KING LYELL COPPER CLAYS

RESOURCE ASSESSMENT

MAY 1997

06_5330
King Lyell Copper Clays Resource Assessment May 1997
Copper Mines of Tasmania Proprietary Limited*
Knight, J.A.; Morrison, K.C. RL3/2006

Copper Clays- Retention Licence Application – Feasibility Statement

The application area for this retention licence lies along the eastern boundary of ML1-1995 and is given in figure 1 and corresponding co-ordinates are given in table 1. A digital file showing this area is provided.[1]

Two previous reports describing the Copper Clays; Open Pit Potential[2] (K. Wills '95) and the King Lyell resource estimation[3] (K. Morrison '97) are provided and are not further summarised here. Each report describes the considerable history of exploration of the Copper Clays while K.Will's report outlines the considerable production and exploration history of these unique mineral resources and former producing mines.

Geological plans and sections outlining the copper clays King Lyell resource are provided in the c.1997 resource estimate by (Morrison '97).
Drill hole locations and assays for current (2005) and pre 2005 drilling are also provided in the compact disc provided.

In situ resources (c.1997) were determined by three-dimensional modelling of the mineralised domain with the aid of surface mapping, assaying and diamond drill holes. A specific cut-off grade has not been established to accompany the economic assessment and specific gravity has not been routinely measured from drill core. The deposit consists of mineralised clays and the specific gravity is assumed to be 2.55g/cm3
"Tonnages were calculated using densities supplied by CMT of 2.55 for mineralised rock [clays] and 2.2 for waste"[4]

The resource estimate was completed in 1997 and the six drill-holes completed in 2005 confirmed the geometry shown on section #3 (6493E) as estimated by Morrison. The 1997 resource was not re-estimated, as the recent holes constrained the 1997 interpretation .

Summaries of in situ resources classified as inferred resources in accordance with JORC are found in the King Lyell resource estimation and this resource has been reported in the March 2006 Mineral resource statement for Copper Mines of Tasmania[5].

Estimates of recoverable reserves and grades are given in the economic assessment "KLCC_2006.xls"[6]. This economic assessment was first authored by Tony Weston in March 1997 after preliminary mine design and mill parameters were established in tis assessment. This assessment was termed a feasibility model in 1997 by Tony Weston but because of the lack of critical detail / formal design, should today be properly considered today as an advanced scoping study.
At the time of writing in 1997-8, the economics of the overall Copper Mines site were tenuous and shortly thereafter the company went into administration. Several feasibility assessments of that era were then put on hold indefinitely. Recently, these feasibility studies/ assessments and this one in particular have been reviewed given improved outlooks for copper prices. Current Vedanta forward copper price assumptions are given as a workbook in the "KLCC_2006.xls file.

Mining methods proposed are described in the economic assessment as a conventional shallow open pit with 60 degree walls.
Tony Weston (Mining Engineer) advised to CMT geologists at the time that 60 degree pit slopes were advised;

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1 ..\..\retention coords\retention licence.str

2 ..\resource estimation\OpenCut potential of the Copper Clays area_ 1995 K. Wills.PDF

3 ..\resource estimation\KL_Cu Clays Resource May97.PDF

4 Kl Cu clays Resource May 97 p. 5

5 ..\resource estimation\Mineral Resource statement Mar 06.pdf

6 KLCC_2006.XLS

" 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." T. Weston Mar 1997

In keeping with the preliminary nature of the 1997 economic assessment, treatment methods were discussed/established by Nick Clarke (Metallurgist) and a proposed mill flowsheet[7] was produced .
This mill flowsheet and a document entitled "Copper Clays Metallurgy –preliminary thoughts",

Metallurgical recovery is estimated at 65% based upon historical recovery from the Lyell Blocks mine (c.1902-1906) and the 1997 review (N. Clarke-preliminary thoughts) discussed above. This data is also supported in K. Will's 1995 report (table 2 Gravity concentrates p. 9) Recent mineralogical testing[8] (c.2005) confirmed Cu mineral species as being native Copper and Chalcocite.
The 2005 drilling campaign in King Lyell Cu Clays provided a 120kg metallurgical sample. This sample was to have been shipped to Burnie research lab for bulk sample testwork consisting of:

i) Desliming in a Mozley cyclone,
ii) Head assay various size fractions
iii) Cu species determination (ore microscopy)
iv) Grinding test
v) Flotation test and tabling of float tail
vi) Flotation and ferric leach of tail

Regrettably, this 120kg sample was split and reduced in error by the onsite lab, the bulk of the sample was accidently discarded, resulting in an insufficient quantity of mineralised sample available for these metallurgical tests to be completed.
The only way to duplicate the sample is to re-drill the deposit.
Product specification has been established as a copper concentrate, to accompany the copper concentrate that CMT provides to Vedantas smelters. The mineral species comprising the concentrate are given in the MODA report Sept '05[8].

Nick Clarke (metallurgist) provided mill operating costs for the milling flow sheet[9] 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.

Infrastructure requirements and discussion of environmental factors including tailings disposal have not been well developed to date owing to the suspension of the 1997 feasibility assessment due to low metal prices and the dire financial conditions experienced by the company at the time.

Estimates of capital and operating costs have been updated as of March 2006 and are outlined in the 250k and 750k worksheets of KLCC_2006.XLS. These assume tailings are treated in CMT's existing facility.

Market assessment: The product is readily saleable copper concentrate, sold to a Vedanta smelter (Titicorin in India)

Price and demand forecasts:

Vedanta 0-5yr price assumptions discussed in 2006-7 fwd Cu prices worksheet of KLCC_2006.XLS.

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7 Cu Clays Mill flowsheet May97.pdf
8 Cu Clays Mineralogy 0905.pdf
9 Cu Clays Mill flowsheet May97.pdf

Environmental Impact information has been provided on form MRDA_R1.doc and outlines CMT's plans to develop the site by:

- Conducting fieldwork consisting of constructing new tracks .
- Prospecting activities which may include; manual digging, bulk sampling and bulk sampling and
- Land disturbance consisting of;
  - small pits (bulk sample) and
  - drilling to correct some serious deficiencies in data quality in the current resource estimate.

Prescribed fees have been paid consisting of :

- Application fee- minerals \$234/km^2
- Annual rent (incl GST)-Minerals \$128.7 per km^2

I refer a letter from Dr. AV (Tony) Brown dated 19 May06 stating:

"please note that………..to grant an application for a retention licence the Minister must be satisfied that:

- *there is sufficient quantity of minerals to justify mining and*
- *that the applicant is justified for economic or other reasons not to proceed to mine. (Mineral Resources Development Act 1995 sections 52(1)(b) and 53(1)(c)).*

In effect this means details of resource studies and economic assessments must be provided. These are most conveniently included in the report on the exploration licence and referred to in the application."

Sufficient quantity of minerals to justify mining -

The inferred resource estimate completed in 1997 by K. Morrison has been reviewed by myself as competent person (as defined) and has been included in Vedanta's Mineral Resource and Mining Ore Reserve statement this year.

Given current economic assessments, the resource as described has sufficient quantity to proceed to mining as shown in the economic assessment. It thus logically qualifies (as one criteria) as an inferred resource as it has reasonable prospects for eventual economic extraction in the current metal price climate.

While complying with JORC requirements, the resource estimate as it stands has serious deficiencies that would preclude it's immediate upgrading to an indicated resource without further work.

Deficiencies include:

- Poor sample recovery (as low as 70%) due to the saturated clay losing form in the drill tube. This recovery problem is not due to driller error, but is a cause of the material itself. A new "sonic" drill recently available in Australia from Boart Longyear may prove to be an effective tool to improve sample recovery.
- Lack of specific gravity determinations. (Can be included in the next phase of delineation diamond drilling)
- Lack of sufficient quality drillholes to allow a confident "high-inferred" classification.
- Lack of sufficient experience with this mineralisation style. This precludes /complicates a rigourous geostatistical estimate because geological continuity is perceived to be along a geo-chemical gradient. This risk is partially offset by successful past production history in adjacent similar deposits (Lyell Consols and Lyell blocks)

The applicant is justified for economic or other reasons not to proceed to mine.

- Resource classification as inferred only, and does not currently include indicated or measured resources; hence proven and possible reserves are not possible without further work.
- Property tenure is not secure, as the retention licence has not been granted and the parts of the existing exploration licence are being relinquished.
- Preliminary feasibility study lacks critical detail to enable an immediate decision to proceed to mining.
- Copper Mines of Tasmania would require a bulk sample and an expanded indicated resource to properly evaluate the full potential of all of the Copper Clays resource, including Consols and Blocks potential resources.

Roger A. Hill P. Geo
Copper Mines of Tasmania

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Contents of CD named: CMT RL3-2006 (D:)

- Exploration licence EL52-1994
    - Chamouni zinc logs
    - CSAMT data
        - Phil Muir appendix D July 05
            - 1_original_CMT_files
            - 2_original_Zonge _files
                - Inversion _ASCII
                - Inversion_sections_geosoft_grd
                - Inversion_sections_images
            - 3_individual_figure_PDF's
            - 4_MapInfo_depth_slices
    - Soil geochem assay data *(need to provide location and meta-data)*
    - Stream sediment assay data *(need to provide location and meta-data)*
- Retention Coords
    - extension to el52-1994.dxf
    - retention licence.str *(need to provide as dxf's when available)*
    - ml_poly.str *(need to provide as dxf's when available)*
- Retention licence 3-2006
    - Economic assessment (mining)
        - Cu Clays Metallurgy-N.Clark 0397.pdf
        - Cu Clays Mill flowsheet May97.pdf
        - Cu Clays Mineralogy 0905.pdf
        - Feasability Statement-Copper Clays.doc
        - KLCC_2006.XLS
        - OP potential Cu clays-K.Wills '95.pdf
    - King Lyell drillholes
        - 2005 Drilling
            - Assays
            - Core logs(6)
        - Pre 2005 drilling
    - Resource Estimation
        - KL_Cu Clays Resource May97.PDF
        - Mineral Resource statement Mar 06.pdf
        - OpenCut potential of the Copper Clays area_ 1995 K. Wills. PDF(2nd copy)
- Annual Report 13 January 2004

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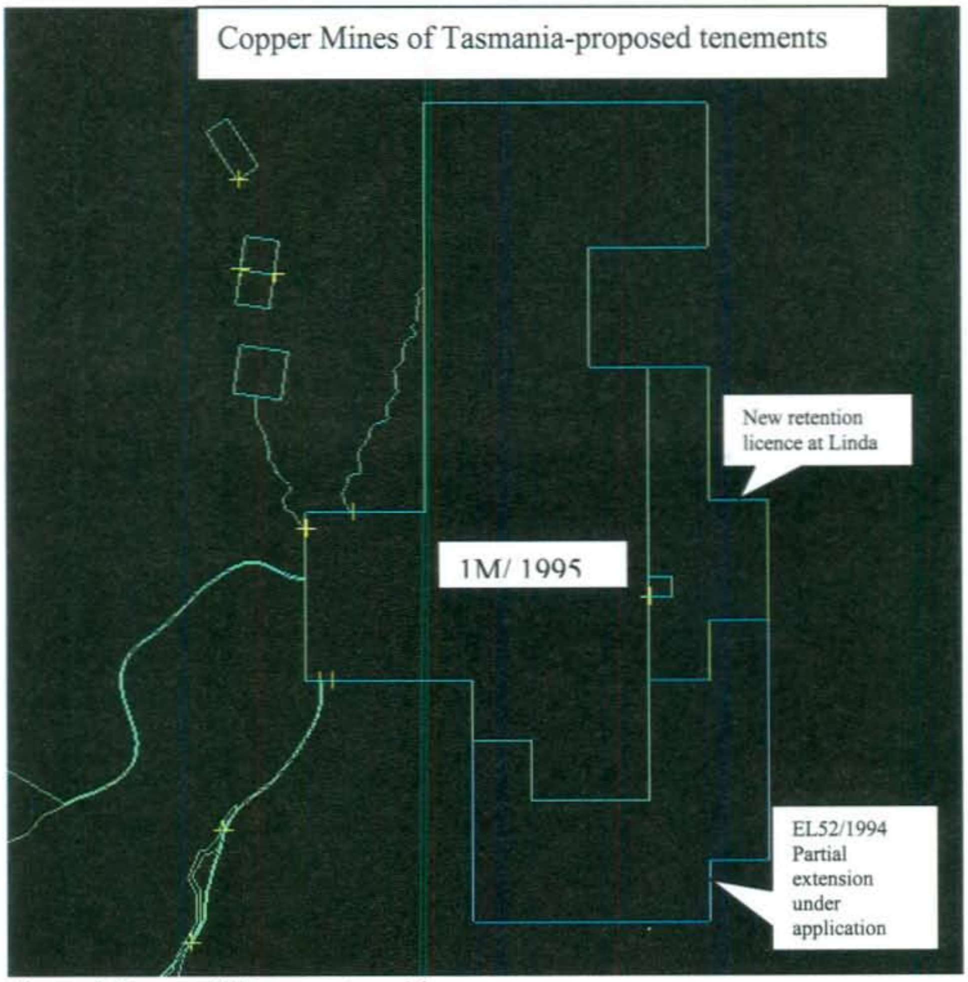


Figure 1 Proposed Tenement position

Retention licence AMG		
point#	East	North
1	383500	5343600
2	384000	5343600
3	384000	5342500
4	384500	5342500
5	384500	5341500
6	384000	5341500
7	384000	5341000
8	383500	5341000

Exploration licence AMG		
point#	East	North
5	384500	5341500
8	383500	5341000
9	384500	5339500
10	384000	5339500
11	384000	5339000
12	382000	5339000
13	382000	5340500
14	382500	5340500
15	382500	5340000
16	383500	5340000

Table 1 Proposed Tenement coordinates AMG AGD66

KING LYELL COPPER CLAYS

RESOURCE ASSESSMENT

MAY 1997

PREPARED BY K.MORRISON & J. KNIGHT

FOR

COPPER MINES OF TASMANIA

SUMMARY

The King Lyell copper clays lie to the east of the King Lyell area, at the head of the Linda Valley. Ten drill holes which intersect the mineralised zone were used in this assessment, together with maps showing the surface outcrop at the western extent, and cross sections from a report by K Wills. K. Morrison defined an overall limit to mineralisation, based on the drill hole data and the field mapping, and this limit has been respected in this assessment.

The mineralised zone is a tongue shaped body, pinching out to surface at the western end, and dipping gently to the east south-east. The body has been interpreted as extending east as far as drill hole M12, just west of the Lyell Highway. The inferred resource is estimated to be 1.2 million tonnes of copper clay with an average grade of 1.37% copper. However, the grades in the western half of the body are consistently higher than in the deeper, eastern half. The average grade in the western 0.6 million tonnes is 2.01% copper.

A conceptual ultimate pit was modelled around the body, assuming an ultimate slope of 60 degrees, to get a feel for stripping ratios. This yielded an overall ratio (tonnes of waste to tonnes of "ore") of 2.3, and 1.3 for the western half only.

The author stresses that this is a very preliminary assessment, based on the field mapping and just ten drill holes. The volumes were calculated by wire-framing from cross sections, with no attempt to model grade distribution at this stage. The grades quoted above are simple arithmetic averages from the drill hole intersections, and the boundaries of mineralisation are based loosely on a minimum grade of 0.2% copper. Therefore the assessment is prepared as a preliminary estimate only, as instructed, and is not claimed to be rigorous.

Comment 2006
still valid as an inferred resource, esp @ high Cu prices.

DATA PREPARATION

Database

All data has been loaded on to a Datamine database. The maps and sections supplied by CMT, from which some data was extracted, are listed in Appendix I.

Topographic data

Topographic data and the locations of creeks and roads were supplied in digital form by CMT as ASCII files

Drill holes

All drill holes have been assumed to be vertical. Collar coordinates were obtained to the nearest half metre from maps supplied by CMT, and elevations were determined to the nearest half metre from the topographic data. Depths were given on the sections prepared by K Wills, and graphical logs in the case of KL1, KL2, and KL13.

Mineralisation Limit

K. Morrison provided an estimate of where the surface geology constrains the extent of mineralisation. This limit has also been included in the database (as a polygon) to control the extent of the model in the cross sections.

The topographic data, drill hole locations, cross section locations, and the mineralisation limit are shown on the base map in Figure 1.

Assays

Average assay values over mineralised intersections were taken from the sections preparaed by K Wills. In the case of KL1, KL2, and KL13, half metre and metre sample values were available, but grades were averaged over the mineralised intersections for consistency with the earlier holes. The intersections are summarised below.

Drill hole	From	To	%Cu
KL2	74.0	77.8	0.86
KL13	42.2	69.4	1.08
KL1	0.0	7.6	1.69
KLC1	45.0	58.0	0.85
KLC2	29.0	38.0	3.11
KL16	36.0	37.5	0.13
ML9	2.0	42.0	2.16
ML10	30.0	37.6	0.44
ML11	47.5	84.0	0.67
ML12	64.0	112.8	0.52

Section location and grids

The topographic data and more recent location maps are based on AMG coordinates. The map showing the location of the cross sections prepared by K. Wills is based on a mine grid. Using the location of drill holes KL1, KL2, and KL13 which are shown in both grids, the location of the cross sections in AMG was determined. AMG coordinates have been used in this evaluation. The section locations are shown on Figure 1.

Topographic profiles

Topographic profiles were obtained for the sections by vertically slicing a wireframe surface (DTM) constructed from the topographic contours, along the sections. There are some minor differences between these profiles and the ones on K. Wills' sections.

Copper Clay Mineralisation

The extent of copper clay mineralisation is shown on Cross Sections 1, 2, and 3 as prepared by K. Wills. These boundaries were digitised into the database as a starting point for modelling the extent of the mineralisation in 3D.

MODELLING

The three cross sections prepared by K.Wills form the basis for the 3D modelling. However, they are too far apart to allow immediate construction of a wireframe to link them. Additional intermediate cross sections were digitised within GUIDE (the graphical side of Datamine) based on limiting the view to include two adjacent original sections, and all drill hole intersections between them. In addition, the location of the overall limit to mineralisation provided by K. Morrison was also displayed, and respected in the construction of the additional cross sections. Topography was also respected where the body outcrops, by using topographic profiles constructed as already described, for all intermediate sections.

The body was extended to the east as far as drill hole M12 which required additional sections beyond those of Wills. The western end of the body was constrained by a small section representing the body just before it pinches out to surface. The Long Section of Wills was used as a guide in modelling the shape of the western and eastern extents of the body.

Once sufficient sections were constructed, they were linked to form a closed wireframe body whose volume can be immediately evaluated.

Grade data is insufficient to allow interpolation of grade distribution. The best (but hardly rigorous !) estimate of overall grade at this stage is simply to average the average grades of the intersections.

The sections corresponding to the locations of K.Wills' sections, as well as the section through ML12 are shown in Figures 2 to 6.

Figure 7 is an isometric view of the polygons defining the mineralised body, together with the drill holes and mineralised intersections.

Figure 8 is a West-East section projection of the wireframe (back side hidden) enclosing the body, together with the drill holes.

CONCEPTUAL PITS

In order to get a preliminary feel for stripping ratio, the body was enclosed in a wireframe representing an open pit based on an ultimate slope of 60 degrees. This pit is only conceptual, and was constructed by digitising (within GUIDE) pit sections around the body on each section, and respecting the topographic sections described above.

Figure 9 shows the pit outlines enclosing the body outlines.

RESULTS

The average grades in the western half of the body are consistently higher than those in the eastern, deeper half of the body. Therefore the results are presented for the case where only the western half of the body is considered, as well as for the whole body as far as drill hole M12.

	Mineralised Zone		Overburden			
	Cubic metres	Tonnes	Cubic metres	Tonnes	S/R	Grade
West only	240,123	612,314	363,469	799,632	1.3	2.01%
Whole body	475,085	1,211,467	1,243,028	2,734,662	2.3	1.37%

Note that the grades have been calculated excluding the intersections from KL16 and ML10 which lie at the very edge of the body as it is now defined (see figures 2 and 3), and whose grades are below the minimum of 0.2% on which the intersections have been (loosely) based. Tonnages were calculated using densities supplied by CMT of 2.55 for mineralised rock, and 2.2 for waste.

APPENDIX I

List of maps and diagrams supplied by CMT

King Lyell Data Compilation Fig. 15 (Dwg No KL1001)

King Lyell Cross Section 1 (6345E) Fig 16 (Dwg No 1002)

King Lyell Cross Section 2 (6410E) Fig 17 (Dwg No 1003)

King Lyell Cross Section 3 (6493E) Fig 18 (Dwg No 1004)

King Lyell Long Section 4 (4658N) Fig 19 (Dwg No 1005)

King Lyell Dholes - Rockchips Cu_ppm 06-Nov-96 (AMG)

King Lyell 18-Aug-96 (AMG)

King Lyell Geology EL52/94 21-Mar-97 (AMG)

King Lyell Geology & Drill Holes EL52/94 21-Mar-97 (AMG)

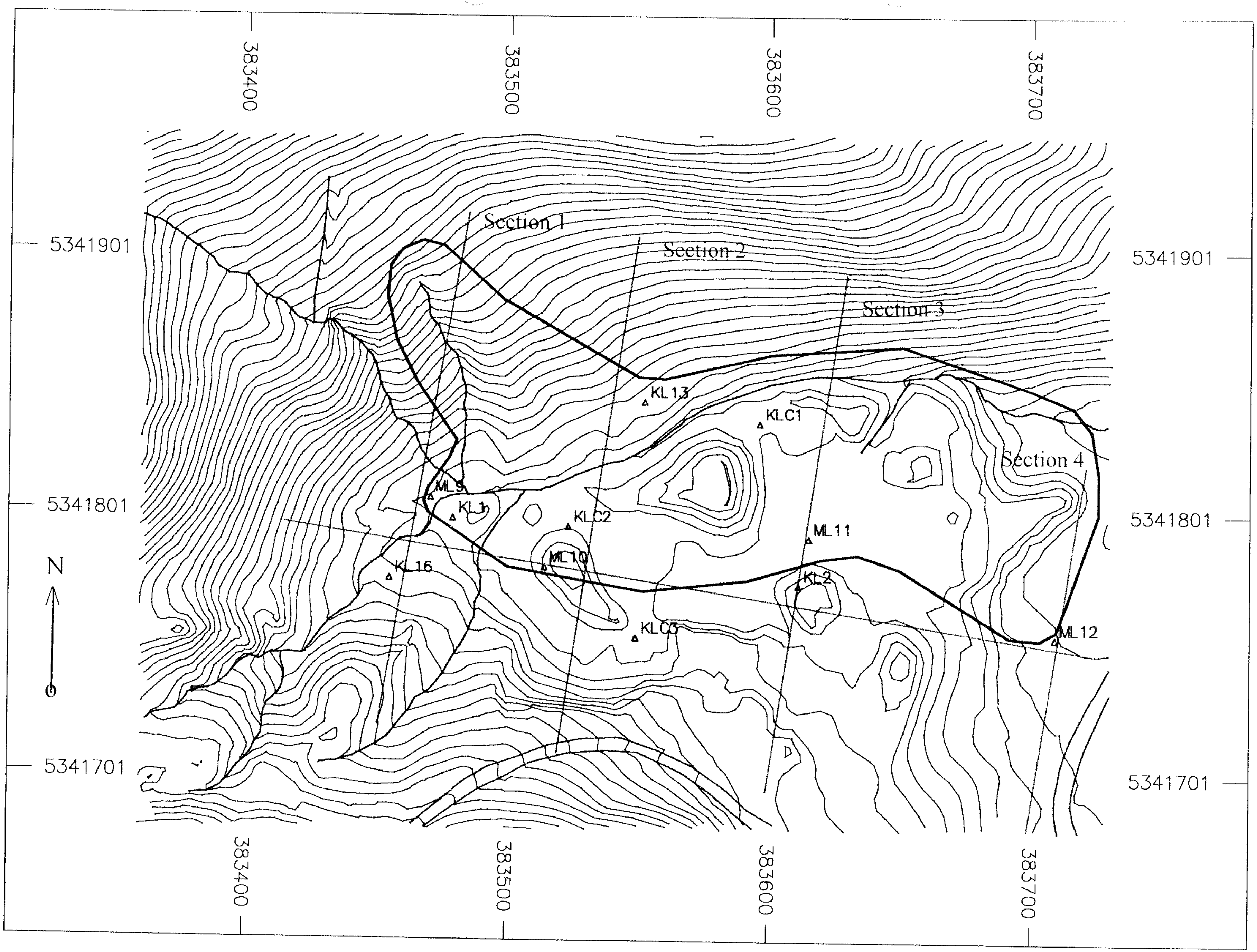


Figure 1. Base map, scale 1:2000, showing topography, creeks, Lyell Highway (red), limit to mineralisation (magenta) drill hole locations (AMG coordinates), and section locations (green)

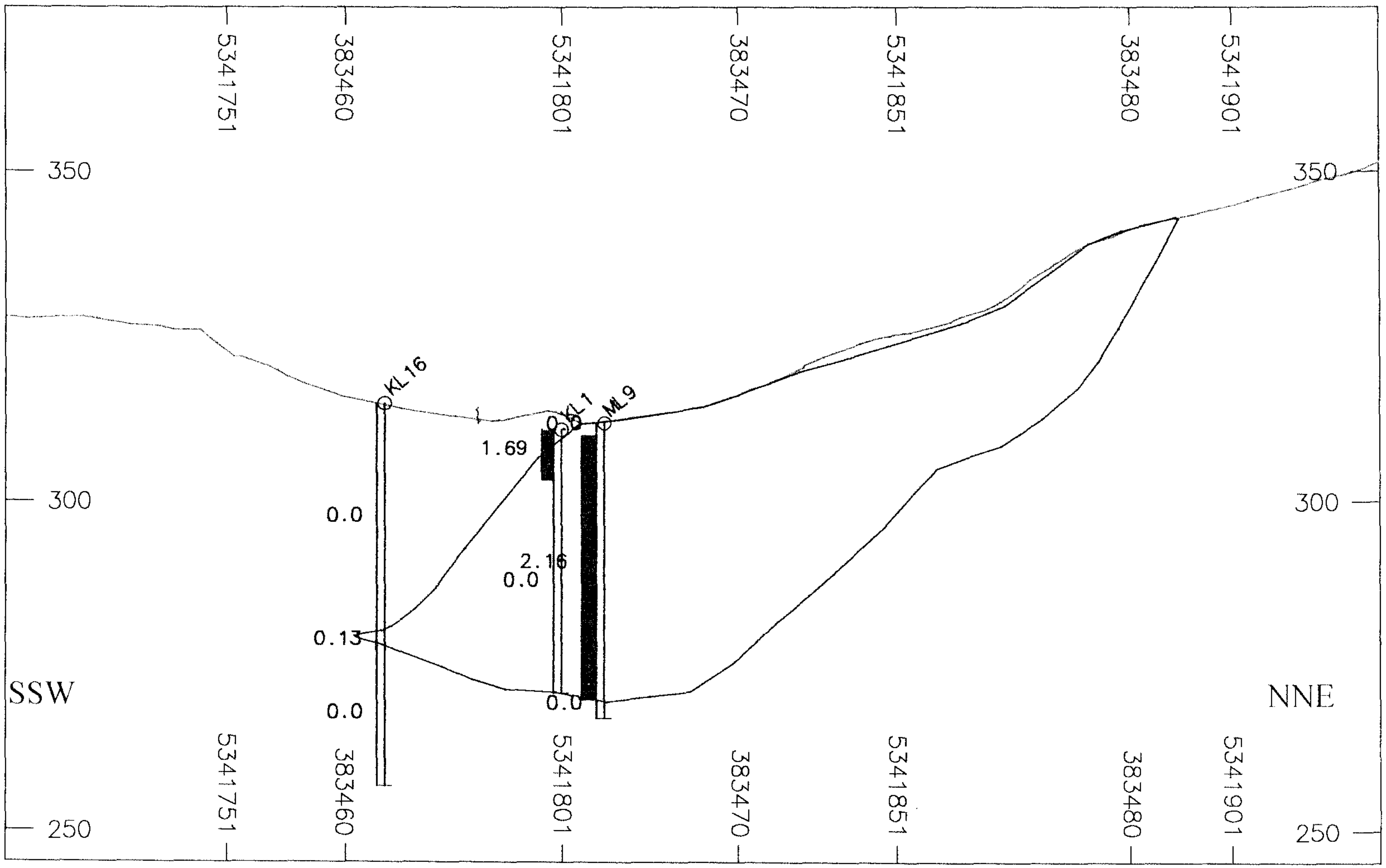


Figure 2. Cross Section 1, showing intersection grades in % Cu.

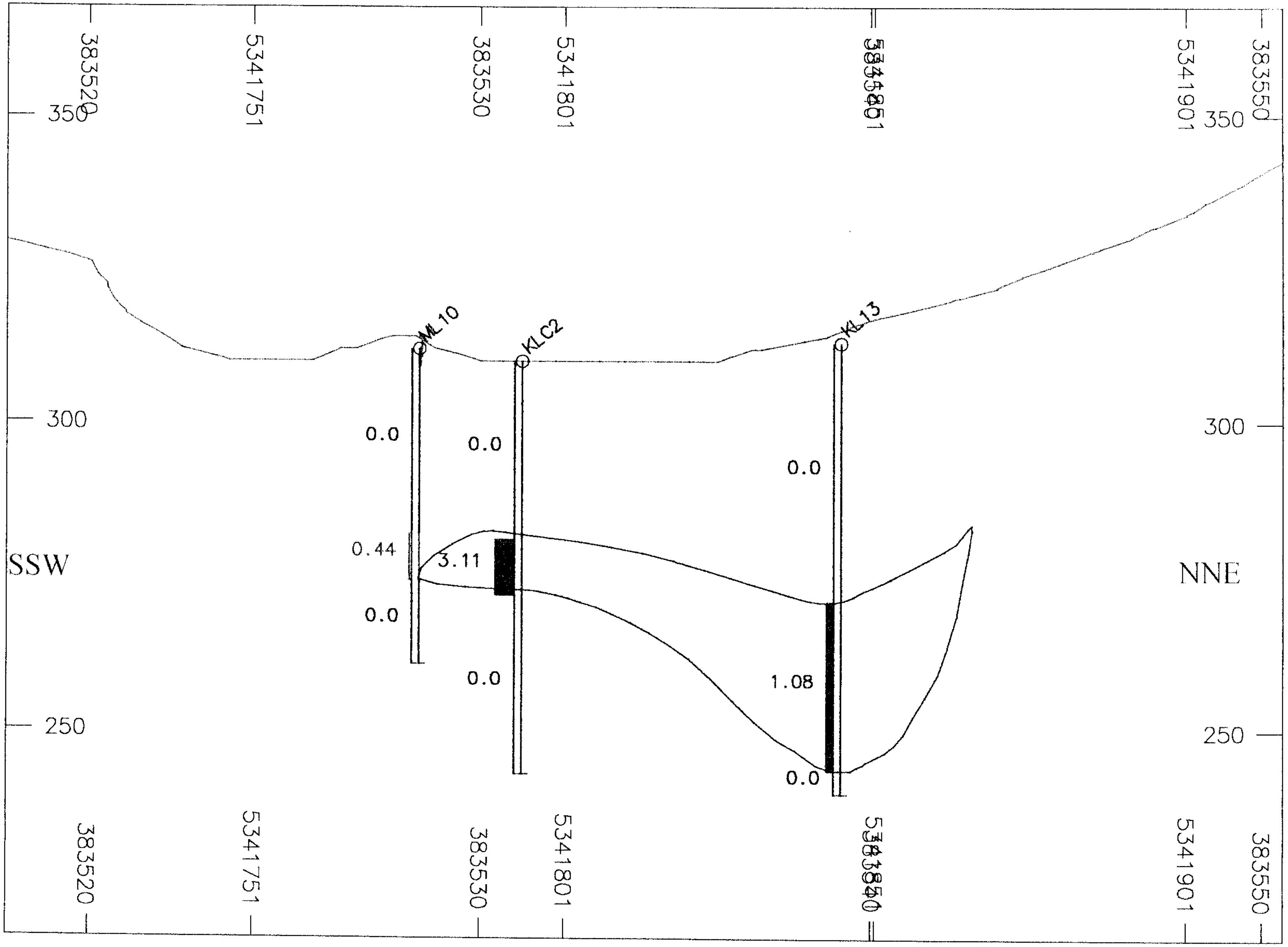


Figure 3. Cross Section 2, showing intersection grades in % Cu.

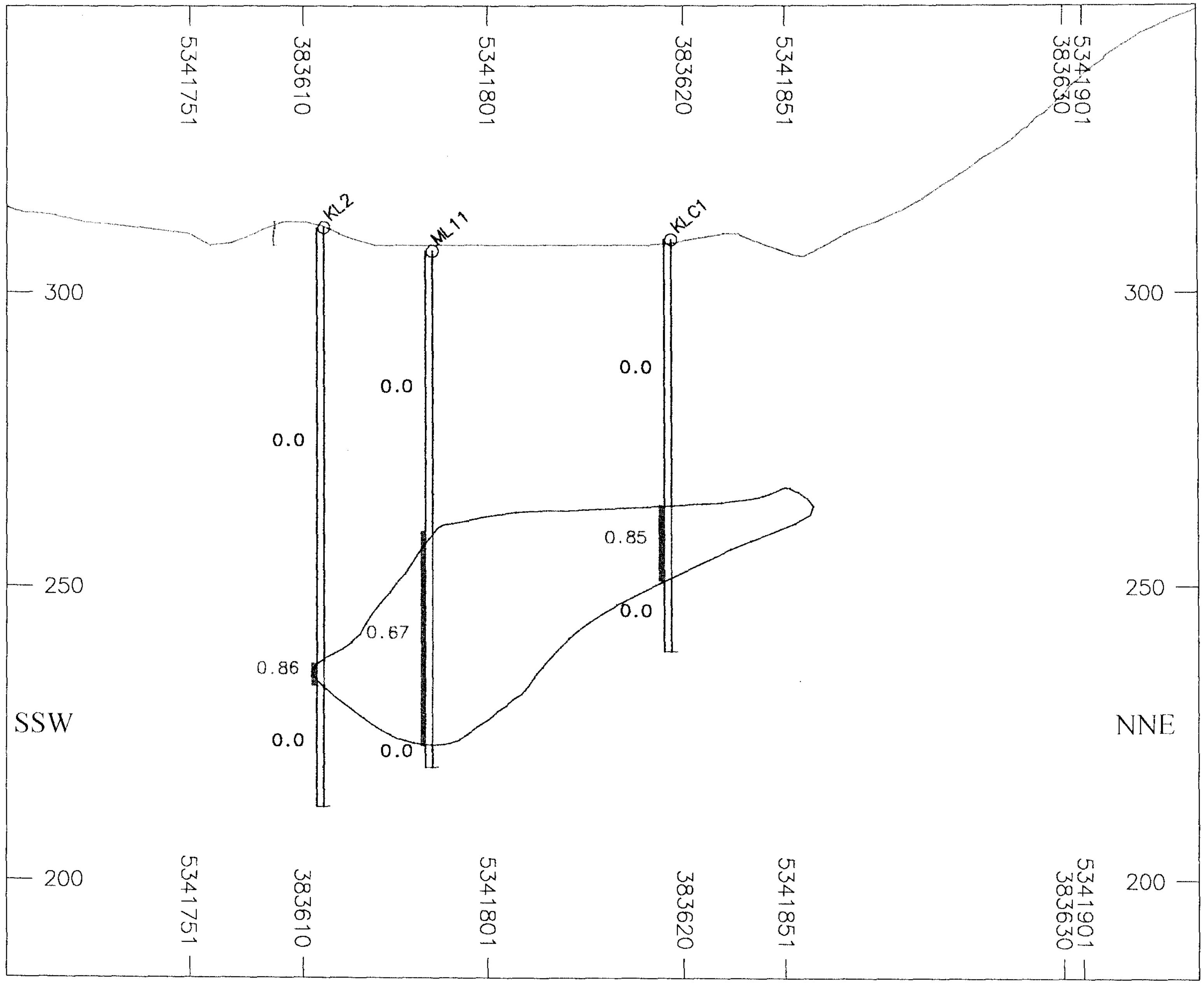


Figure 4. Cross Section 3, showing intersection grades in % Cu.

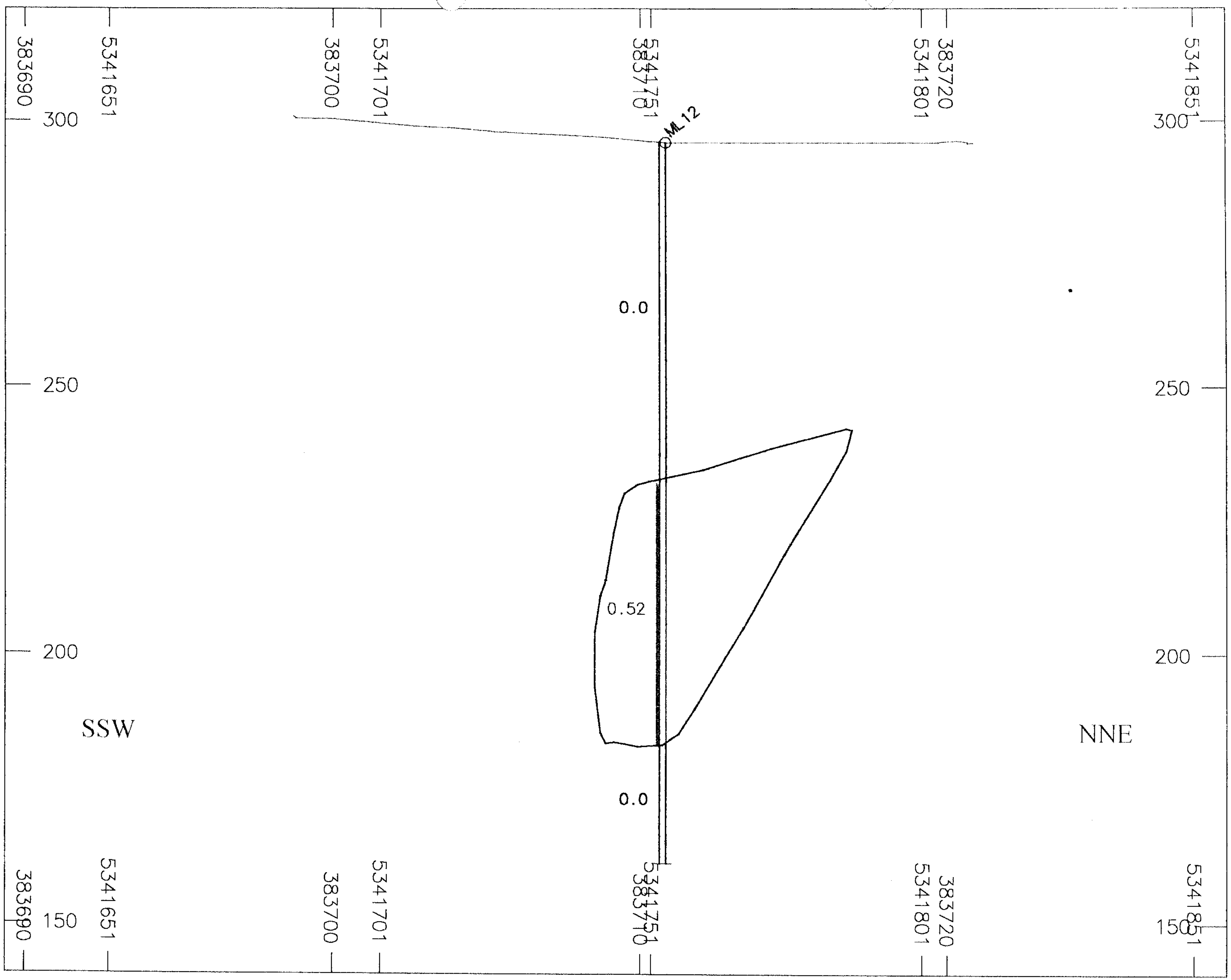


Figure 5. Cross Section 4 through drill hole ML12, showing intersection grades in % Cu.

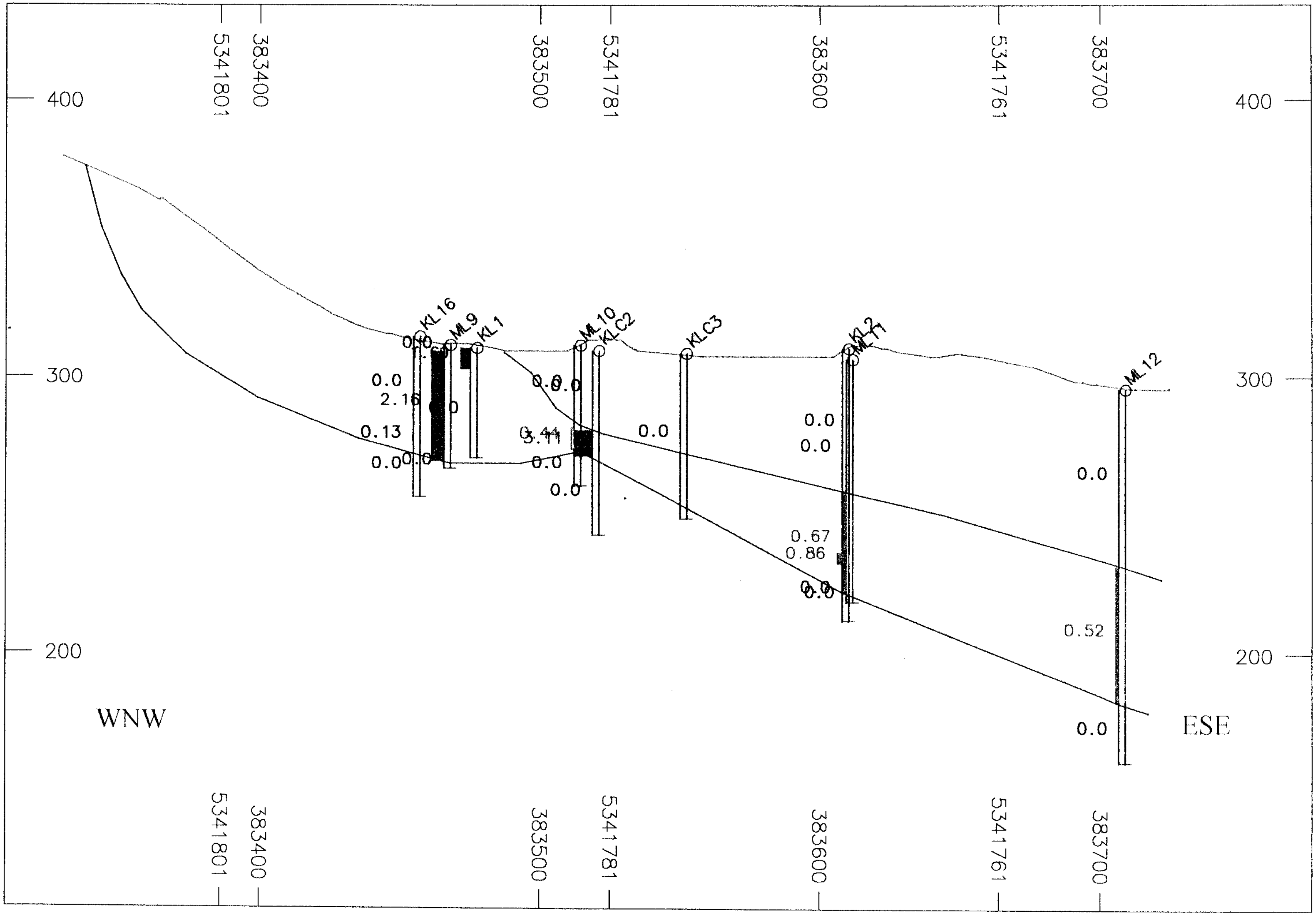


Figure 6. Longitudinal Projection through drill hole ML12, showing intersection grades in % Cu.

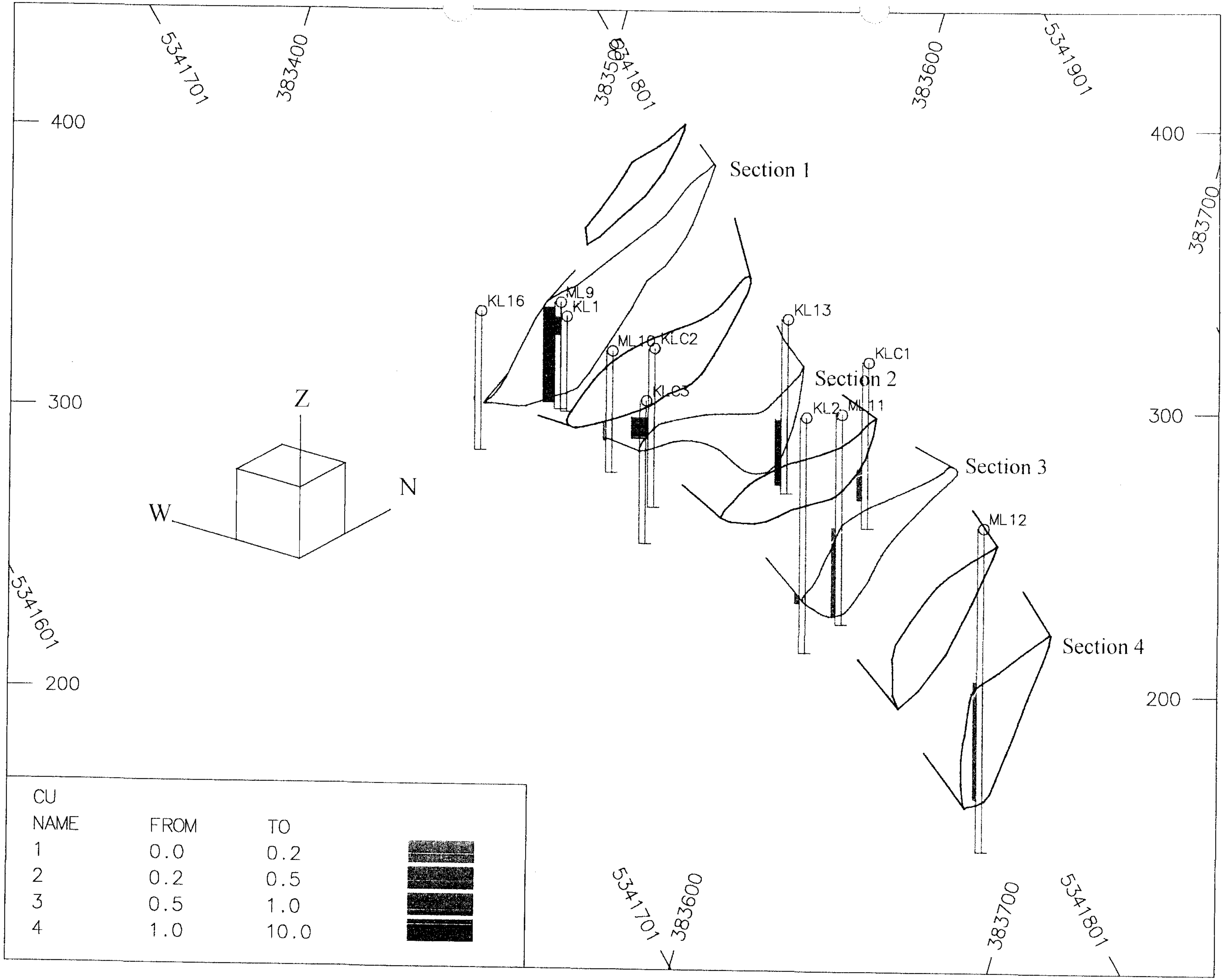
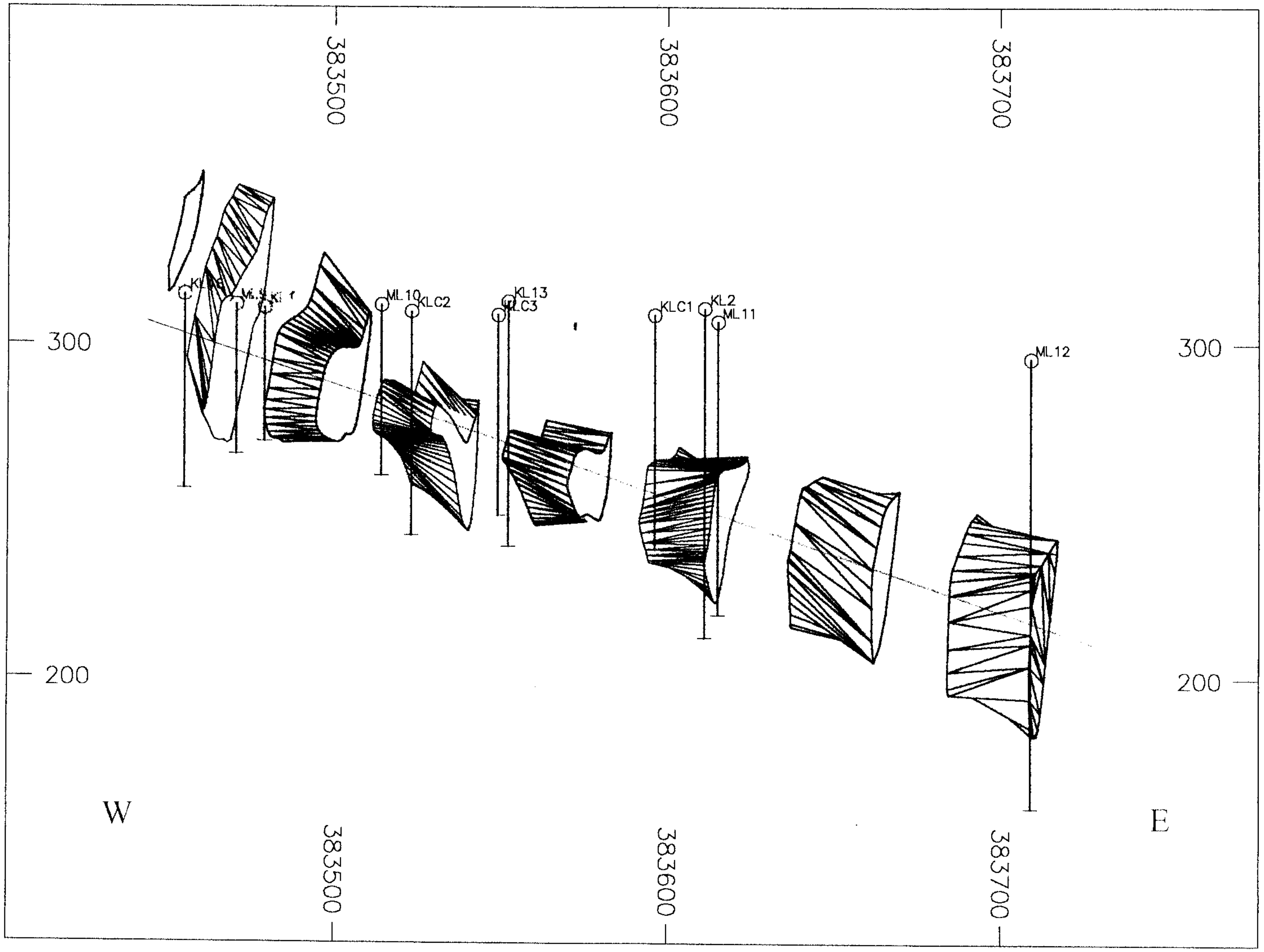


Figure 7. Isometric Projection showing model sections, drill holes and mineralised intersections



~ 15°

Figure 8. West to East Longitudinal Projection, showing wireframe around mineralised body (back side hidden), sections, and drill holes

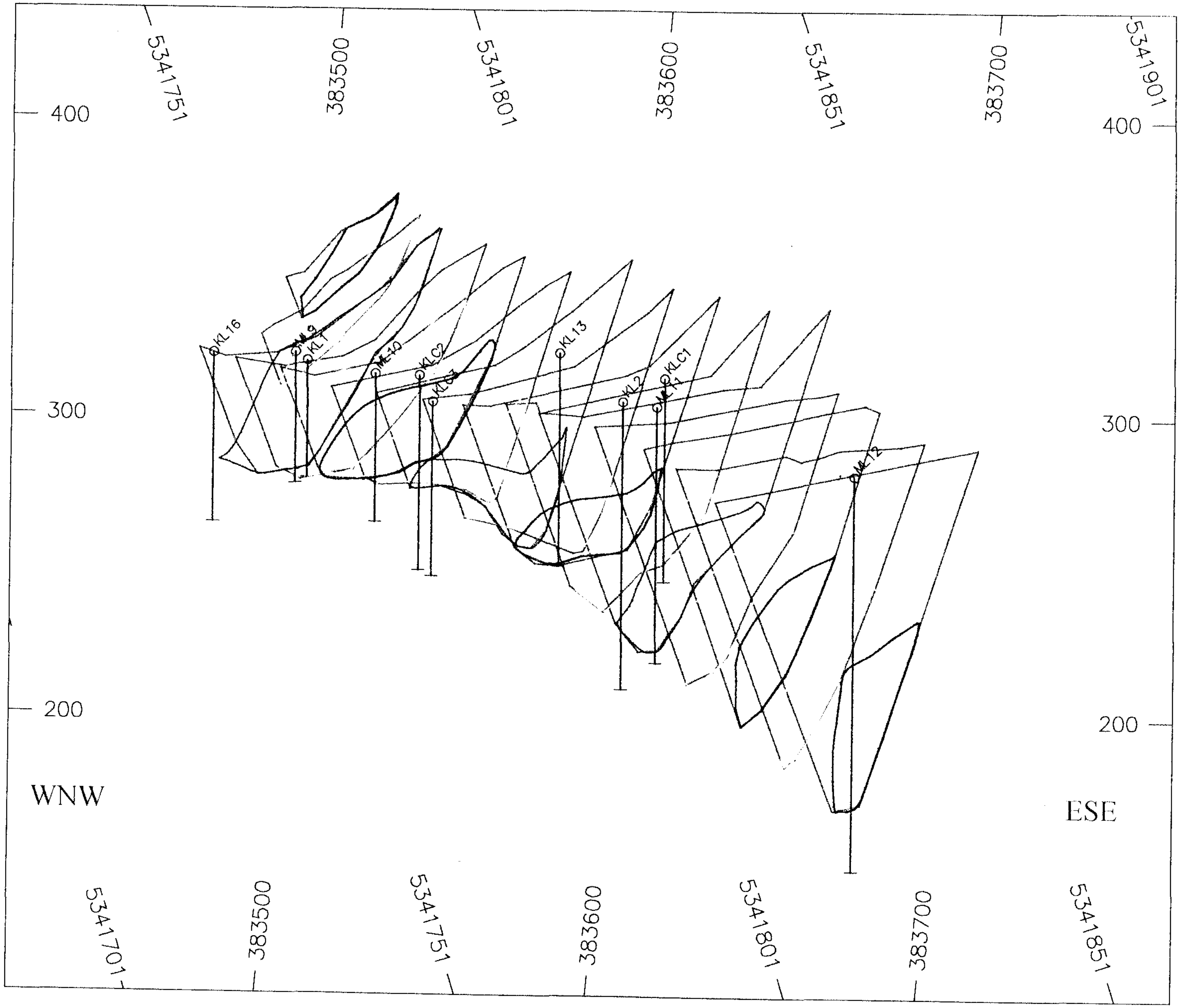


Figure 9. Isometric Projection,showing conceptual pit outlines surrounding mineralised body outlines

	Fwd Cu prices… Assume US$800/yr decline in Cu metal price pers cf -CMT Finance mgr "Arvind"…..June 6'06									
Year	**2006-7**	**2007-8**	**2008-9**	**2009-10**	**2010-11**	**2011-12**	**2011-12**	**2011-13**		
$US Cu	7400	6600	5800	5000	4200	3400	2600	1800		
$US/lb	$ 3.36	$ 2.99	$ 2.63	$ 2.27	$ 1.91	$ 1.54	$ 1.18	$ 0.82		
C:\00TCR\06_5328\retention licence 3-2006\economic assesment (mining)\[KLCC_2006.XLS]2006-7 Fwd Cu prices										

KING LYELL COPPER CLAYS							
250,000 TONNE PER YEAR PRODUCTION							
By:	Tony Weston		…….updated metal prices March 2006 Roger Hill				
Date:	01/03/1997						
File:	g:\mining\engineer\tony\excel\klcc.xls						
C:\00TCR\06_5328\retention licence 3-2006\economic assesment (mining)\[KLCC_2006.XLS]250k							
This is is based on the **existing resource / reserve** at the King Lyell							
Mining Physicals							
Mining - ore	250,000	tpa	7.20				
- 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				**Appraisal method**			
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	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 { Cu Clays Mill Flowsheet May 1997.pdf } 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	adjusted in 2006	Total ($)	Rate ($/t)
Front end loader	(hrs)	7,300	($/hr)	60	438,000	1.75
Operators	(ea)	6	($/year)	100,000	600,000	2.40
Supervisor	(ea)	1	($/year)	120,000	120,000	0.48
Metallurgist	(ea)	0.5	($/year)	120,000	60,000	0.24
On Costs		30%			234,000	0.94
Power					250,000	1.00
Consumables					240,000	0.96
Maintenance Spares					600,000	2.40
Maintenance manpower	(hrs)	4,380	($/hr)	60	262,800	1.05
Flocculant					90,000	0.36
Miscellaneous					100,000	0.40
Management Overhead		10%			150,000	0.60
Total					3,144,800	**12.58**

Realisation Costs

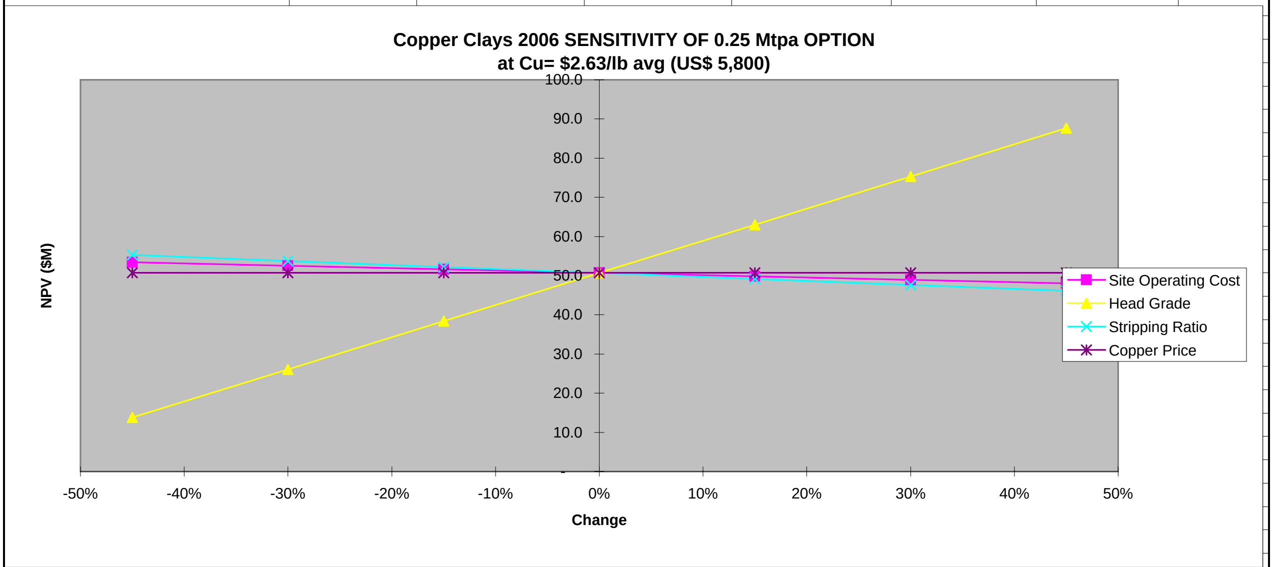
		Activity	Rate ($US)	Rate ($A)
Exchange Rate	0.75			
Haulage of cons to cons shed	(t)	Concentrates	-	2.00
Truck, rail haulage and wharfage	(t)	Concentrates	-	25.60
Shipping	(t)	Concentrates	28.00	37.33
Smelting charges	(t)	Concentrates	90.00	120.00
Refining charge	(lb)	Copper	0.09	0.12

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	$ 12.58	3,144,800	2,654,211				
Haulage of cons to cons shed	$ 2.00	10,111	8,534				
Total Site Operating Cost		6,014,131	5,074,185				
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		7,510,568	6,337,178				
Revenue		Vedanta fwd Copper price assumptions					
		2007-8	**2008-9**	**2009-10**	**2010-11**	**2011-12**	
Copper price	3.36	3.36	2.99	2.63	2.27	1.91	US$/lb
US$/lb	3.36						US$/lb
US$/t	7,405	7,400	6,600	5,800	5,000	4,200	US$/t
Exchange Rate	0.75	0.75	0.75	0.75	0.75	0.75	
Copper Price $A/t		$ 9,870	$ 8,800	$ 7,730	$ 6,670	$ 5,600	A$/t
Years	Rate						
	($/unit)						
Revenue		$ 21,331,538	$ 19,019,000	$ 16,706,463	$ 14,415,538	$ 12,103,000	

Discounted Cash Flow							
Discount Rate		15%					
Year	0	$ 13,820,969					
	1	$ 12,681,822					
	2	$ 10,369,285					
	3	$ 8,078,360					
	4	$ 5,765,822					
NPV		**$ 50,716,258**					
NPV Sensitivities							
Change		-45%	-30%	-15%	0%	15%	30%
		3307772	4209892	5112011	6014131	6916251	7818370
Site Operating Cost ($M)	50.7	53.4	52.5	51.6	50.7	49.8	48.9
		0.8	1.0	1.2	1.4	1.6	1.8
Head Grade ($M)	50.7	13.8	26.1	38.4	50.7	63.0	75.3
		1.9	2.4	3.0	3.5	4.0	4.5
Stripping Ratio ($M)	50.7	55.3	53.8	52.2	50.7	49.2	47.7
		1.85	2.35	2.86	3.36	3.86	4.37
Copper Price ($M)	50.7	50.7	50.7	50.7	50.7	50.7	50.7
	-45%	-30%	-15%	0%	15%	30%	45%
Site Operating Cost	53.4	52.5	51.6	50.7	49.8	48.9	48.0
Head Grade	13.8	26.1	38.4	50.7	63.0	75.3	87.6
Stripping Ratio	55.3	53.8	52.2	50.7	49.2	47.7	46.1
Copper Price	50.7	50.7	50.7	50.7	50.7	50.7	50.7



KING LYELL COPPER CLAYS			
750,000 TONNE PER YEAR PRODUCTION			
By:	Tony Weston	……………originally in 1997	
Date:	01/03/1997	……………updated in 2006 by Roger Hill	
File:	g:\mining\engineer\tony\excel\klcc.xls		
This is is based on the **existing resource/reserve at the King Lyell** only			
5.4Mt containing 1.2Mt ore at 1.4%			
Mining Physicals			
Mining - ore	750,000	tpa	
- waste	2,622,000	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	Increased 50% from Reedy oxide
- hauling	$ 1.78	/t	Increased 50% from Reedy oxide
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 { Cu Clays Mill Flowsheet May 1997.pdf } 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					Total	Rate
	Activity		Rate	adjusted in 2006	($)	($/t)
Front end loader	(hrs)	7,300	($/hr)	60	438,000	0.58
Operators	(ea)	6	($/year)	100,000	600,000	0.80
Supervisor	(ea)	1	($/year)	120,000	120,000	0.16
Metallurgist	(ea)	0.5	($/year)	120,000	60,000	0.08
On Costs		30%			234,000	0.31
Power					250,000	0.33
Consumables					240,000	0.32
Maintenance Spares					600,000	0.80
Maintenance manpower	(hrs)	8,760	($/hr)	60	525,600	0.70
Flocculant					90,000	0.12
Miscellaneous					100,000	0.13
Management Overhead		10%			150,000	0.20
Total					3,407,600	4.54

Realisation Costs		Activity	Rate	Rate
			($US)	($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	Mechanical
			(t)	(t)
Feed Preparation		360	120	200
Nuggets		90	50	60
Fine		320	90	60
Total		770	260	320

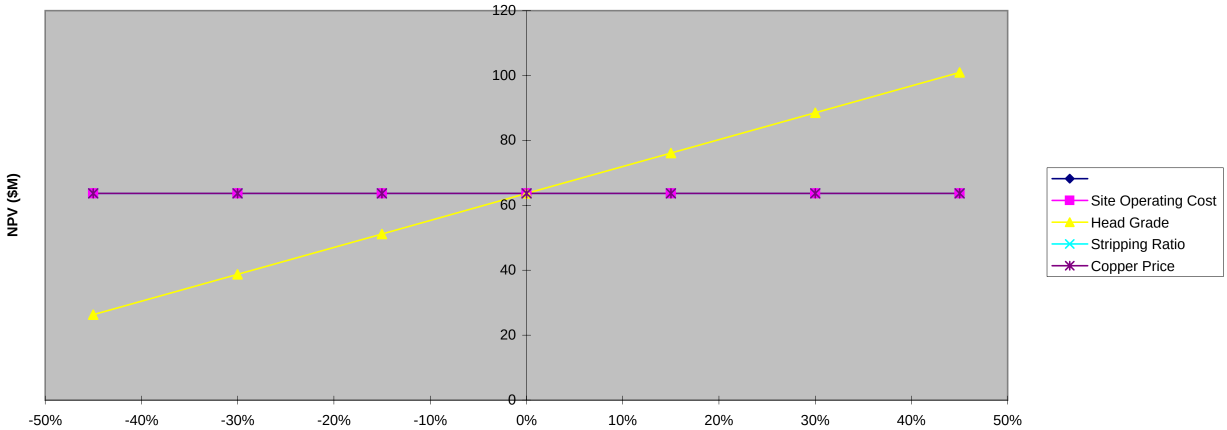
Scale power pro-rata on throughput rate, other items pro-rata to the power of 0.6
as per discussions with Nick Clarke

King Lyell Plant		kW	Steel	Mechanical
			(t)	(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		Annual	Year 1	Year 2
Mining - ore	(t)	750,000	750000	461,000
Mining - waste	(t)	2,622,000	2,622,000	1,610,703
Milling	(t)	750,000	750000	461,000
Grade	(%Cu)	1.4	1.4	1.4
Copper in feed	(t)	10,500	10,500	6,454
Recovery	(%)	65	65	65
Copper in concentrates	(t)	6,825	6,825	4,195
Concentrate grade	(%)	45	45	45
Concentrates	(t)	15,167	15,167	9,322
Smelter copper recovery	(%)	95	95	95
Payable copper	(t)	6,484	6,484	3,985
Payable copper	(lb)	14,290,185	14,290,185	8,783,700
Costs				
Years	Rate	Annual	Year 1	Year 2
	($/t)	($)		
Mining - ore	$ 2.16	1,616,250	1,616,250	993,455
Mining - waste	$ 2.66	6,961,410	6,961,410	4,276,418
Milling	$ 4.54	3,407,600	3,407,600	2,094,538
Haulage of cons to cons shed	$ 2.00	30,333	30,333	18,645
Total Site Operating Cost		12,015,593	12,015,593	7,383,056
Truck and rail	$ 25.60	388,267	388,267	238,655
Shipping	$ 37.33	566,222	566,222	348,038
Smelting charge	$ 120.00	1,820,000	1,820,000	1,118,693
Refining Charge	$ 0.12	1,714,822	1,714,822	1,054,044
Total		16,504,904	16,504,904	10,142,486
Revenue		Vedanta fwd Copper price assumptions		
		2007-8	**2008-9**	
Copper price	3.36	3.36	2.99	US$/lb
US$/lb	3.36			US$/lb
US$/t	7,405	7,400	6,600	US$/t
Exchange Rate	0.75	0.75	0.75	
Copper Price $A/t		$ 9,870	$ 8,800	A$/t
0	0			
	0			
Revenue		$ 47,489,708	$ 16,160,791	
Discounted Cash Flow				
Discount Rate		15%		
Year	0	$ -		
	1	$ 47,489,708		
	2	$ 16,160,791		
Total		**$ 63,650,499**		

NPV Sensitivities								
Change			-45%	-30%	-15%	0%	15%	30%
			6608576	8410915	10213254	12015593	13817932	15620271
Site Operating Cost	($M)	63.7	63.7	63.7	63.7	63.7	63.7	63.7
			0.8	1.0	1.2	1.4	1.6	1.8
Head Grade	($M)	63.7	26.3	38.8	51.2	63.7	76.1	88.6
			1.9	2.4	3.0	3.5	4.0	4.5
Stripping Ratio	($M)	63.7	63.7	63.7	63.7	63.7	63.7	63.7
			1.85	2.35	2.86	3.36	3.86	4.37
Copper Price	($M)	63.7	63.7	63.7	63.7	63.7	63.7	63.7
		-45%	-30%	-15%	0%	15%	30%	45%
Site Operating Cost		63.7	63.7	63.7	63.7	63.7	63.7	63.7
Head Grade		26.3	38.8	51.2	63.7	76.1	88.6	101.0
Stripping Ratio		63.7	63.7	63.7	63.7	63.7	63.7	63.7
Copper Price		63.7	63.7	63.7	63.7	63.7	63.7	63.7

SENSITIVITY OF 0.75 Mtpa OPTION
at Cu= $3.00/lb (US$ 6,600)



Change Note that due to short duration of mining, exposure to decreasing Cu price is minimised.

CMT Mt Lyell

Cu CLAY MINERALOGY

SEPTEMBER 2005

McArthur Ore Deposit Assessments Pty Ltd

Gary J McArthur PhD FAusIMM MMICA MSEG

MODA

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COPPER MINES OF TASMANIA

Cu Clay Mineralogy
September 2005

Method

6 composite samples from drillhole CMPKL02 were submitted by Roger Hill (CMT) for mineralogical assessment of Cu mineralogy. Each sample, representing 4m downhole, consisted of three size fractions: coarse +38μm, medium CS1+CS2 and fine CS3+CS4+CS5. These were mounted as polished thin sections by Australian Petrographics in Queanbeyan, N.S.W.

A revised version of the standard MODA technique for quantitative mineralogy was adopted.

For each of the 3 size fractions, 100 grains containing any Cu mineral were selected at random and the area % of each mineral present was visually estimated. The minerals logged were: pyrite FeS_2 (Py), chalcopyrite $CuFeS_2$ (Cp), bornite Cu_5FeS_4 (Bn), chalcocite+digenite $Cu_{2-x}S$ (Cc), covellite CuS (Cv), cuprite Cu_2O (Ct), native copper Cu (Cu) and gangue (i.e. all other minerasls) (Ga).

Due to the limited number of actual grains mounted on the slide for the coarse fractions, 100 grains containing Cu could not be found. Actual grains logged for these fractions varied from 11-38.

The last 3 samples (the deepest in the drillhole) were found to be extremely low grade (0.01-0.05%Cu) and after discussions with Roger Hill, no mineralogical assessment of these samples was undertaken.

A tally was made from the summed area percentages for each mineral and these were converted to weight percentages using theoretical mineral densities. These were in turn converted to percentage of Cu metal using the following theoretical compositions. Fractions were weighted for the totals according to the product of size fraction weight % and the assayed Cu content.

Mineral	Py	Cp	Bn	Cc	Cv	Ct	Cu	Ga
Density	5.01	4.20	5.15	5.65	4.60	6.10	8.94	2.70
%Cu	0	35.9	63.3	79.9	66.5	88.8	100.0	0

Results

Cu Residence

The following tables summarise the mineral residence of Cu for each size fraction of each sample.

Sample	Fraction	Percentage of Cu metal residing in					
		Chalcopyrite	Bornite	Chalcocite	Covellite	Cuprite	Native Copper
CMPKL02 43-47m 0.2%Cu	Coarse	5.9	0.0	22.7	11.4	0.0	60.0
	Medium	0.3	0.0	1.4	0.1	4.4	93.9
	Fine	9.2	0.1	7.2	1.4	5.7	76.4
	TOTAL	**1.4**	**0.02**	**2.4**	**0.4**	**4.5**	**91.4**

Sample	Fraction	Percentage of Cu metal residing in					
		Chalcopyrite	Bornite	Chalcocite	Covellite	Cuprite	Native Copper
CMPKL02 47-51m 1.3%Cu	Coarse	0.1	0.0	0.0	0.0	2.9	97.1
	Medium	1.5	0.0	12.5	0.0	11.7	74.3
	Fine	1.4	0.0	89.1	0.5	4.6	4.4
	TOTAL	**0.5**	**0.0**	**10.3**	**0.04**	**4.7**	**84.4**

Sample	Fraction	Percentage of Cu metal residing in					
		Chalcopyrite	Bornite	Chalcocite	Covellite	Cuprite	Native Copper
CMPKL02 51-55m 1.6%Cu	Coarse	0.0	0.0	100.0	0.0	0.0	0.0
	Medium	9.0	0.0	26.1	0.0	64.8	0.0
	Fine	1.5	0.0	83.4	0.1	15.0	0.0
	TOTAL	**0.6**	**0.0**	**94.7**	**0.02**	**4.7**	**0.0**

Sample	Fraction	Percentage of Cu metal residing in					
		Chalcopyrite	Bornite	Chalcocite	Covellite	Cuprite	Native Copper
CMPKL02	Coarse	**0.3**	**0.0**	**53.6**	**0.5**	**1.2**	**44.4**
43-55m	Medium	**5.4**	**0.0**	**19.2**	**0.0**	**39.3**	**36.1**
1.0%Cu	Fine	**1.8**	**0.0**	**82.4**	**0.3**	**10.1**	**5.4**
TOTAL	TOTAL	**0.6**	**0.0**	**50.5**	**0.1**	**4.7**	**44.1**

Mineral Associations

- Chalcopyrite – mostly liberated, but sometimes with pyrite
- Bornite – quite rare, but normally included in chalcocite
- Chalcocite - predominantly liberated, but occasionally hosting pyrite or covellite
- Covellite – quite rare, but always hosted by chalcocite
- Cuprite – mainly rimming native copper, but also commonly liberated, especially in the 51-55m sample
- Native copper – often liberated, but usually with narrow cuprite rims. In the coarse fractions it often occurs as small isolated grains hosted by gangue

Miscellaneous Comments

Other minerals observed during this assessment were (in order of predominance):

- Quartz
- Ferruginous clays
- Sericite/clay
- Hematite
- Carbonate
- Sphalerite
- Magnetite
- Goethite
- Galena

Oxidised fragments of grinding media were also observed.

G.J.McArthur PhD FAusIMM MMICA MSEG - Principal Geologist
21.9.05

24/3/97

Memorandum

DATE:	March 24, 1997
TO:	Peter Benjamin
FROM:	Nick Clarke
RE:	Copper Clay Metallurgy - Preliminary Thoughts
CC:	Peter Williams, Shane Polle, Ken Morrison

1. INTRODUCTION

Some work has been completed on a sample of copper clays from an outcrop at King Lyell. The work comprises wet and dry sieve sizing and cyclosizing, assay of size fractions and inspection of some size fractions under the stereo microscope. The report by Kevin Wills has also been read, and the MLMRC flotation results reported there have been summarised. This memorandum records the results of work to date, and some resulting preliminary thoughts.

2. SUMMARY

Copper concentrations in the copper clay sample analysed are highest in the coarsest fraction, and decline with decreasing size. A considerable amount of the coarse fractions consisted of agglomerated clay balls. Presumably if these were broken up, the grade of the coarser fractions would increase.

The copper appeared to be present as minerals tentatively identified as cuprite, malachite, chrysocolla, chlacocite and possibly chalcopyrite. From comments by Ken Morrison, it is assumed the cuprite was formed by oxidation of native copper. No native copper was observed. The cuprite appeared quite crystalline.

The copper minerals broke down quite readily, to particles in the range 10 - 30 microns, and possibly finer. It seems likely from this and K. Wills theories of formation that all the copper is present as discrete copper mineral, but it is possible some is adsorbed onto the clays.

Historical and current information suggests that recoveries on ore of treatable grade and type would be in the range 60 - 70% could be achieved by either gravity or flotation separation. In practice, it is likely both head grades and recoveries would be highly variable. A process of desliming and conventional gravity separation might achieve recoveries in the 60% range, based on the sample analysed. Use of centrifugal separators could improve recovery and might avoid the need for desliming. Leaching might achieve higher recoveries, and could generate saleable copper metal on site. However, difficulties with liquid-solid separation and capital cost may militate against leaching.

The viability of an operation to recover copper from the copper clays is somewhat doubtful. The grade is very low for the available tonnage. Capital costs would be substantial, since even a simple gravity recovery plant would require proper means of tailings disposal. Possibly, a plant could be established by mining the King Lyell first, and following with the Blocks.

A small amount of metallurgical testwork is proposed. It is suggested this be followed by a desk-top scoping study, to determine if an operation could potentially be feasible, and what direction further work should take.

3. RESULTS

3.1 Assays

Table 1 records the results of the size analysis and assays on size fractions. Table 2 gives copper distributions by size.

It is apparent there is rather poor agreement between calculated and measured assays. The measured head assay was 2.70% Cu, whereas the assay calculated from the size fractions was 2.44% Cu. The measured assay of the -38 micron fraction was 1.24% Cu, whereas the calculated assay from cyclosizer fractions was 0.80% Cu. Some very coarse lumps resulting from the size analysis are not included in the assays shown. The lumps were assayed later, and ran (from memory) about 10% Cu. This would bring the calculated assay up to better agreement with measured. The poor agreement between measured and calculated assays on the -38 micron fraction could be due to loss of - Cone 5 material, although much of it was recovered. If this is the cause of the difference, then the lost material would have been very low grade.

Inspection of Table 2 shows that grades are highest in the coarsest fraction, and decline steadily with size.

3.2 Mineralogy

The +106 micron fraction was selected to examine under the stereomicroscope. All grains appeared a uniform pale brown. Weak acid was applied and it was found that most of the lumps consisted of agglomerated fines. This size fraction was rewashed on a 106 micron sieve, with gentle brushing to break down the "clay" lumps, and 84% of the weight passed through the sieve. The remaining oversize material contained particles tentatively identified as chrysocolla, malachite and a dark red brown substance likely to be cuprite. Around 10% of the mass was transparent silicates (quartz?) and 20% clay lumps remained.

The material washed through the sieve contained a lot of clay material plus ?cuprite and ?malachite in all sizes down to 20 microns and finer.

The +38 micron fraction was inspected, and contained relatively little clay lump. The major species present was ?quartz, with a substantial amount of ?cuprite and rather less ?malachite. There was visibly less copper mineral present than in the +106 micron fraction.

The -38 micron dry sieve fraction was observed, and appeared to consist dominantly of ?quartz, with 20- 30% clay lump, and trace ?cuprite and rare ?malachite.

The -38 micron wet fraction was inspected. By dispersing in water, it was possible to observe minor cuprite, but the sample appeared to consist dominantly of clay lump.

No recognisable native copper was seen in any fractions. Presumably, from Ken Morrison's observations, all native copper had converted to cuprite. Since native copper can certainly be stable in air, there is presumably some reason why the native copper in the copper clays is so reactive.

Very rare grey black and very friable mineral was seen, possibly chalcocite. Rare fresh and sharp edged sulphide was also seen, and it was thought could have arisen from contamination in screening, but could also have come from the copper clays.

It seems quite likely from the assays and observation that most, possibly all, of the copper is present as discrete copper mineral and tends to be fairly coarse. However, because of the generally soft nature of the minerals, they are readily broken down to finer sizes. It is possible some copper is absorbed onto clay mineral.

3.3 Flotation

Table 3 summarises the results of flotation tests conducted by MLMRC on a number of copper clay samples. Unfortunately only one test records the reagents used, which were xanthate, a dithiophosphate and Aero 404, which is recommended for tarnished and secondary copper minerals.

One test tried sulphidisation on a mids product without success - it is assumed from the mention of this that no other tests used sulphidisation.

Results of flotation appear to have been quite good. Three out of the four samples had about 80% of the copper in the "sand" fraction, in two cases in about 25% of the weight. Flotation recovery of the copper in this fraction was about 90%. Flotation recovery overall varied from 31% to 77%. The lowest recovery was from a sample with head grade 0.13% Cu, but one high grade sample (head grade 1.8%) also gave poor overall recovery because 54% of the copper reported to the slimes. However, nothing is known of how the sand/slime separation was done, or what the approximate split size was.

It appears from these results that a considerable variation in metallurgical response to physical separation procedures could be expected and would be driven mainly by the size distribution of the copper.

It is interesting that the recovery from Lyell Blocks ore by gravity methods was quoted at 72%.

4. PROCESS OPTIONS

A number of approaches could be envisaged for recovering copper:

a) Deslime (10 microns?) and float the oversize, potentially in the existing plant, probably more successfully with a tailored reagent regime. Maximum recovery about 70%.

b) Deslime (38 microns) and recover copper from oversize on say spirals. Maximum recovery about 60%.

c) Recover copper using a centrifugal concentrator, with or without prior desliming. Possible machines are a continuous Falcon or Knelson concentrator, Kelsey jig or Mosley Multi-gravity concentrator. The latter two are quite expensive, but cannot be ruled out at this stage. Maximum recovery unknown, but probably about 70% on the ore sample examined.

d) For the above three approaches, very coarse material might go directly to concentrate, and the concentrate could probably be mixed with existing sulphide flotation concentrate for sale and treatment.

e) Recover the copper by leaching. Acid leaching is the simplest approach, but given the presence of native copper and/or cuprite, will probably be unsuccessful. Ammonia/ammonium carbonate leaching would probably be technically successful - in fact almost ideally suited to this ore. Leaching followed by solvent extraction and electrowinning would have the benefit of producing copper on site. The biggest difficulty I think would be liquid-solid separation - desliming to eliminate clay fines would probably be necessary. Other problems would be capital cost and ammonia recovery costs. Recovery could be >90% if the clay fraction is leached, but more likely about 70% with desliming and direct discard of the clay.

I have little doubt from data available now that most or all of the above methods would be technically feasible. The highest recovery would likely be from leaching, and the lowest from desliming and spirals. Cost would probably be in the same order as recovery. The big question is whether any of these techniques could be economically viable. King Lyell would obviously be the best prospect on the basis of grade, but with very little tonnage available to justify an operation.

5. VIABILITY

The combination of low grade and low tonnage for the copper clay resource means that it is likely to be difficult to mine and treat economically. Capital and operating costs for a small plant would be high, and the best chance would probably be if enough tonnes could be proved up for say an operation treating 1 million tpa over 5 years.

On the downside, capital cost would be significant, even for a simple deslime and spiral or float type operation. The simplest flowsheet would involve perhaps a log washer or agitator to disperse the clays in water, a screen to remove coarse material and trash, spiral concentrators for recovery, possibly a thickener and either a tailings dam or a pipeline to the existing tailings dam. Addition of flocculant and possibly a clarification agent would likely be necessary to produce a clear overflow for discharge to the environment or re-use in the plant.

Taking account of the cost of tailings disposal, the cost of $1.50/t for copper recovery seems too low, for a small scale operation. For a resource of say 600,000 t, and a throughput of say 200,000 tpa, 1 shift operator represents a cost of $1/t ore.

The cost of copper transport and recovery will depend very much on concentrate grade if the copper is sold as a concentrate. However, it is important to note that the cost of realisation depends on the amount of copper in the ore, so low grade ore has low realisation costs.

On the upside, perhaps mining costs could be reduced below those quoted by Kevin Wills. Would it be possible to recover clay using a floating dredge/pump and so perhaps reduce the stripping ratio?

There is some possibility that grades in drill holes could be underestimates, given the relatively coarse size of the copper. The sludge assays quoted by Kevin suggest that lost material might be higher in grade. If coarse and dense copper occurs in a clay matrix, is the copper more or less likely to be recovered in drilling than the clay?

However, it seems unlikely that actual grades would be much higher than those estimated from drill holes, since the Lyell Blocks operation was presumably trying to follow high grade material, and head grades declined from 4.3% at the start of operations to 1.5% by the end. Still, 1.5% is substantially higher than 0.5% Cu.

For the Blocks, there also seems to be a reasonable chance of some gold and silver which would be recovered in a gravity or flotation process - the amount seems highly variable. If we assume 0.2 g/t Au and 3 g/t silver, that would add about $4/t to the ore value.

6. RECOMMENDATIONS

6.1 Metallurgical Testwork

I suggest the following approach:

1) Re-screen both surface exposure and drill samples by wet sieving with gentle brushing to break up clay lump. Brushing should be continued until the water runs clear. To avoid damaging the sieve, this could be done on say a 75 micron sieve, then the undersize poured through a 38 micron sieve. The +38 micron fraction should be dry screened on 600, 300, 150, 75 and 38 micron screens. Assay a portion of the -38 micron combined dry and wet fraction.

The combined -38 dry + wet fraction of undersize should be carefully dispersed in water in a 1 litre beaker, with say 100 mm of water depth. Disperse avoiding swirling, then allow to settle for 120 minutes (+- 10 minutes OK). Decant off the suspended solids, but be careful to avoid disturbing the settled solids. Leave about 1 cm of water in the beaker. Make up to 10 mm water depth, and repeat. Repeat again. This should eliminate most material finer than around 4 microns. Cyclosize, and collect the first dustbin full of cyclosizer - cone 5 product, then discard the remainder of the - cone 5. Cones 1-3 and 4 & 5 may be combined. Assay all sieve fractions, cone 1-3, cone 4-5, -cone 5 cyclosizer and beaker decantation fines separately, for copper only. Also make up samples of the combined sieve size fractions, cones 1-5 and -5 combined, and beaker decant solids and assay for gold and silver.

2) Prepare a sample of surface exposure and drill "core" material for gravity separation , by the same procedure as used for preparing the cyclosizer feed sample. Send settled and decanted solids to Central Mineralogical Services for Superpanning and microscopic examination.

3) Leach a sample of unsized surface ore in sulphuric acid. Record the acid consumption and measure the copper extraction.

4) Leach a sample of unsized surface ore in acid drainage liquor (say from sample point H3 & H4). If necessary, supplement the AD with sulphuric acid.

5) Leach a sample of unsized surface ore in ammonia/ammonium carbonate under atmospheric conditions.

6) Deslime about 2 kg of surface exposure ore by decanting from a bucket (allow 10 minutes settling time per cm of water depth, and repeat the decantation operation 3 times). Assay and discard the suspended solids. Carry out a flotation test on the settled solids. It may be necessary to obtain a suitable reagent from Cytec, depending on what we have available.

The above testwork will indicate what methods of concentration could be viable, and what recovery might be expected.

6.2 Scoping Study

Before proceeding further, I suggest a desktop scoping study should be carried out, to see whether a project could potentially be feasible.

At this stage I would suggest that for the project to have a chance, and to have a worthwhile impact on cash flow, it would be necessary for the copper clays in the Blocks at least to be economic to treat, so that a reasonable plant capacity could be established based on mining King Lyell first, then the Blocks.

6.3 Drill Core Loss

Would it be possible to establish the effect of core loss on drill grade by drilling a short hole in accessible copper clays, then confirming grade from a bulk sample?

Nick Clarke

TABLE 1: Size Analysis

Size Analysis on:	Cu Clay Sample 1 - King Lyell Outcrop
Sampled:	

Cyclo Cone	Sieve Size Microns	Wgt g	Cu %	Zn %	Pb %	Wgt Held %	Cum Wgt Pass %
	850					0	100.00
	600	5.09	5.80	0.50	0.017	2.56	97.44
	425	9.27	5.80	0.50	0.017	4.67	92.77
	300	10.17	5.80	0.50	0.017	5.12	87.65
	212	5.79	5.80	0.50	0.017	2.91	84.74
	150	0.00	0.00	0.00	0.000	0.00	84.74
	106	8.58	6.35	0.48	0.015	4.32	80.42
	75	0.00	0.00	0.00	0.000	0.00	80.42
	53	8.58	6.85	0.43	0.011	4.32	76.10
	38	3.63	3.65	0.39	0.011	1.83	74.27
	-38	147.52	1.24	0.36	0.016	74.27	-
Cone 1	45.2	-	1.23	0.26	0.010	0.44	73.83
Cone 2	33.1	-	1.23	0.26	0.010	1.01	72.82
Cone 3	23.7	-	1.23	0.26	0.010	4.31	68.51
Cone 4	17.2	-	0.67	0.22	0.007	6.24	62.27
Cone 5	12.3	-	0.67	0.22	0.007	6.20	56.08
- Cone 5	-12.3	-	0.79	0.42	0.019	56.08	-
Calc -38			0.80	0.37			
Total Calculated		198.63	2.44	0.39	0.016	100.00	-
Total Measured		200	2.70	0.40	0.015		

Wet Sieve Weights	
Head g	200
+38W g	55.27
-38W g	143.56
-38D g	3.96
Checks:	
-38 Dry	3.96
Total	
Dry Calc	55.07
Dry Meas	55.27
+38 Wet	55.27
-38 Wet	143.56
Total	
Wet Calc	198.83
Wet Meas	200.00

Cyclosizer Data		
	Wgt g	Wgt %
Cone 1	0.31	0.590
Cone 2	0.72	1.358
Cone 3	3.08	5.807
Cone 4	4.45	8.400
Cone 5	4.42	8.342
-Cone 5	-	75.504
Head	52.98	100.000

Particle S.G.	2.7
Elutriation Temperature	11
Elutriation Flowrate	180
Elutriation Time, mins	30

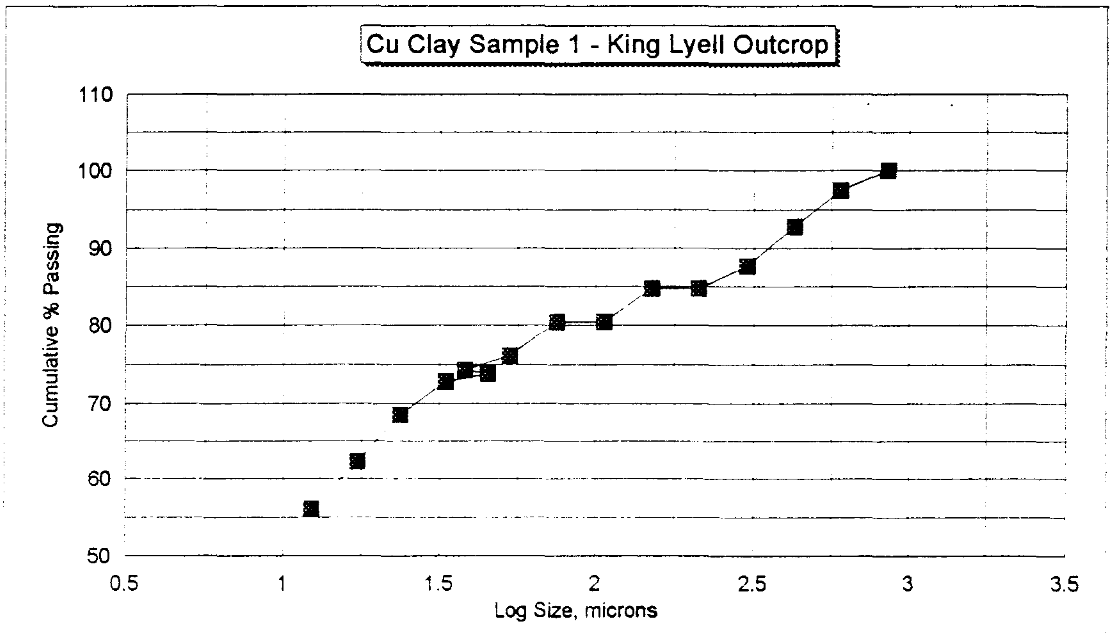


TABLE 2: Copper Clay King Lyell Outcrop Sample

Sieve Size Microns	Weight % Held	Cu %	Cu Units	Cu Dist %	Corr Cu Dist %	Cum Cu Dist %
212	15.26	5.80	88.51	36.23	36.23	36.23
106	4.32	6.35	27.43	11.23	11.23	47.46
53	4.32	6.85	29.59	12.11	12.11	59.57
38	1.83	3.65	6.68	2.73	2.73	62.30
-38 Measd	74.27	1.24	92.09	37.70	37.70	100.00
-38 Calcd	74.28	0.80	59.72			
23.7	5.76	1.23	7.08	2.90	4.47	66.78
12.3	12.44	0.67	8.33	3.41	5.26	72.04
-12.3	56.08	0.79	44.30	18.13	27.96	100.00
Total Calc	100	2.44	244.31	100.00		
Head Assay	100	2.70				

TABLE 3: Summary of MLMRC Flotation Results on Copper Clays

Sample	Head	Sands						Slime				
	% Cu	% Cu	% Wgt	Cu Dist %	Cu Rec Sand %	Cu Rec Total %	Conc Cu %	% Cu	% Wgt	Cu Dist %	Cu Rec %	Conc Cu %
Native Copper Clays	2.3	6.67	26.53	77.60	91.49	71.00	26.81	0.70	73.47	22.40		
Cu Blocks Float II	0.1	0.13	50.30	57.30	55.15	31.60	0.86	0.09	49.75	43.01		
Consols 3	0.9	1.78	22.70	86.72	89.46	41.60	17.30	0.61	77.30	53.50		
Cu Blocks V	0.3	0.34	64.80	84.80	91.27	77.40	10.48	0.11	35.20	15.20		
King Lyell Outcrop (1996)	1.4	5.91	25.70	62.30				1.24	74.30	37.70		

MEMORANDUM

TO:	Peter Benjamin
FROM:	Tony Weston
DATE:	1 August, 1997
SUBJECT:	King Lyell Copper Clays
CC:	Hamish Bohannan, Colin Farr, Nick Clarke, Peter Williams

Scope

To recommend a strategy for further exploration work on the King Lyell copper clay deposit, and other copper clays in general.

The study was intended to briefly examine the potential economics of the King Lyell, without detailed geology, mining or milling plans. High grading was not considered.

Conclusions

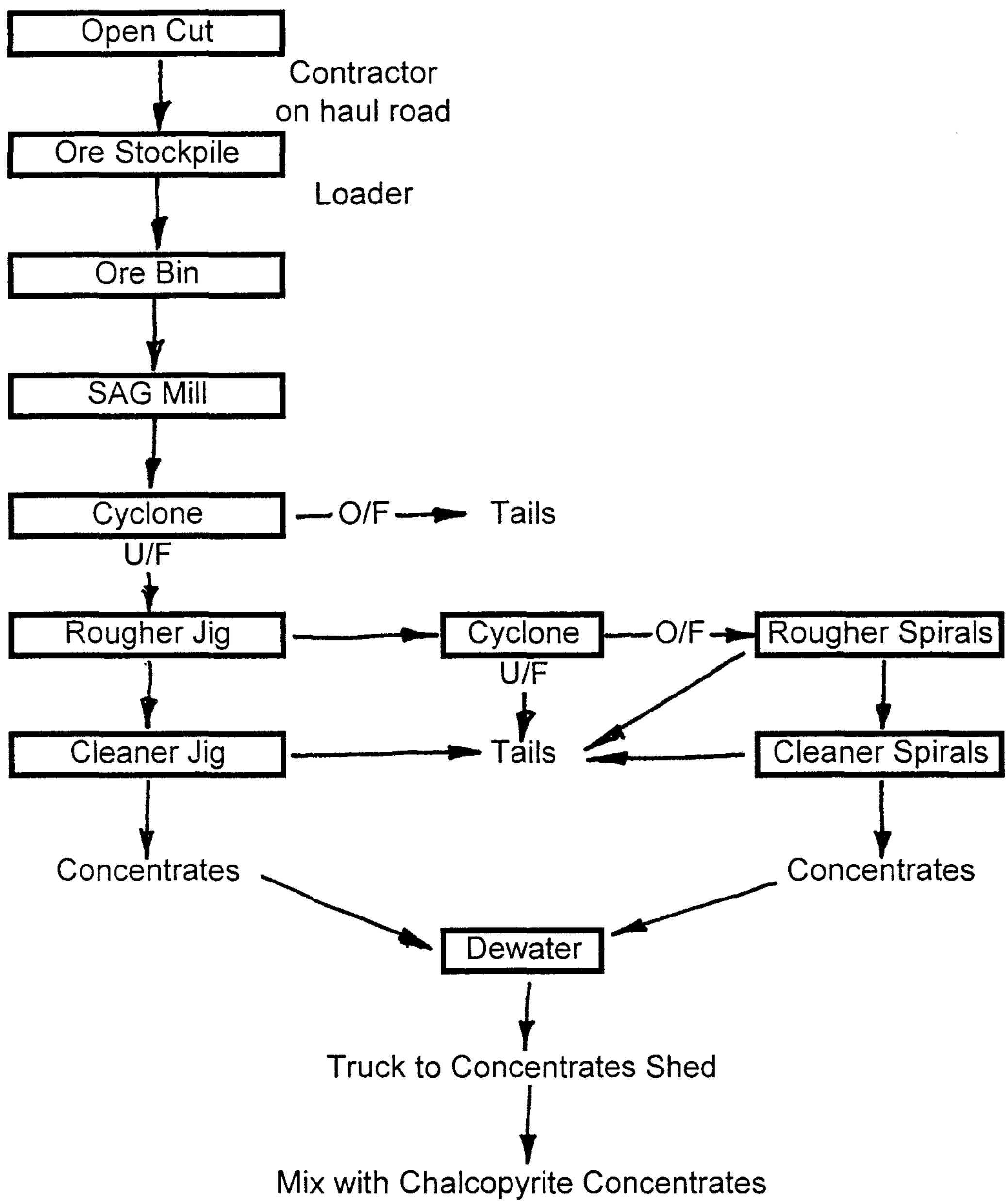
- the current lower grade (full) King Lyell resource has a negative NPV at 250,000 tpa due to the small size of the deposit and the capital cost of setting up a mill
- a conceptual larger resource at a similar grade, mined at 750,000 tpa for ten years would have a positive NPV, but sensitive to changes in operating costs, head grade, stripping ratio and copper price
- environmental and operating strategies for this area of the Linda Valley would need to be well thought through

Recommendations

- check the overall potential resource for the King Lyell and other copper clays
- if this is likely to be in the range from 5 to 10 million tonnes continue with further exploration work
- if less than 5 million tonnes it is unlikely for the copper clays to be attractive

Tony Weston
Senior Mining Engineer

PROPOSED KING LYELL COPPER CLAYS FLOW CHART



O/F - overflow
U/F - underflow

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