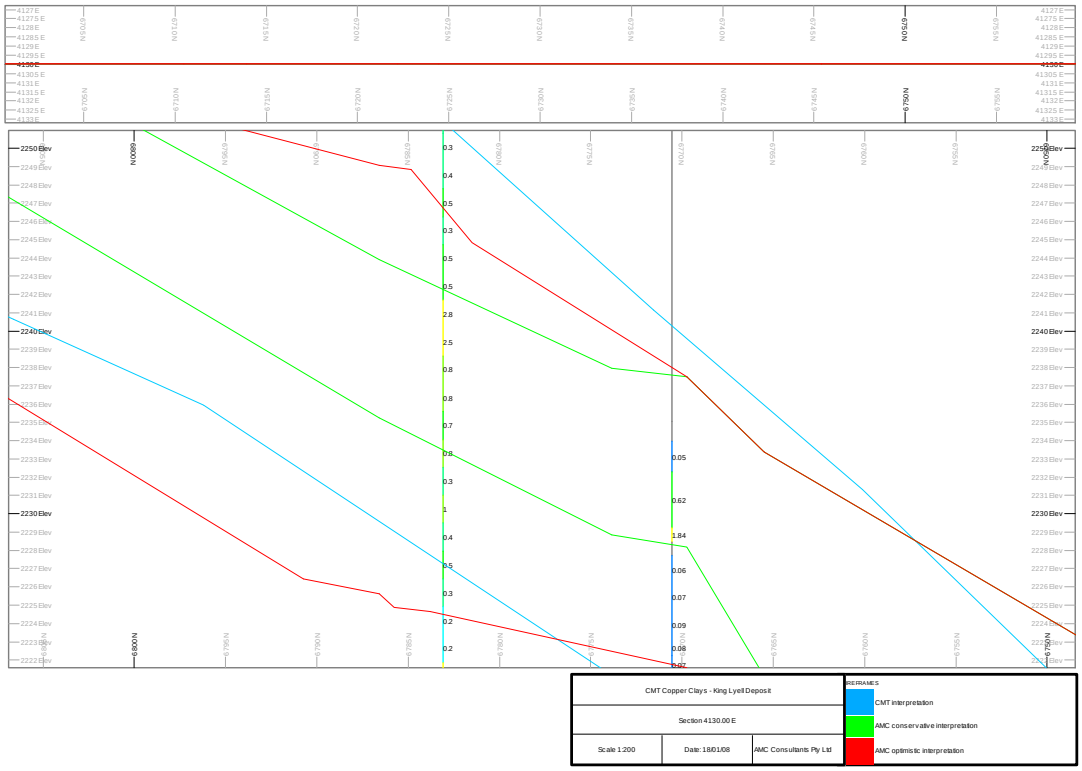


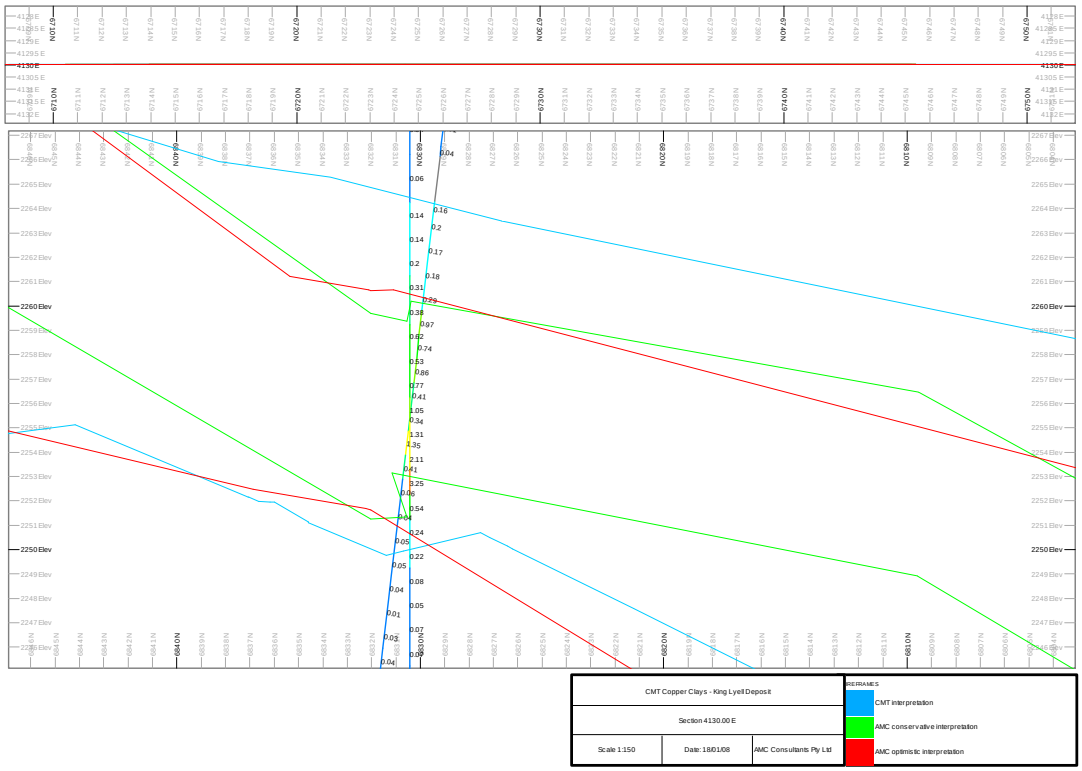
CMT Copper Clays - King Lyell Deposit		VIBERAMES CMT interpretation
Section 6780.00 N		AMC conservative interpretation
Scale 1:1000	Date: 18/01/08	AMC Consultants Pty Ltd AMC optimistic interpretation

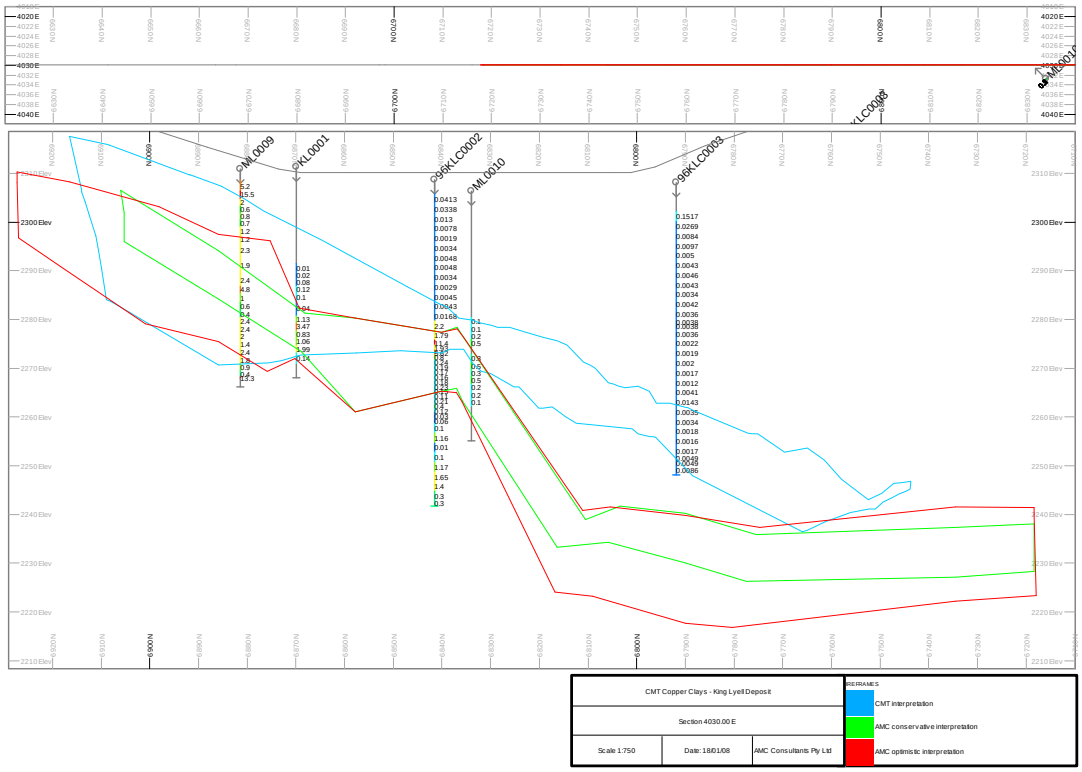




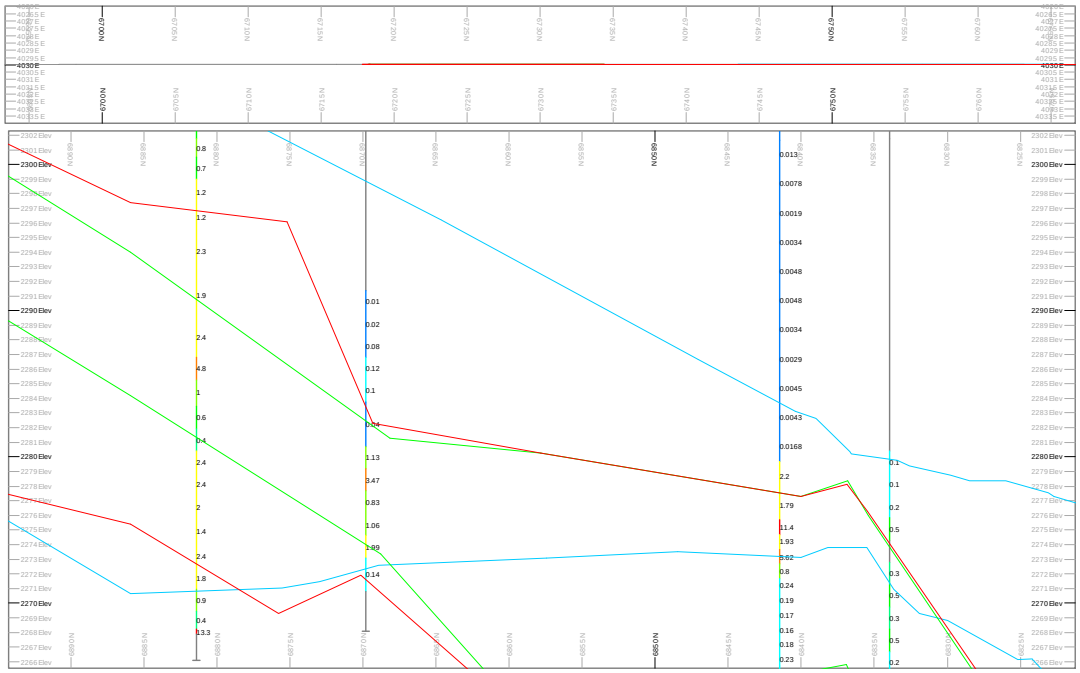
CMT Copper Clays - King Lyell Deposit		VIBERAMES CMT interpretation
Section 4130.00 E		AMC conservative interpretation
Scale 1:750	Date: 18/01/08	AMC Consultants Pty Ltd AMC optimistic interpretation


CMT Copper Clays - King Lyell Deposit
Section 4130.00 E
Scale 1:200
Date: 18/01/08
AMC Consultants Pty Ltd
BERRAMES
CMT interpretation
AMC conservative interpretation
AMC optimistic interpretation


CMT Copper Clays - King Lyell Deposit
Section 4130.00 E
Scale 1:150
Date: 18/01/08
AMC Consultants Pty Ltd
BERRAMES
CMT interpretation
AMC conservative interpretation
AMC optimistic interpretation



CMT Copper Clays - King Lyell Deposit		BERRAMES CMT interpretation
Section 4030.00 E		AMC conservative interpretation
Scale 1:750	Date: 18/01/08	AMC Consultants Pty Ltd AMC optimistic interpretation



CMT Copper Clays - King Lyell Deposit		BERRAMES CMT interpretation
Section 4030.00 E		AMC conservative interpretation
Scale 1:250	Date: 18/01/08	AMC Consultants Pty Ltd AMC optimistic interpretation

APPENDIX E

HOLES USED IN ESTIMATING TONNES AND GRADE

Appendix x3

List of holes used for resource estimation

KL0001
KL0002
KL0013
KL0016
ML0008
ML0009
ML0010
ML0011
ML0012
05KLD002
05KLD006
96KLC0001
96KLC0002

APPENDIX F

COMPARISON OF MODEL CONTENTS

ESTIMATE GENERATED using AMC optimistic wireframe cu_optr/pt

14/01/2008

Global Resource

ZONE	TONNES	Cu%
Waste	########	
Ore	########	1.07

Resource with CUT OFF GRADE > 0.7%

ZONE	TONNES	Cu %
Ore	########	1.53

ESTIMATE GENERATED using AMC conservative wireframe cu_wftr/pt

11/01/2008

Global Resource

ZONE	TONNES	Cu%
Waste	########	
Ore	########	0.90

Resource with CUT OFF GRADE > 0.7%

ZONE	TONNES	Cu %
Ore	########	1.61

ESTIMATE GENERATED USING CMT cu_mintr/pt

17/01/2008

Global Resource

ZONE	TONNES	Cu%
Waste	########	
Ore	########	0.64

Resource with CUT OFF GRADE > 0.7%

ZONE	TONNES	Cu %
Ore	919,227	1.23

RAW STATISTICS

AMC Raw drillhole data subset within cu_optr/pt (Exploration potential)	No. Samples	Max	Min	Mean	Std Dev	Variance	Co-eff Variation
Length (m)	146	14	0.04	1.51	1.14	1.3	0.76
Copper (%)	141	15.5	0.01	1.22	2.13	4.54	1.75
AMC Raw drillhole data subset within cu_wftr/pt (conservative)	**No. Samples**	**Max**	**Min**	**Mean**	**Std Dev**	**Variance**	**Co-eff Variation**
Length (m)	142	14	0.04	1.52	1.15	1.32	N/A
Copper (%)	137	15.5	0.01	1.15	1.88	3.55	N/A
CMT Raw drillhole data subset within cu_mintr/pt	**No. Samples**	**Max**	**Min**	**Mean**	**Std Dev**	**Variance**	**Co-eff Variation**
Length (m)	140	3.05	0.31	1.44	0.51	0.26	N/A
Copper (%)	133	11.4	0.002	0.84	1.38	1.92	N/A

COMPOSITED STATISTICS

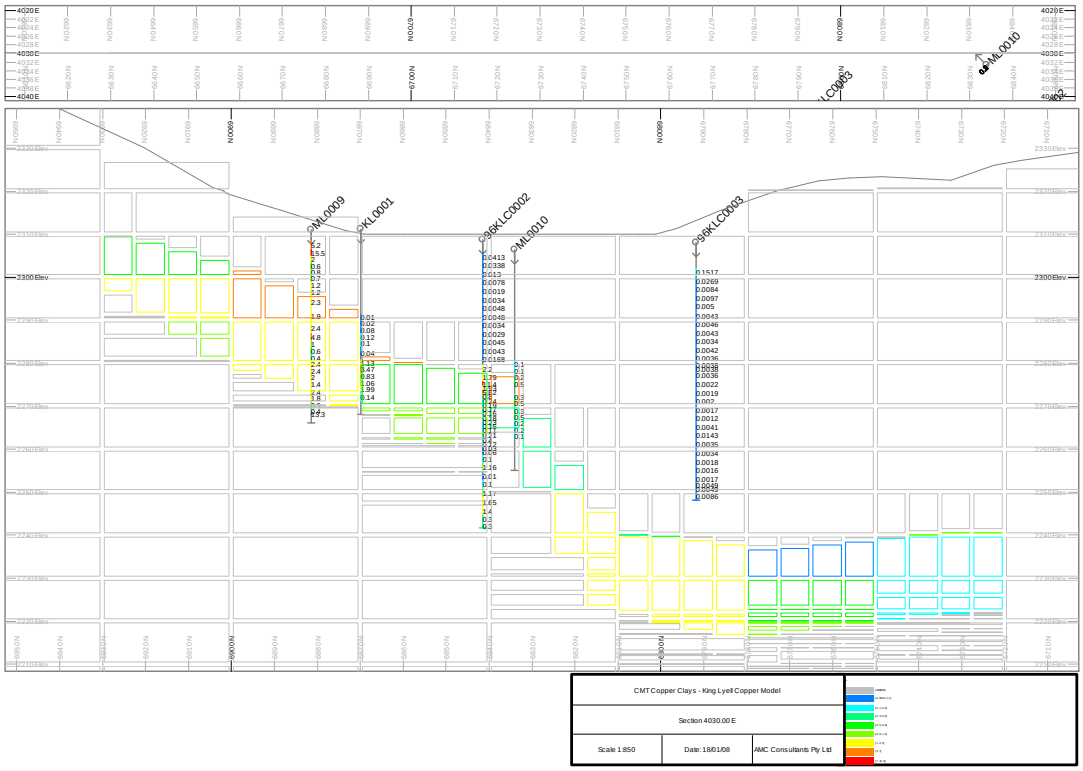
AMC Composited 1m drillhole data subset within cu_optr/pt (Exploration potential)	No. Samples	Max	Min	Mean	Std Dev	Variance	Co-eff Variation
Length (m)	220	0.54	1	0.99	0.06	0.004	0.06
Copper (%)	206	14.16	0.01	1.22	1.8	3.23	1.48

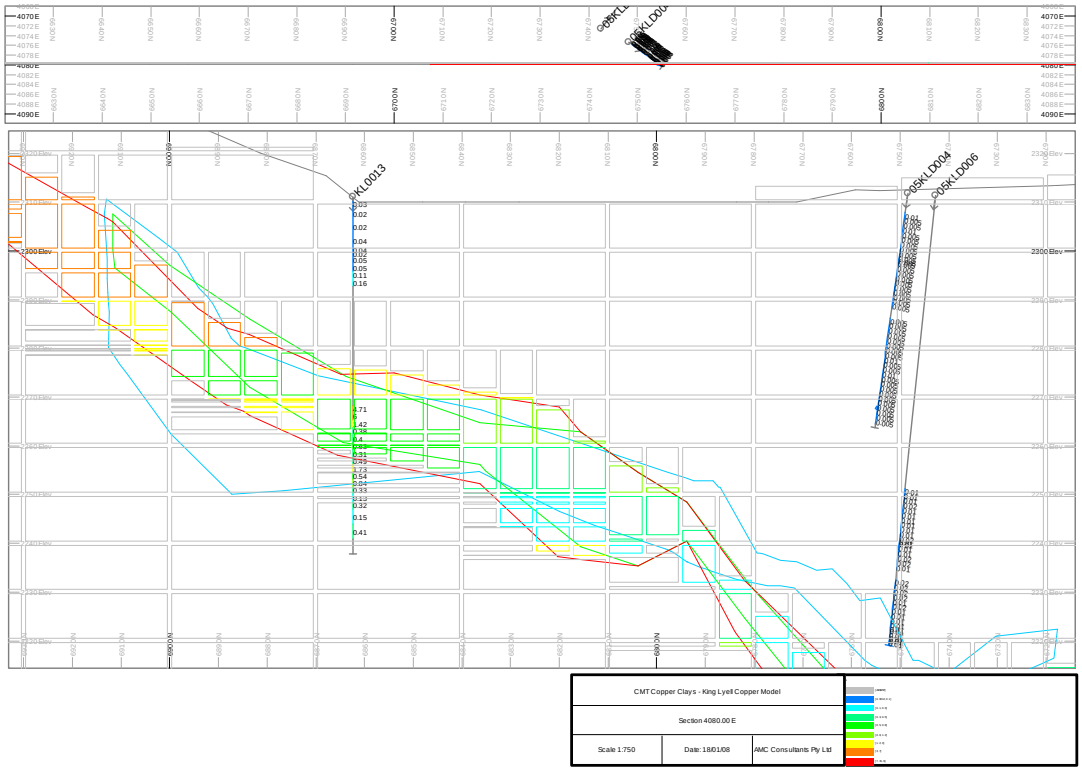
AMC Composited 1m drillhole data subset within cu_wftr/pt (conservative)	No. Samples	Max	Min	Mean	Std Dev	Variance	Co-eff Variation
Length (m)	215	0.54	1	0.99	0.06	0.004	0.06
Copper (%)	201	14.16	0.01	1.21	1.75	3.06	N/A

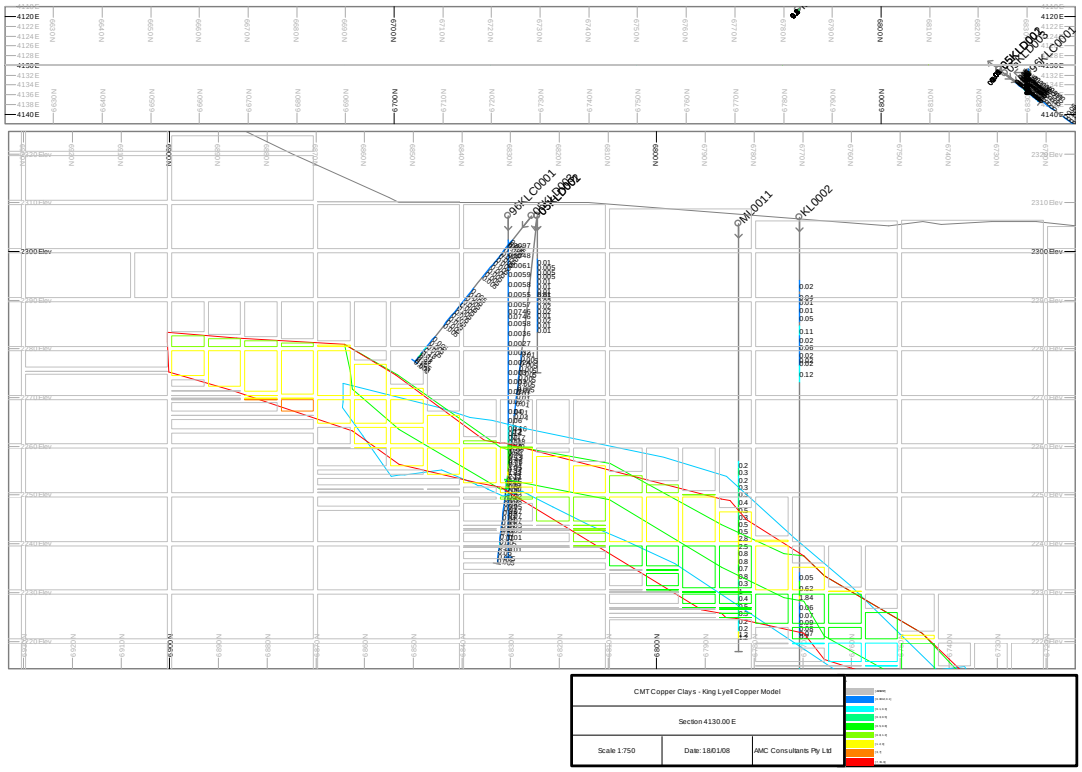
CMT Composited 1m drillhole data subset within cu_mintr/pt	No. Samples	Max	Min	Mean	Std Dev	Variance	Co-eff Variation
Length (m)	201	1	0.50	0.99	0.05	0.002	N/A
Copper (%)	193	11.4	0.002	0.86	1.25	1.57	N/A

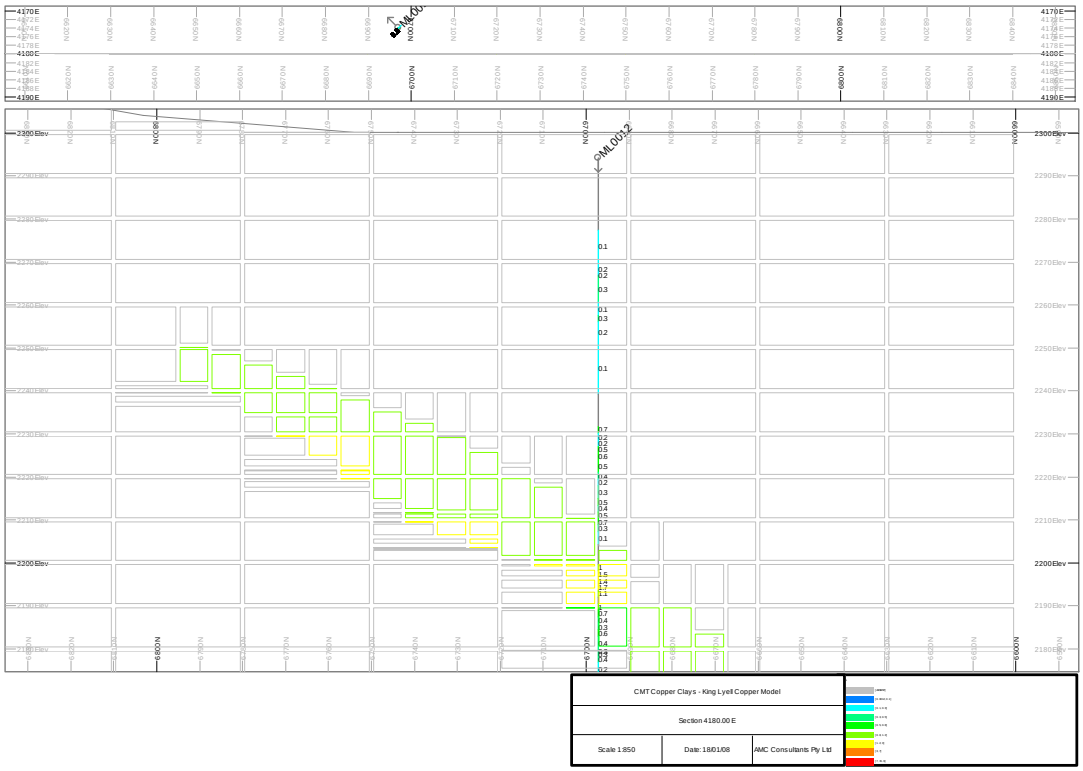
APPENDIX G

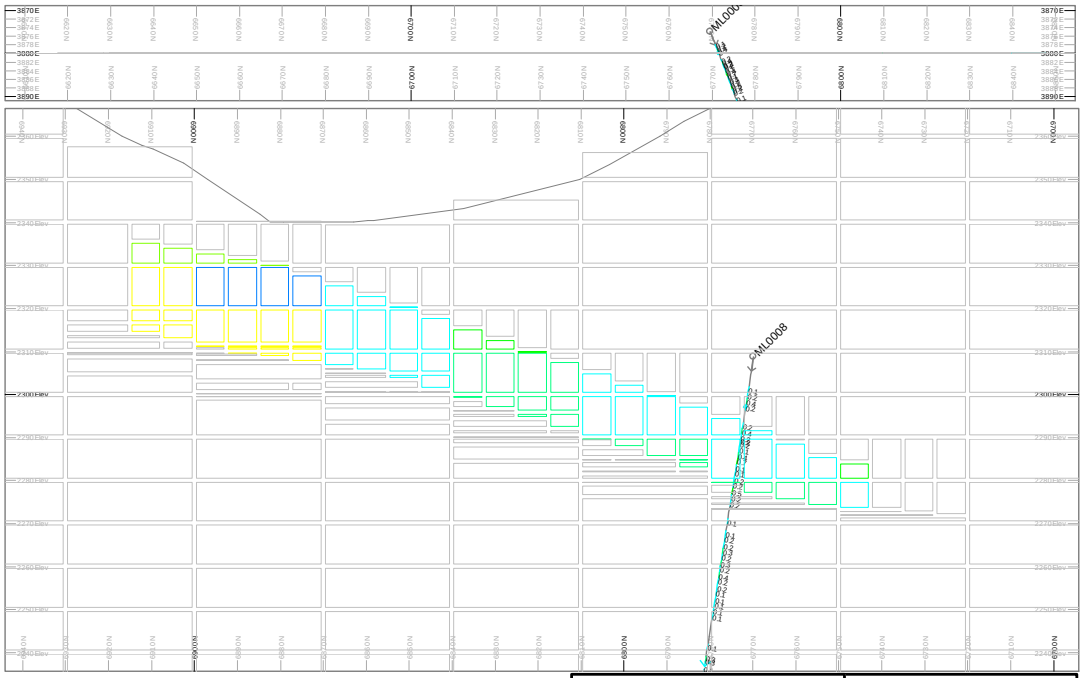
UPSIDE MODEL SECTIONS

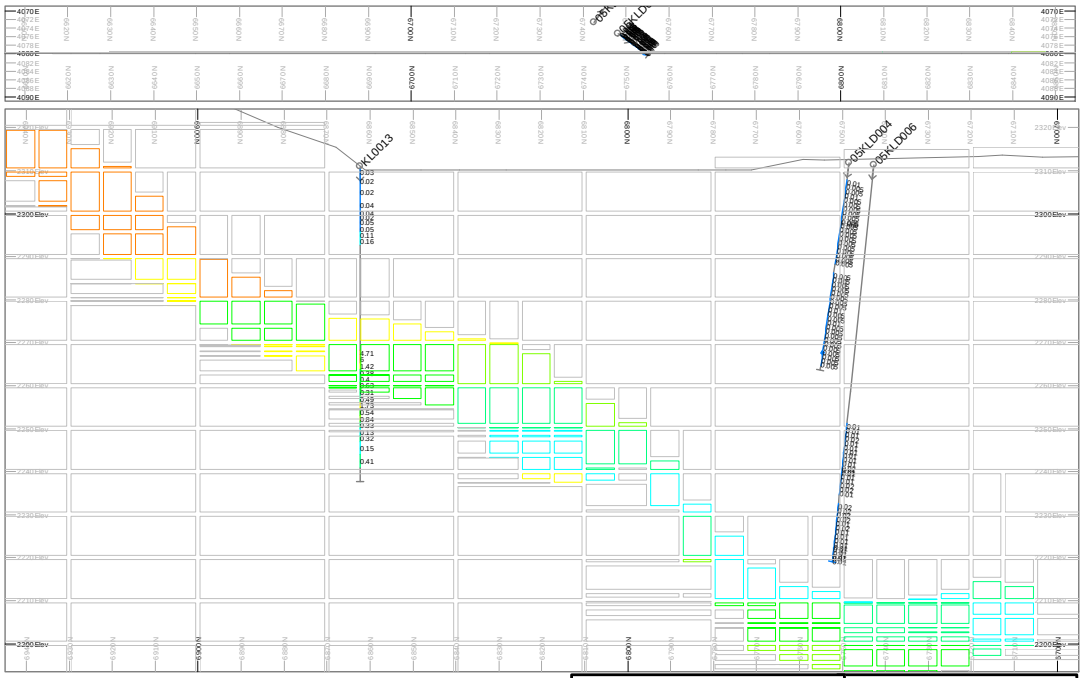

CML0009
PKL0001
O9SKLC0002
CML0010
O9SKLC0003
CML0010
CMT Copper Clays - King Lyell Copper Model
Section 4030.00 E
Scale 1:850
Date: 18/01/08
AMC Consultants Pty Ltd

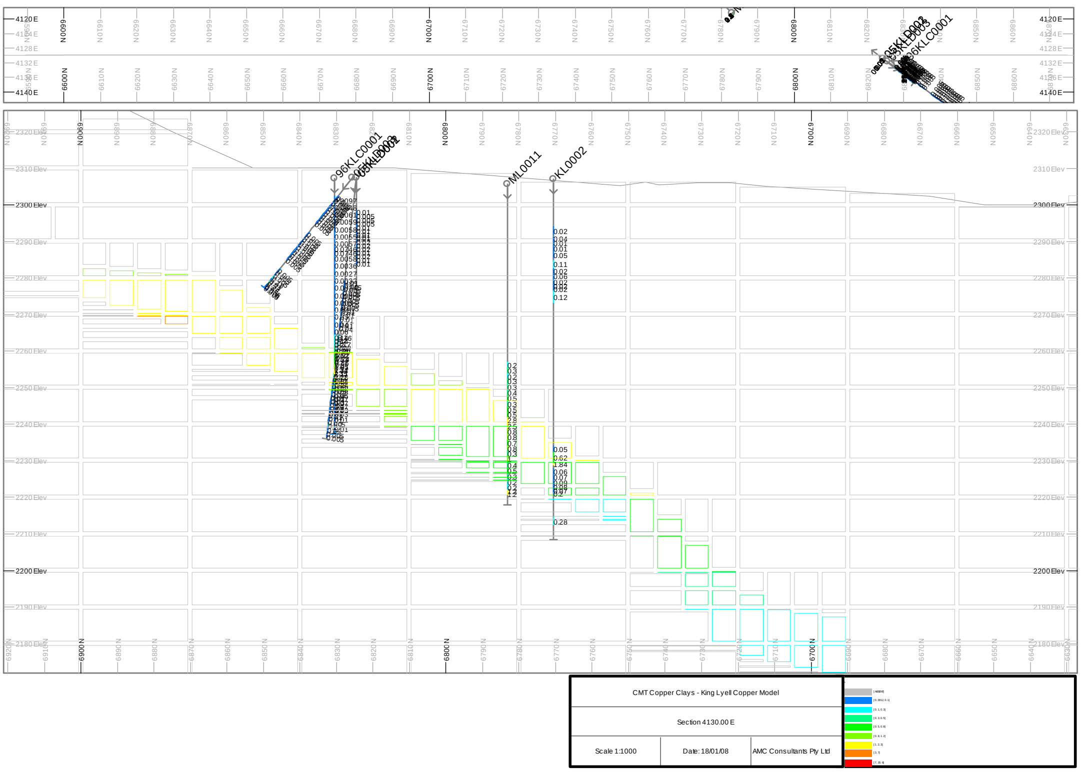

O9SKLD0005
PKL0013
O9SKLD004
O9SKLD006
CMT Copper Clays - King Lyell Copper Model
Section 4080.00 E
Scale 1:750
Date: 18/01/08
AMC Consultants Pty Ltd


CMT Copper Clays - King Lyell Copper Model
Section 4130.00 E
Scale 1:750
Date: 18/01/08
AMC Consultants Pty Ltd


CMT Copper Clays - King Lyell Copper Model
Section 4180.00 E
Scale 1:850
Date: 18/01/08
AMC Consultants Pty Ltd


CML0008
CMT Copper Clays - King Lyell Copper Model
Section 3880.00 E
Scale 1:850
Date: 18/01/08
AMC Consultants Pty Ltd


05KLD0013
05KLD0004
05KLD0006
CMT Copper Clays - King Lyell Copper Model
Section 4080.00 E
Scale 1:850
Date: 18/01/08
AMC Consultants Pty Ltd


CMT Copper Clays - King Lyell Copper Model
Section 4130.00 E
Scale 1:1000
Date: 18/01/08
AMC Consultants Pty Ltd
96KLC0001
05KLD003
05KLD002
ML0011
KL0002
2320 Elev
2310 Elev
2300 Elev
2290 Elev
2280 Elev
2270 Elev
2260 Elev
2250 Elev
2240 Elev
2230 Elev
2220 Elev
2210 Elev
2200 Elev
2190 Elev
2180 Elev
4120 E
4124 E
4128 E
4132 E
4136 E
4140 E
6600 N
6700 N
6800 N
6900 N

APPENDIX H

COPPER CLAYS CORE PHOTOGRAPHS

CMT Copper Clays: KL001 core
Trays 1-2 : ~18.2 m-32.2 m (EOH)

~18.2 m

32.2 m
(EOH)

CMT Copper Clays: KL002 core
Trays 1-4 : ~24.5 m-51.6 m

~24.5 m

~41.5 m

~41.5 m

51.6 m

CMT Copper Clays: KL002 core
Trays 5-8 : 51.6 m-~71.7 m

51.6 m

62.8 m

62.8 m

~71.7 m

CMT Copper Clays: KL002 core
Tray 9 : ~71.7 m-72.2 m (EOH)

~71.7 m

72.2 m
(EOH)

CMT Copper Clays: KL003 core
Trays 1-4 : ~4.7 m-~36.0 m

~4.7 m

~15.7 m

~15.7 m

~36.0 m

CMT Copper Clays: KL003 core
Trays 5 : ~36.0 m-43.0 m (EOH)

~36.0 m

43.0 m (EOH)

CMT Copper Clays: KL004 core
Trays 1-4 : ~3.7 m-31.9 m

~3.7 m

15.0 m

~15.8 m

31.9 m

CMT Copper Clays: KL004 core
Trays 5-8 : 31.9 m-~53.3 m (EOH)

31.9 m

41.4 m

41.4 m

~53.3 m (EOH)

CMT Copper Clays: KL006 core
Trays 1-4 : ~52.0 m-~75.0 m

~52.0 m

~67.0 m

~67.0 m

~75.0 m

CMT Copper Clays: KL006 core
Trays 5,7-8 : ~75.0 m-93.7 m (EOH)

~75.0 m

~91.9 m

~91.9 m

93.7 m (EOH)

12 REFERENCES

Berkman, D. A. 1989 Monograph 9. *Field Geologists Manual*. Third Edition, AUSIMM

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Wills, K. 1995 *Open Cut Potential of the Copper Clays Area Mount Lyell, Tasmania*, Copper Mines of Tasmania Pty Ltd

Hoek, E. and Bray, J. 1981. Rock Slope Engineering. The Institution of Mining and Metallurgy, London, Ed. 3, pp 358.

Wyllie, D. C. and Mah, C. W. 2004. Rock Slope Engineering, Civil and Mining. Spon Press, New York, Ed. 4, pp 431.