AUSTMINEX N.L.

BENAMBRA PROJECT

MINE REOPENING FEASIBILITY STUDY

Summary Report

November 2001

Prepared By

Austminex NL ABN 56 005 470 799
Level 6 150 Queen Street Melbourne VIC 3000
PO Box 339 Collins Street West 8007
Telephone 61 3 9670 5111 Facsimile 61 3 9670 8111
email info@austminex.com.au
www.austminex.com.au

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BENAMBRA PROJECT STUDY

EXECUTIVE SUMMARY

The Benambra Copper/Zinc Mine is located in the highlands of northeastern Victoria, approximately 300km from Melbourne. The mine was operated by Denehurst from 1992 for four years, producing copper and zinc concentrates. Operations at the mine were suspended in July 1996 due to a combination of factors with the plant being placed on care and maintenance.

Austminex NL has an exclusive option with Nick Brooke of Pricewaterhouse Coopers, of 215 Spring Street Melbourne, for the purchase of the Benambra leases and infrastructure. At the time of writing (23rd October 2001) the agreement is for the option to be exercised at any time up until October 2002. There are monthly incremental payments of $10,000 per month until April 2002, at which time the monthly payments are increased to $25,000. The final bullet payment for purchase of the leases and facilities is $317,000. Negotiations are currently in progress with Collex (who have bought out Denehurst's first secured creditor and are seeking to settle the amount owing to the second secured creditor – Nissho Iwai) to reduce the monthly payments to $3,000 and the final payment to $100,000. An extension of time to exercise the option will be sought from the Administrator to coincide with the current mining lease expiry date of April 2004.

MINE RE-OPENING STUDY

For Austminex to make the decision to exercise the exclusive option to acquire the Benambra base metal project, it was necessary that a mine reopening study be conducted. The purpose of the Study was to confirm that a practical and economic project, with at least a ten year mine life, could be achieved. The aim of the Study was to confirm and add to the resources and mineable reserves, review further exploration potential, develop mine designs, establish a process for enhanced metallurgical performance, confirm the requirements for process plant refurbishment and enhancement, investigate opportunities for an alternative power supply, establish capital and operating costs, establish markets and handling procedures for copper and zinc concentrates, establish appropriate environmental strategies and bond levels and produce an updated work plan for approval by statutory authorities.

A study team was formed which comprised Kevin Tomlinson – Managing Director, Andrew McDougall – Project Manager, Adrian Molinia – Mine Design and Financial Modelling, Harry Boughen – Metallurgy and Process Design, Gary McArthur (Moda) – Resource Evaluation and Reserve Assessment, Ray Hazeldene Exploration Geology and Fiona Robertson –Funding. The team members had extensive experience in base metals mining.

The Study started in late October 2000, with the commencement of resource confirmation drilling. This was followed by resource extension drilling, which was completed in July 2001. In this period extensive metallurgical testing was

successfully conducted on samples obtained from the drilling programme. All other aspects of the Study have been completed or have reached an advanced state.

DRILLING

Thirty-eight holes, totalling 8,091m were drilled at the Wilga Mine and Currawong Deposit during the year. They were drilled to collect samples for metallurgical testing, check for water in the Wilga Mine and to increase the resource/reserve base for future mining.

RESOURCE AND RESERVE ESTIMATES

Data from previous drilling campaigns and mine production records and surveys were combined with the results from the drilling programme and were incorporated into the resource data base, from which new resource estimates were prepared for the Wilga and Currawong deposits. The revised Resources for Wilga and Currawong are given in the tables below. The outcome of this work was an increase in combined Measured and Indicated Resources of 45%, as compared to those estimated in December 2000.

Wilga Measured and Indicated Resource

	Tonnes	Copper %	Zinc %	Silver g/t	Gold g/t
TOTAL	3.0 Mt	3.3	6.3	37	0.53

Currawong Indicated Resource

	Tonnes	Copper %	Zinc %	Silver g/t	Gold g/t
TOTAL	7.7 Mt	2.0	4.2	38	1.2

Currawong Inferred Resource

	Tonnes	Copper %	Zinc %	Silver g/t	Gold g/t
TOTAL	1.3 Mt	1.9	4.3	42	1.4

An ore reserve was subsequently estimated in August 2001 based upon the resource and taking into account metallurgical, mining and economic parameters. The total Ore Reserve is 6.1 million tonnes as detailed in the tables below. This result has met the Company's objective of an Ore Reserve capable of sustaining a 10 year mine life at a throughput rate of 600,000 tonnes per annum. Further drilling is planned in the early years of production with the aim of further extending project life.

The Ore Reserves of the Wilga and Currawong deposits as estimated by McArthur Ore Deposit Assessments Pty Ltd (MODA) are:

Wilga Probable Ore Reserves

	Tonnes	Copper %	Zinc %	Silver g/t	Gold g/t
TOTAL	1.6Mt	2.4	6.1	35	0.5

Currawong Probable Ore Reserves

	Tonnes	Copper %	Zinc %	Silver g/t	Gold g/t
TOTAL	4.5Mt	2.1	4.4	37	1.3

Ore Reserves are estimated by applying appropriate technical and economic considerations and factors to Measured and Indicated Resources. A significant consideration is the impact of costs, and at Benambra there is some uncertainty in the cost estimates for mine closure environmental work, as mine closure requirements have not yet been fully determined by the DNRE and the Company. The Company has adopted what it considers as reasonable requirements of good environmental practice in arriving at cost parameters for Ore Reserves purposes. Owing to these uncertainties, no Ore Reserves are classified to a confidence level higher than Probable.

MINING

Work has concentrated on producing a mine plan based on the Measured and Indicated resources at Benambra. Sufficient Ore Reserves have now been defined in recent drilling by Austminex to sustain production at 600,000 tonnes per year.

Access to the existing Wilga Mine workings is currently prevented by a plug of fill extending from the portal down the drive for approximately 30m. Similarly the previous ventilation shafts and adit have been sealed with fill. The fill can be readily removed to reinstate access, but for the purposes of the Study this was not done in order that Austminex not assume environmental responsibility for the mine. Mine records indicate that the existing decline, drives and openings should be in sound condition. A probe hole drilled to within 5m of the base of the mine did not encounter any water.

The current reopening plan is to mine Wilga ahead of Currawong, with Wilga providing the first 2.5 years of production, followed by Currawong for a future 7.5 years. The initial years of mining will be carried out using underground mining contractors. The contractors will be expected to supply all manning, equipment and services to carry out the mining operation at the Benambra Mine Site. Indicative establishment and mining costs were compiled from contracting company responses and in house knowledge.

The mining method selected for both Wilga and Currawong is accessed by decline and is predominantly Sub-level Open Stoping with introduced waste backfill. The basic layout is 20m-wide open stopes separated by 10m-wide vertical pillars. The stopes are taken to the full height and cross-sectional width of the economic mineralisation and alternate stopes are backfilled with waste rock to allow pillar recovery without major collapse or surface subsidence. A proper cave layout will allow the pillars to be mass blasted into the unfilled stope voids and drawn down using cave draw conditions under introduced waste fill.

Unplanned dilution was allocated as a consistent 2-metre envelope on the exposed stope walls and backs, using grades modelled from local waste assays. For pillars, a constant 30% dilution component was allocated for both Wilga and Currawong.

Recovery factors of 95% for primary stopes and 85% for pillars were applied.

For Post Pillar Cut and Fill stopes 82% recovery and 12% dilution have been applied, based on previous experience of this method when mucking on a waste fill floor.

METALLURGICAL TESTING

A comprehensive laboratory test programme was commissioned to develop an appropriate flow sheet for the treatment of ore from both the Wilga and Currawong ore bodies. Throughout the test programme, the aim was to use commonly available flotation reagents and to avoid the use of exotic chemicals wherever possible. In addition, every effort was made to minimise the number of treatment stages and the number of recycle streams, so that the flow sheet should be as simple as possible. Over 150 laboratory flotation tests were conducted at Optimet in Adelaide.

By changes to the suite of flotation reagents used and by the incorporation of ultra-fine grinding (down to 10 micron) of both copper and zinc primary rougher concentrate, significant improvements in both concentrate grade and recovery were demonstrated over those achieved during the previous operations. The improvements in metallurgical performance more than compensate for the increased operating and capital costs of the new process.

The predicted process concentrate grades and recoveries, based on the laboratory test programme, are summarized in the table below.

Product	Copper Concentrate		Zinc Concentrate	
Parameter	Wilga	Currawong	Wilga	Currawong
Concentrate Grade	25%Cu	26%Cu	50%Zn	51%Zn
Metal Recovery	83%	83%	85%	80%

Locked cycle laboratory tests were completed to demonstrate that the chosen flowsheet was stable and that the open cycle test results could be reproduced.

Tests indicated that the finely ground product can be readily settled and filtered.

Test results show that flotation tailings from the general run of ore from Benambra is unlikely to be a candidate for profitable recovery of gold by cyanidation, either directly or following some form of pre-treatment such as roasting or bacterial oxidation. Ore that might contain sufficient value to warrant further consideration for gold recovery is likely to be small in tonnage and difficult to isolate. For these higher gold grade pockets higher arsenic and bismuth levels are likely to pose operational, marketing and environmental problems.

At the request of Alan Martin, an investigation was conducted into the Indium content of the Benambra concentrates and possible economic benefits. The outcome is that Indium and other minor elements will not enhance the project. Further details are provided at the end of the metallurgical summary section.

The ore processing facility will involve plant and equipment that is commonly and widely used throughout the mining industry. Although ultra-fine grinding is a relatively recent application to flotation plants, the equipment involved is both proven and robust for mineral industry applications.

The samples remaining at Optimet, following the metallurgical testwork programme, have been packaged up in 4 x 200 litre drums and sent back to the mine site for safe keeping. In this way there will be no ongoing storage charges. A portion of an amalgamated concentrate sample was retained by Optimet, in the event that a decision is made to send a sample to WMT in Perth for an Activox partial leach test.

PROCESSING PLANT AND INFRASTRUCTURE

It is proposed that the existing treatment plant be refurbished and expanded to incorporate features established from the metallurgical testwork. Metplant Consulting engineers were engaged to review the existing plant and to establish the equipment and services necessary to meet the throughput and metallurgical performance requirements. There have been extensive equipment selection and layout reviews for 300,000 tpa and 600,000 tpa throughput options, based on providing a plant, which will be technically, operationally and cost effective. The plant is based on proven equipment and technology.

The existing Tailings Management Facility is in good condition but will require two lifts (for a 600,000 tpa production rate the first wall lift needs to occur in the second year of production) of the wall over the life of the operation, in order to accommodate the tailings. The existing wall crest height is RL 1174.6m (the plastic liner covering the top 1.6m of the upstream face requires remedial work prior to the resumption of mining operations), with approvals in place for the wall to be raised to a final crest height of RL 1200m. The current production proposal of 600,000 tpa for 10 years will require a final crest height of approximately RL 1187m. This is contingent on material for raising the wall and for mine back fill coming from a pit excavated in the back of the tailings facility. URS environmental consultants were engaged to provide advice for the wall raising and the final closure strategy.

The final transport arrangements from the site are still under review. It is proposed that the concentrate be trucked direct to Australian smelters at Port Kembla and Newcastle or trucked to the Bairnsdale inland port and railed through to Geelong for shipment to overseas smelters. Preliminary proposals and cost structures were obtained for the direct trucking and combined trucking/rail options.

CONCENTRATE MARKETING

Austminex engaged the services of Colken, industry specialists, to assist with advice on the marketability and marketing of the copper and zinc concentrates. Revised and improved concentrate specifications were established from the comprehensive metallurgical testwork.

It has been confirmed from smelters and traders that the concentrates contain no deleterious elements and that they are readily marketable. Smelting terms and conditions have been based on standard, prevailing, frame contracts as the Benambra concentrate production is not sufficiently large or of premium quality to attract special favourable terms. There have been preliminary offtake agreement discussions with Australian and overseas smelters. There are no firm offtake agreements in place.

ENVIRONMENT

Studies were commissioned which confirmed that the condition of the site and in particular the Tailings Management Facility (TMF), the surrounding forest and the streams and rivers, which pass through the property, are in good standing.

Austminex has taken over the responsibility from the DNRE for regular environmental monitoring and water sampling. The monitoring results are collected by Ron Thumerer, the Austminex employed caretaker and Thiess Environmental Services and are sent to the DNRE for their scrutiny and records. The cost to Austminex for this ongoing monitoring is around $2,000 per month. The DNRE retain ultimate responsibility for the property and in particular the condition and water level in the tailings facility.

Whilst the Tailings Management Facility is currently in excellent condition, with the tailings held in a benign condition under water cover, it was necessary that Austminex establish the best environmental end of mine life closure strategy

for the facility, so that there are no long term environmental impacts or obligations on the Company.

Austminex commissioned URS, environmental consultants, who have proposed the world's best practice method of water cover closure, as being the best option for Benambra. This is supported by the experience gained from the Canadian MEND Studies and since the cessation of mining operations at Benambra in 1996. The approved closure strategy, as currently incorporated in the licence to operate, for the tailings facility, is water removal and the application of a multi layered dry cover. It is recognised by the statutory authorities that this is no longer a satisfactory or achievable method of closure.

Austminex has submitted a comprehensive report, supporting the water cover closure strategy, to the DNRE and the EPA for their approval and for the establishment of an acceptable environmental bond. This proposal has not been readily accepted by the DNRE and EPA and it is believed that they will have additional, as yet unspecified, requirements. The URS proposal, if accepted, should result in a bond for full site remediation of approximately $4.0M. Based on the DNRE and EPA deliberations, they may well require long term site monitoring and a bond level considerably higher than this, with a portion retained for a long period of time (possibly in perpetuity), post closure and rehabilitation.

Further to what is written above, there have been recent discussions (23rd November 2001) between URS and the EPA regarding the EPA's closure strategy position. It appears that the EPA has moved a considerable way in accepting a wet closure, but will not accept a straight water cover or the organic waste layer. They are seeking a combination of a water cover and a layer of crushed rock over the entire surface of the tailings. Austminex has still to be formally advised of the requirements and the details. Once these are known the cost of complying with the EPA closure strategy can be established. A file note indicating the position, as understood at present, is included in the attached Statutory Approvals summary report.

STATUTORY APPROVALS

Austminex has received legal advice from John McMullan, Solicitors, that the approvals, which were in place for the mine when it previously operated, are in good standing. The current Environmetal Effects Statement (EES) is applicable only to ML1865 and to underground mining operations at Wilga and Currawong, treatment of ore at Waxlip Spur, non use of cyanide as a reagent and disposal of tailings in the existing facility. Any future activities which involve underground mining which can't be accessed from Wilga or Currawong, open cut mining, significant variations to the treatment process or the disposal of waste materials in a location other than the existing tailings facility will require the preparation and approval of a new EES.

The following leases and their status, comprise the Benambra Tenements (verification of status and conditions can be obtained from Graham Robertson – Australian Mineral Services ph. 03 9842 7694):

- Mining Lease ML1865 – held in the name of Denehurst and is current until April 2004, with the process for renewal to commence at least 6 months prior to expiry. At present transfer of this title to Austminex will require an immediate environmental bond payment/facility to be made by Austminex to the DNRE and will also result in Austminex having full environmental responsibility and liability for the cost of the agreed final closure strategy. Austminex has written to George Buckland (DNRE) seeking some relief from these requirements, so that Austminex can take over the title without having to accept full environmental responsibility until the decision to proceed with mine redevelopment is made and operations commence. A response to this letter is awaited.
- Mineral Licences MIN 4279 and MIN 4281 – these are in the name of Denehurst.
- Exploration Licence EL 3458 (Banksia) – this is in the name of Denehurst, is pre native title and has been recently renewed, with currency until 10th August 2002. There is currently a nil expenditure requirement on this licence.
- Exploration Licence Applications ELA 3980 and ELA 3984 (now combined into ELA 4599) – the former ELA's 3980 & 3984 were held in the name of Denehurst. They were scheduled to expire in August 2001. The Denenhurst Administrator indicated that he was not willing to pay the cost of renewal. The DNRE was seeking a reduction in area covered by these two ELAs. Austminex conducted a review of prospectivity and was able to recommend areas for relinquishment and new boundaries. The revised areas have been consolidated into one new ELA 4599 and upon recommendation and agreement of the Administrator, the application has been made directly in the name of Austminex NL. The application process is in progress and is subject to public advertising and native title negotiations. The advertising has taken place but to date there have been no discussions with native title parties.

For mining and processing to recommence it is necessary that a Variation to the Work Plan be submitted to the DNRE for approval. The Variation to the Work Plan details any changes in quantities, mining and processing methods and the strategy for long term closure at the conclusion of mining operations. This document has been prepared and submitted to the DNRE for approval. The DNRE has responded with a list of questions for clarification and additional explanation. There has been no formal response by Austminex to the DNRE to date. Most of the questions only require straight forward responses, with the major issue still to be resolved being the tailings facility closure strategy. The Variation to the Work Plan cannot be formally approved by the DNRE until Austminex exercises the option to purchase from the Administrator and the licences are transferred by the DNRE to Austminex.

The DNRE have indicated it will immediately require a non negotiable, preliminary, environmental bond of $1.5M, upon transfer of the licences to Austminex. Also Austminex will immediately take on all environmental responsibility, including the property closure requirements and in particular the

long term closure of the tailings facility. The quantum of the actual environmental bond will be based on the cost of implementing an agreed closure strategy. It is not in Austminex interest to exercise the option and take over the leases until there is an assured economic project with full knowledge of the closure requirements and until the closure strategy and quantum of the bond are agreed to the satisfaction of both parties. Note the comment above which refers to Austminex making application for an interim transfer of licence without having to accept environmental liability for the property.

CAPITAL AND OPERATING COSTS

Capital and operating costs have been established from estimates prepared by consulting engineers, contract mining companies, service organizations and in house knowledge.

For the 600,000 tpa throughput rate the capital costs are estimated at:

- Processing plant refurbishment and expansion (Metplant +/- 10% Estimate) - $20.6M
- Mine reopening and initial stope development (contract rates and in house) - $2.5M
- Mine purchase, final feasibility, housing and infrastructure - $2.3M

Capital costs were estimated for alternative production and plant scenarios for financial analysis:

- 300,000 tpa plant: refurbished, improved, fine grinding, new crushing plant (Metplant +/- 20% Estimate) - $11.6M
- 300,000 tpa plant: refurbished, no operability improvements, fine grinding, existing crushing plant (Metplant Estimate) - $6M to $9M.

Projected cash operating costs for the 600,000 tpa improved operability option, based on diesel generated power, are US 38 cents/lb for zinc and US 64 cents/lb copper. It needs to be noted that the operating costs are neat and did not include a contingency cost allowance.

FINANCIAL MODELS

A range of models based on different production, capital and operating cost requirement scenarios was considered during the study process.

The project economics indicated that the project was enhanced by an increase in scale of operation from the originally intended 300,000 tpa to 600,000 tpa

The Base Case 600,000 tpa model, based on August Bankwest 27 month metal price forecasts and consensus long term metal price forecasts and with no allowance for royalty payments to the Victorian Government and Western Mining, provided a $10M NPV and 18% IRR, at a 10% discount rate.

A very basic 300,000 tpa model with refurbishment, no process enhancements, provision for fine grinding (giving an unachievable plant capex of $4.0M), minimum mining rehabilitation costs based on Wilga only, with no allowance for Government and Western Mining royalties and October

Bankwest 27 month metal price forecasts and revised long term forecasts, provided a negative $6.3M NPV and a negative 19% IRR, at a 10% discount rate.

PROJECT ENHANCEMENTS

- Power Supply

  For the previous mining operations, and at present, there is no connection of the mine site to grid power. Power was provided by on site diesel driven generators. The electrical energy requirements for processing the ore are high (approximately 5MW) and form a significant component of the operating costs. Alternative power supply options have been investigated and pursued. The energy charges for grid power are 75 to 100% cheaper than diesel generation (grid @ 7.5c/kWh versus diesel @ 13.5c/kWh, based on the availability of the Diesel Fuel Rebate and an on site fuel cost of 36.9c/l) and have the potential of improving Project economics through lower ore processing costs (potential saving of $3.50/tonne of ore treated at the 600,000 tpa rate).

  Other forms of energy generation, using forest/organic waste material for firing a boiler for steam generation have been investigated and don't appear to provide power at a rate cheaper than that currently established for diesel. There are also logistic and potential environmental issues associated with this form of generation.

  TXU, the owners of the grid system in the Bairnsdale/Omeo/Benambra area, have provided a back of the envelope capital cost estimate of $20M for providing a grid connection of sufficient capacity to the mine. To confirm the capital cost of the connection, TXU have quoted a study cost of $133,000 + GST. TXU have stated that they will not contribute to the capital cost of providing the grid connection.

- The Company has held discussions with State and Federal Government authorities, seeking their combined financial support for bringing grid power to site. A recent letter received from the Victorian Minister for State and Regional Development indicated that the Victorian Government will not be providing funds towards an upgraded and extended power supply.

- Exploration

  The Benambra tenements host a very prospective belt of volcanogenic lithologies that have already been proven to host base metal deposits. Typically, the volcanogenic massive sulphide (VMS) deposits occur in clusters and it is evident from a review of previous exploration that there are many targets worthy of further exploration.

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BENAMBRA PROJECT

DRILL HOLES AND INTERSECTIONS 2000/2001

Currawong Drill Holes and Intersections 2000 - 2001.xls

Wilga Drill Holes and Intersections 2000 - 2001.xls

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BENAMBRA PROJECT

MINERAL RESOURCES
AND
ORE RESERVE ASSESSMENT

FINAL REPORT SUMMARY

SEPTEMBER 2001

Prepared by
G.J.McArthur PhD FAusIMM CPGeo MMICA MSEG
(McArthur Ore Deposit Assessments Pty Ltd)

A.B.Molinia BE(Hons) FAusIMM CPMin
(McArthur Ore Deposit Assessments Pty Ltd

Assisted by
S.M.Richardson BSc(Hons) MAusIMM (McArthur Ore Deposit Assessments Pty Ltd)
R.Hazeldene Grad.Dip.App.Geol MAusIMM (Mindeco Pty Ltd)

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1. SUMMARY

1.1 MINERAL RESOURCES

The Mineral Resources of the Wilga and Currawong deposits, as estimated by McArthur Ore Deposit Assessments Pty Ltd (MODA) in July 2001 are:

TABLE 1 - WILGA MINERAL RESOURCES

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Measured	1585	3.96	3.4	0.53	6.4	37	0.52
Indicated	1454	3.79	3.3	0.47	6.2	36	0.54
Inferred	244	3.28	4.0	0.26	2.9	28	0.21
TOTAL (rounded)	**3300**	**3.83**	**3.4**	**0.48**	**6.1**	**36**	**0.50**

CURRAWONG MINERAL RESOURCES

TABLE 2 - A LENS

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Indicated	268	4.13	2.0	0.78	4.0	37	1.5
Inferred	23	4.01	2.3	0.59	3.1	29	1.3
TOTAL	**291**	**4.12**	**2.1**	**0.76**	**3.9**	**37**	**1.5**

B LENS

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Indicated	1302	4.37	1.7	0.73	3.2	38	1.1
Inferred	244	4.38	1.8	1.1	4.0	41	1.7
TOTAL	**1546**	**4.37**	**1.7**	**0.78**	**3.3**	**38**	**1.2**

J LENS

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Indicated	296	4.39	1.9	0.32	2.2	26	0.71
Inferred	0						
TOTAL	**296**	**4.39**	**1.9**	**0.32**	**2.2**	**26**	**0.71**

K LENS

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Indicated	511	4.10	1.0	1.1	4.9	45	1.4
Inferred	0						
TOTAL	**511**	**4.10**	**1.0**	**1.1**	**4.9**	**45**	**1.4**

M LENS

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Indicated	5301	4.11	2.1	0.74	4.4	38	1.2
Inferred	1084	4.04	2.0	0.89	4.4	43	1.3
TOTAL	**6385**	**4.09**	**2.1**	**0.77**	**4.4**	**39**	**1.2**

ALL CURRAWONG LENSES **(rounded)**

Category	'000 Tonnes	Density	%Cu	%Pb	%Zn	g/t Ag	g/t Au
Indicated	**7700**	**4.16**	**2.0**	**0.75**	**4.2**	**38**	**1.2**
Inferred	**1300**	**4.10**	**1.9**	**0.92**	**4.3**	**42**	**1.4**
TOTAL	**9000**	**4.15**	**2.0**	**0.77**	**4.2**	**39**	**1.2**

1.1.1 Summary of resource estimation method

The Mineral Resources at Benambra have been defined geologically using the massive sulphide envelope, without any grade cutoff.

MODA used Datamine software to undertake a thorough re-assessment of the pre-existing Wilga and Currawong geological data and merged this with new data provided by Austminex drilling.

The resources were estimated using industry-standard geostatistical techniques with Measured, Indicated and Inferred resource categories assigned from the actual calculated kriging errors.

Those resources mined previously by Denehurst were allocated by MODA using available Denehurst mine survey data. There is a significant discrepancy between the resource within the mined-out envelope and the production reported by Denehurst. This can only be properly resolved once the mine is re-established and re-surveyed. However, a large proportion of the current in-situ resource is located in sill pillars between levels that are non-recoverable.

The Measured Resources that fall within the planned stopes at Wilga have been downgraded to Probable Reserve status due to current uncertainties in cost estimates for mine closure environmental work.

1.2 ORE RESERVES

The Ore Reserves of the Wilga and Currawong deposits, as estimated by McArthur Ore Deposit Assessments Pty Ltd (MODA) in July 2001 are:

TABLE 3 - WILGA ORE RESERVES

(all Probable)

Lens	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%Zneq
Wilga	1636	2.4	0.51	6.1	35	0.50	12.0
TOTAL	**1636**	**2.4**	**0.51**	**6.1**	**35**	**0.50**	**12.0**
rounded	**1.6Mt**						

TABLE 4 - CURRAWONG ORE RESERVES

(all Probable)

Lens	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%Zneq
A	93	1.5	1.2	6.2	50	1.9	10.0
B	253	1.9	1.3	4.3	44	1.9	9.0
J	24	2.7	0.3	2.0	25	0.6	8.7
K	195	1.1	1.6	6.3	51	2.0	9.0
M	3890	2.2	0.7	4.3	36	1.2	9.6
TOTAL	**4455**	**2.1**	**0.78**	**4.4**	**37**	**1.3**	**9.6**
rounded	**4.5Mt**						

1.2.1 Summary of reserve estimation method

The mining method selected for both Wilga and Currawong is predominantly Sub-level Open Stoping with introduced waste backfill. In general, a layout of 20m-wide open stopes separated by 10m-wide vertical pillars is proposed. The planned mining method is discussed in detail in the separate Mining Volume.

Ore Reserves were estimated by designing stope and pillar layouts in 3D around a nominated Zn-equivalent cutoff, thereby excluding sub-economic mineralisation from the mine design. For Wilga, a grade cutoff of 9% Zn-equivalent (%Zneq = [2.5*%Cu]+[%Zn], as determined from financial modelling) was used. However, at Currawong, a lower 8% Zn-equivalent grade cutoff was necessary to create geometry favourable for mining the inherently lower grade mineralisation.

Stope and pillar design at Wilga also had the requirement to allow mining around existing backfilled mining voids.

Unplanned dilution was allocated as a consistent 2-metre envelope on the exposed stope walls and backs, using grades modelled from local waste assays. For pillars, a constant 30% dilution component was allocated for both Wilga and Currawong.

Recovery factors of 95% for primary stopes and 85% for pillars were applied. For Post Pillar Cut and Fill stopes 82% recovery and 12% dilution were used, based on previous experience of this method when mucking on a waste fill floor.

1.3 STATEMENT OF COMPETENCY

The authors who prepared this resource and reserve estimate, are employees of McArthur Ore Deposit Assessments Pty Ltd (MODA), Fellows of the Australasian Institute of Mining and Metallurgy (AusIMM) and Chartered Professionals (CPGeo, CPMin). Dr McArthur is also a Member of the Mineral Industry Consultants Association (MICA). Both authors have 25-30 years experience as professionals in the mining industry, mainly involved in the underground extraction of base metal deposits. They also have extensive experience in the estimation of resources and reserves, particularly in the volcanic-hosted massive sulphide style of mineralisation, similar to Benambra. As such, the authors both meet the formal requirements as defined in the JORC Code (AusIMM, 1999), to be *Competent Persons* for the estimation of Benambra Mineral Resources and Ore Reserves.

2. GEOLOGY

2.1 INTRODUCTION

The Wilga and Currawong deposits comprise lenses of polymetallic massive sulphide, hosted by a strongly deformed, upward-younging, Upper Silurian volcano-sedimentary sequence (Fig.1).

They are interpreted as examples of *Volcanic Hosted Massive Sulphide* (VHMS) deposits, (Allen and Barr, 1990) from the following observations:

- The massive sulphide bodies and associated alteration zones are overprinted by early cleavages (S_1 and S_2) indicating that mineralisation and alteration formed prior to deformation. Both the host rocks and earliest deformation (D_1 and D_2) are Late Silurian in age.
- There is a tight stratigraphic control on the massive sulphide bodies.
- Stratigraphy, depositional setting and alteration style, are similar to other less deformed VHMS deposits.

Allen's interpretation suggests the massive sulphides formed within siltstone, in the proximal facies association of a low profile, non-explosive, moderate to deep water submarine volcano, composed of turbiditic sediments interbedded with numerous rhyolitic to basaltic sills, lavas and associated hyaloclastites.

All orientations referred to in this report, relate to local mine grid (see section 3.2.1).

2.2 REGIONAL GEOLOGY

2.2.1 Stratigraphy

The Benambra massive sulphide deposits occur in an isolated structural remnant of the Middle-Upper Silurian Cowombat Rift, known as the Limestone Creek Graben. The Cowombat Rift is the southernmost of a number of Silurian depositional basins in the Palaeozoic Lachlan fold belt that contain VHMS deposits. Middle-Upper Silurian rocks are interpreted as structural relics of the Cowombat rift and are termed the Enano Group.

The Enano group comprises the following units starting from the stratigraphic base:

Thorkidaan Volcanics: A 2-3 km thick pile of massive coherent porphyritic rhyolites with minor volcaniclastic sandstone, siltstone and conglomerate.

Cowombat Siltstone: Conformably overlies and interfingers with the Thorkidaan Volcanics in the Wilga Currawong area. It is up to 500m thick and generally shows a trend from limestone and thick-bedded coarse clastics near the base to siltstone – mudstone, with thin sandstone turbidites, towards the top.

Gibsons Folly Formation: Conformably overlies the Cowombat siltstone and is around 500m thick. Characterised by tabular to lenticular dacite, andesite and

basalts enclosed in mudstone and sandstone turbidites and hosts the Currawong and Wilga massive sulphide deposits. Massive porphyritic rhyolite bodies occur locally. Both the Wilga and Currawong deposits occur footwall to a distinctive amygdaloidal andesite / basalt unit. Allen (1987) claims this hangingwall unit can be correlated through several drillholes from Wilga through to Currawong.

2.2.2 Structure

Allen, (1987) recognised three main generations of deformation in the Wilga–Currawong area. The first, (D_1), is represented by a regional, bedding parallel foliation and rare small folds. The main regional east trending folds (F_2) in the Wilga–Currawong area are related to the second deformation (D_2) and comprise tight to isoclinal overturned folds with axial planes dipping north, parallel to the Indi Fault. On a regional scale the Wilga and Currawong deposits occur on the mainly upright limb of a poorly defined overturned F_2 syncline, which has been disrupted by S_2 parallel shear zones. Overprinting the F_2 structures are N to NE trending upright, open F_3 folds and steep faults. Offset appears to be consistently north side up across these structures.

D_1 and D_2 have only affected Silurian and older rocks and may be part of the same progressive event, indicating these structures relate to the Late Silurian Bindian deformation. D_3 structures also affect the Devonian Snowy River Volcanics, suggesting these correlate with the Middle Devonian Tabberabberan deformation (Allen and Barr, 1990).

During a structural interpretation of the Currawong deposit in 1995, ERA-Maptech put forward an alternative structural interpretation. They recognized not only an early fault set equivalent to Allen's F_2 fault orientation and a late fault set equivalent to his F_3 faults, but also an intermediate set of faults trending NE - E which were interpreted to truncate F_2 structures but predate F_3 structures. It was postulated that these faults might have formed contemporaneously with the earlier (F_2) set, possibly as transfer faults.

Etheridge (1989), suggested that formation and subsequent deformation of the Cowombat Rift was controlled by regional NE trending strike slip faults. Subsidence and uplift was spatially and temporally complex, occurring on bends and splays of the regional fault system. Synvolcanic extension and subsequent folding and faulting in the Wilga-Currawong area could be explained by dextral displacement on the regional fault system during the Middle Silurian, followed by sinistral displacement on the same fault system during the Late Silurian to Early Devonian.

2.3 DEPOSIT GEOLOGY

2.3.1 Wilga

The Wilga deposit comprises a single lens of pyrite-sphalerite-chalcopyrite massive sulphide with less abundant pyrite-chalcopyrite-chlorite "stringer" mineralisation. Mineralisation occurs 50 – 150m below surface, is moderately north dipping and largely conformable to stratigraphy and the dominant cleavage. That portion of the lens dominated by massive sulphide has a strike length of approximately 300m, down-dip extent of 230m and maximum thickness of 40m. "Stringer" sulphide

mineralisation extends for a further 200m along strike to the west but is generally less than 3m thick (see Fig.2 and Appendix I).

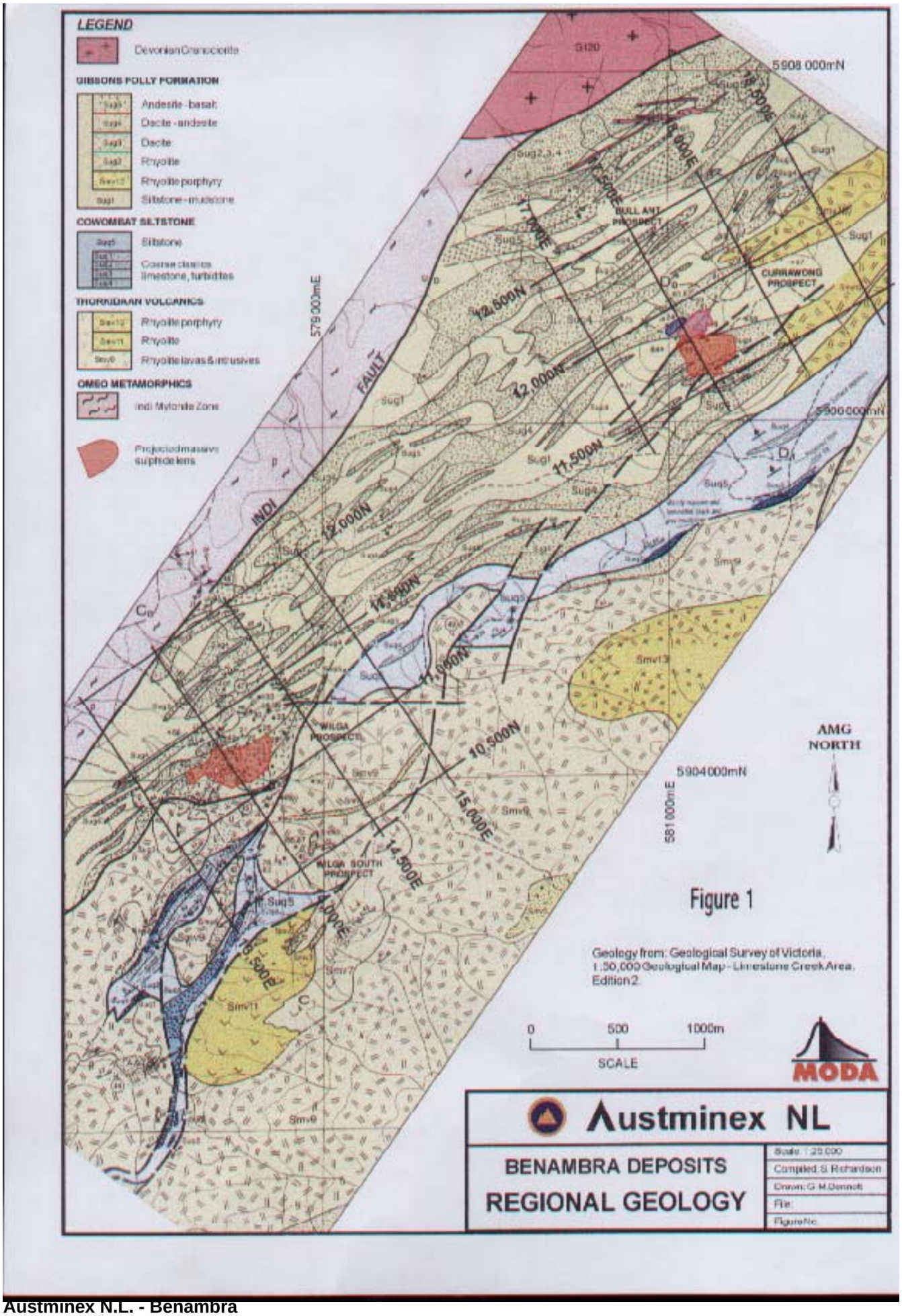
Massive sulphide mineralisation is composed of <1cm thick bands of pyrite, alternating with sphalerite-rich pyritic bands containing blebs of chalcopyrite and minor galena. Locally chalcopyrite can also occur as fine bands and result in chalcopyrite-rich zones up to 5m thick, Page (1989). Gangue minerals comprise less than 20% of the massive sulphide and are dominantly quartz, sericite, carbonate and magnetite, except in chalcopyrite rich zones where chlorite is the main gangue mineral.

Chalcopyrite rich “stringer” mineralisation comprises a significant portion of the lens (estimated from Cu/Zn ratio at around 20% of the pre-mining volume). It is composed of irregular veins, patches and massive chalcopyrite±pyrite in an intensely chlorite±silica-altered rock. The main occurrence of this mineralisation is in the central down-dip portion of the lens, where it interfingers up-dip with dominantly massive sulphide mineralisation. It also dominates the thin western end of the deposit. This style of mineralisation has gradational contacts with enclosing rocks suggesting a replacement origin, Allen and Barr (1990).

Rocks overlying and lateral to the mineralised lens are strongly quartz-sericite-altered with stringer and disseminated pyrite. This alteration extends up to 20m into the hangingwall and has been intersected in drill holes 400m down-dip from the main lens.

Immediately overlying and laterally equivalent to the massive sulphide lens is a complex sequence of dacite-andesite lava/shallow intrusive and associated hyaloclastite breccia, enclosed within background turbiditic sediment. Available drill-log data is not of sufficient detail or consistency to allow correlation of individual units in this complicated, highly variable sequence. An exception is a prominent basaltic lava/hyaloclastite breccia unit up to 80m thick, which is seen in the hangingwall over the entire deposit.

The immediate footwall of the Wilga deposit comprises a few centimetres to 20 m of highly disrupted sediment containing lenses and fragments of sandstone in a strongly cleaved, mudstone matrix. This unit is interpreted as a D_2 shear zone, which has separated the orebody from its original footwall, Allen (1987). Stratigraphically beneath the interpreted shear zone are Thorkidaan rhyolite lavas and associated volcaniclastics, which are overturned on the basis of younging directions.


LEGEND
Devonian Granodiorite
GIBSONS FOLLY FORMATION
Andesite - basalt
Dacite - andesite
Dacite
Rhyolite
Rhyolite porphyry
Siltstone - mudstone
COWOMBAT SILTSTONE
Siltstone
Coarse clastics limestone, turbidites
THORNIDAAN VOLCANICS
Rhyolite porphyry
Rhyolite
Rhyolite lavas & intrusives
OMEO METAMORPHICS
Indi Mylonite Zone
Projected massive sulphide lens
INDI FAULT
BULL ANT PROSPECT
CURRAWONG PROSPECT
WILGA PROSPECT
WILGA SOUTH PROSPECT
5908 000mN
5900 000mN
5904 000mN
579 000mE
581 000mE
12,500N
12,000N
11,500N
11,000N
10,500N
15,000E
14,500E
16,500E
AMG NORTH
Figure 1
Geology from: Geological Survey of Victoria, 1:50,000 Geological Map - Limestone Creek Area, Edition 2.
0 500 1000m
SCALE
MODA
Austminex NL
BENAMBRA DEPOSITS
REGIONAL GEOLOGY
Scale: 1:25 000
Compiled: S. Richardson
Drawn: G. M. Dennett
File:
Figure No:

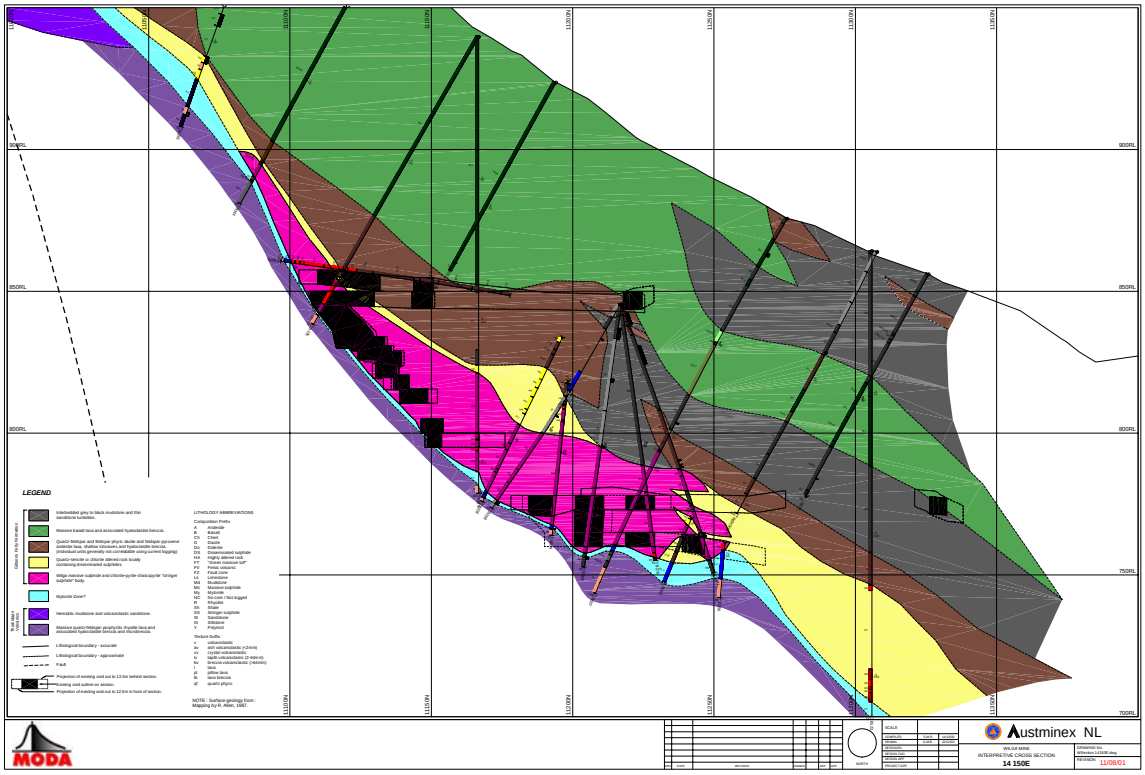


Figure 2 *– Interpretive Wilga cross-section at 14150E (for scale, note 50m grid)*

2.3.2 <u>Currawong</u>

The Currawong deposit comprises at least 5 massive sulphide lenses, 100 – 300m below surface. The lenses are shallow to moderately north dipping and are largely conformable to stratigraphy and the dominant cleavage. The main lens is now referred to as **M lens**, whilst the others are referred to as **A, B, J** and **K**[1] lenses (Fig.3 and Appendix II). **M lens** has a strike length of at least 325m, a down-dip extent of 350m and maximum thickness of 40m.

Due to complex structure and stratigraphy, the relationship between the lenses is not fully resolved, with previous workers suggesting either:

- Mineralisation occurs on opposite limbs of an isoclinal F2 syncline, disrupted by sub-vertical F_3 faults, or
- The different lenses represent F_3 fault repetitions of an originally single massive sulphide lens.

The latter, simpler interpretation is favoured by MODA and has been the basis for interpretation and modelling of the Currawong lenses. A broad, east-trending, sub-vertical F_3 fault/shear zone, referred to as the Currawong fault, is inferred to offset **M Lens** from **B Lens**. **J** and **K Lenses** are inferred to be portions of the main lens caught up in this structural zone and appear to be bounded by subsidiary faults. **A Lens** is offset from **B Lens** on a sub-parallel structure. Movement is inferred to be largely dip-slip, with north side up displacement.

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[1] Previous Currawong assessments by other companies and consultants used a slightly different lens naming system.

Mineralisation is dominated by massive to banded pyrite±sphalerite±chalcopyrite ±magnetite-bearing massive sulphide. Some parts of the lenses also contain significant galena, arsenopyrite, and electrum. In general, the Currawong mineralisation is finer grained, more massive and more pyritic than at Wilga and contains only minor chalcopyrite “stringer” mineralisation of the style prominent at Wilga. Gangue minerals are quartz, sericite, chlorite, carbonate and minor talc.

An envelope of strong hydrothermal alteration occurs up to 20m above and 30m below the massive sulphide lenses and is laterally continuous over large distances, at approximately the same stratigraphic level. The bulk of the alteration envelope is silica-sericite and contains disseminated and vein pyrite. Localised more to the lens margins and as internal waste, is strong to intense chlorite alteration, which often appears to post-date and replace quartz-sericite alteration. Chalcopyrite veins and local replacements are strongly associated with this alteration style.

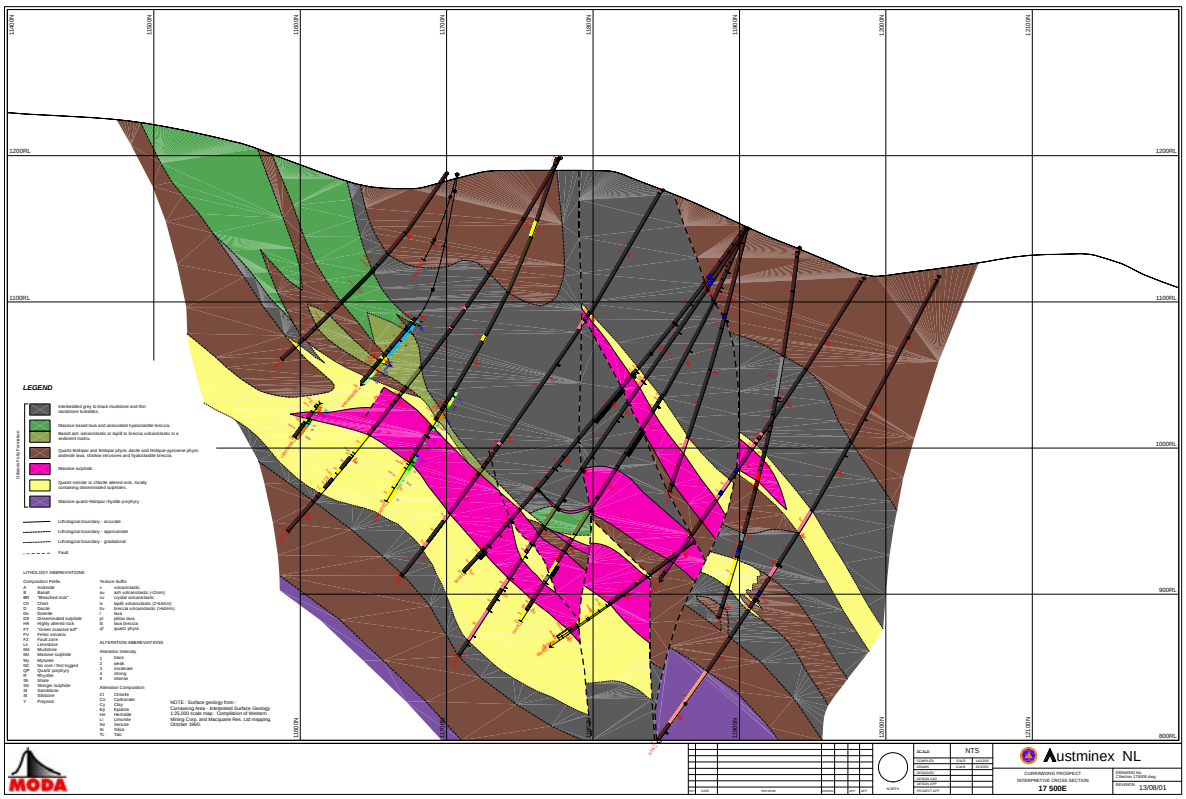

Figure 3 *– Interpretive Currawong cross-section at 17500E. All massive sulphide lenses are represented from A (at far right), B, J, K, to M (at left). For scale, note 100m grid.*

Lithologies present at Currawong are the same as at Wilga. Background sedimentation comprises mudstones and turbiditic sandstones. Intercalated with the sediments is a complex sequence of dacite - andesite and basalt lava, shallow intrusives and associated hyaloclastites. Unlike Wilga, the original footwall to mineralisation has not been sheared off, with mudstones and felsic volcanics conformably passing down into the Cowombat Siltstone, except where the Currawong rhyolite porphyry has intruded the base of the Gibsons Folly Formation.

3. AVAILABLE DATA

3.1 INTRODUCTORY CAVEATS

A large bulk of the geological work undertaken by MODA for this assessment was involved in the reconstruction of a digital database from old WMC/Macquarie/Denehurst data made available on CD by Austminex.

Whilst survey and assay data was in a reasonable state and required only minor amendments after validity checks, the digital geological data is scant and of dubious quality.

The main problem with the geological data is the lack of standard rocktype nomenclature and the tendency for drill log quality to deteriorate towards recent times.

Benambra has a long history of geological assessment over almost 25 years. Assessments have been undertaken by:

- Western Mining Corporation
- Rod Allen's major PhD study
- Macquarie Resources
- Denehurst
- several consultant firms

each with varying nomenclature, slants/specialties and quality of work. This has made the time-limited re-assessment by MODA rather difficult. Validity checking by MODA has also been made troublesome by the incomplete documentary archive. In addition, time constraints have not permitted the optimal re-logging of drillcore, which, in the long run, is the only way to adequately correlate the varying units of the host rocks enclosing the massive sulphides.

Nevertheless, some attempt was made by MODA to resolve lithological problems by reference to original hand-written drill logs with subsequent revision of the digital rocktype codes. This was a time-consuming exercise and only a partial solution. There still remain many instances of adjacent drillholes showing vastly different rocktypes for the same stratigraphic unit.

Despite the host rock problems, definition of the massive sulphide bodies was far less perplexing. In places, there were minor revisions made to the hangingwall or footwall positions, to accord with the available assay data.

There was a distinct paucity of structural data (e.g. faults) in the drill logs. Detailed underground mapping at Wilga by Denehurst, revealed faults with sometimes moderate displacements, that were difficult to correlate in 3D.

3.2 DATA PROCESSING

3.2.1 Mine grids

Wilga and Currawong are covered by two separate but similar, historic mine grid systems, derived from the original Western Mining exploration grid. These grids have been used by all previous workers and were also used for this resource assessment. An earlier MODA proposal to establish one global grid for the entire property has been deferred, pending a positive decision on the mine go-ahead.

A summary of the relationship of the mine grids to AMG is shown in Table 5.

TABLE 5
GRID RELATIONSHIPS

GRID	***AMG bearing of Mine Grid North***	***Mine Grid RL above AHD***[2]
Wilga	328° 14' 07"	173.20m
Currawong	330° 31' 09"	147.12m

3.2.2 Diamond drilling data

Introduction

Drill hole information used in the current resource estimate was derived from data gathered by the four companies that have undertaken drilling at Benambra since 1976. A summary of the period of involvement of each company is shown in Table 6.

TABLE 6
DRILLING OWNERSHIP

COMPANY	FROM	***TO***
Western Mining Corp. Ltd.	1976, Hole B001	1984, Hole B101
Macquarie Resources Ltd.	1988, Hole B103 and UW01	1990, Hole B182, and UW38
Denehurst Ltd.	1991, Hole B200	1996, Hole B298
Austminex NL	2000, Hole BERD0001	2001, Hole BEDD0037

The amount of drilling undertaken at each deposit is shown on Table 7. This table is a guide only and includes underground drilling and a subjective number of exploration holes in the vicinity of each deposit.

[2] To correct mine grid RL to Australian Height Datum (AHD), subtract the figures quoted.

TABLE 7
DRILLING STATISTICS

COMPANY	***WILGA***	***CURRAWONG***
Western Mining Corp Ltd	31 holes for 5,560m	40 holes for 13,061m
Macquarie Resources Ltd	82 holes for 6,403m	32 holes for 9,961m
Denehurst Ltd	77 holes for 5,983m	23 holes for 6,514m
Austminex NL	13 holes for 1,781m	26 holes for 6,073m
TOTAL	**203 holes for 19,727m**	**121 holes for 35,609m**

Drill hole collar surveying

Collar survey data was obtained from Denehurst archived data. All pre-Austminex holes in the vicinity of the deposits, appear to have a surveyed pick-up, but the quality of survey control is unknown.

All Austminex hole-collars were picked-up by licensed surveyors Crowther and Sadler Pty. Ltd., of Bairnsdale. During these collar pick-ups, some checks of old drill hole collar co-ordinates were undertaken. Most checks were within several centimetres of the original pick-ups but some early holes differed by 1-20m. Any anomalies were corrected within the current database.

Downhole survey data

Downhole survey data for pre-Austminex drilling was obtained from archived data. All downhole surveys appear to be obtained from a downhole camera, although some shorter holes have only a collar survey. Different methods of data recording were used by each company and are summarised in Table 8.

TABLE 8
DOWNHOLE SURVEYS

COMPANY	***DOWN-HOLE SURVEYS FROM***
Western Mining Corp. Ltd.	20m graph derived data
Macquarie Resources Ltd.	Approx. 20-30m spaced raw camera data
Denehurst Ltd.	Approx. 30m spaced raw camera data
Austminex NL	25m graph derived data

Austminex holes were surveyed during drilling, using an Eastman single shot camera, at nominal 30m intervals. During drilling of percussion pre-collars a survey camera was used inside the RC rods to gauge the amount of lift but hole azimuth could not be obtained. Upon completion, pre-collars were lined with PVC and dip and azimuth data obtained.

Survey cameras were provided by the drilling contractors. Consequently, several cameras were used over the course of the Austminex programme. The camera used during the entire 2001 programme was checked against a surveyed jig, set up

approximately along mine grid south, near the mine office. The check indicated no camera corrections were required.

Hole azimuth and inclination data were plotted against depth. The trend of hole deviation was established and any spurious (mainly azimuth) readings were discarded. 25m spaced data were read from the graph and entered into the survey database.

Geology

Digital data in the current database was reconstructed from Western Mining, Macquarie and Denehurst data, sourced from Denehurst archives, provided by Austminex. The archived geological data was incomplete and suffered from the lack of a standard nomenclature and is effectively restricted to lithology information. Time constraints did not permit the re-logging of core, although during the 2001 Currawong drilling programme, some check logging was undertaken of problem areas in old holes. In the long run the only way to adequately resolve many of the host rock conflicts will be to re-log some drill holes.

Lithology data from earlier drilling was made available with Denehurst lithology mnemonics. Structural and alteration data has never been transposed from the original Western Mining or Macquarie Resources logs and was therefore not available. Several generations of Western Mining logs and Macquarie Resources logs (original paper logs), were used to attempt to resolve common conflicts with surrounding holes.

Many Western Mining and Macquarie Resources holes were missing from both the Currawong and Wilga digital databases. To make the dataset as complete as possible, information for these holes was entered from original paper logs. However, lithology data from several holes, mostly Macquarie Resources drilling at Wilga (possible geotechnical holes), could not be found. Geological logs from Macquarie underground holes at Wilga were not available during modelling of the massive sulphide boundaries but were later found and entered into the database. They therefore appear on the provided geological cross sections.

Denehurst drilling data was originally recorded using the Datcol system, so that all data, including structural data, was available digitally. However, Denehurst logging contains abundant inconsistencies, such as lithological disagreements between adjacent drill holes. Since descriptive logs are non-existent, these inconsistencies usually could not be resolved.

Before Austminex drilling commenced, it was decided to log new core with a geological logging system based on that used at the Hellyer-Que River mines, by Aberfoyle Limited. A summary of logging codes used by Austminex is attached in Appendix III. Consequently, the original lithology digital database, comprising Denehurst codes, was converted to use the new Austminex geology codes. Where a Denehurst mnemonic was used with a specific local relevance, such as "*SS*" for *chalcopyrite-chlorite stringer sulphide*, this was retained in the new database.

The current digital geology database is effectively restricted to lithology data, although fault and alteration data from Austminex holes at Currawong were entered into separate datasets. A large amount of structural and alteration data exists on paper drill logs but has not been entered into any known database. This has hampered the current interpretation and should be addressed in any future work programme.

3.2.3 Austminex drilling procedures

Drilling programmes

13 holes for 1781m at Wilga and 26 holes for 6073m at Currawong were completed by Austminex between October 2000 and July 2001 (Figs. 4-5). To maintain a high drilling rate and to reduce costs, early holes were drilled using percussion pre-collars to the top of the zone of interest. Holes were then completed through the mineralised zone using an NQ or HQ tail. This initially proved a successful strategy, especially at Wilga where the lengths of pre-collars were generally less than 100m. At Currawong however, deeper depth to mineralisation required pre-collars of around 150-180m and deviation became a significant problem. This lead to some holes failing to hit the designed target and consequently percussion pre-collars were abandoned after hole BERD0027[3].

Two different drilling contractors were used during the Austminex programmes. Mineral Probe of Brisbane, were used for holes BERD001-9A and the pre-collars of holes BERD0010-12. Pre-collars were drilled using a Moroka RC rig and diamond tails drilled using a track mounted Longyear LF70. Diamond tails for BERD0010-12 and holes BERD0013-BEDD0037, were drilled by Cobar Drilling, of Rushworth, Victoria, using various drill rig types. Existing drill pads and tracks were utilised in general but when new pads were required, these were prepared by local earthmoving contractors.

Percussion chip and core processing

Chips from pre-collars were collected at one-metre intervals during percussion drilling, then washed and stored in standard percussion chip sample trays.

Drill core was transported to the Benambra mine site by four-wheel drive for processing by technicians and geologists. The following tasks were performed on the core:

- Trays washed to remove mud and drilling fluids
- Core runs measured and checked against core blocks. Discrepancies referred back to the drillers and any required adjustments made
- Metre marks placed on the core to assist in logging by the geologist
- RQD measurements recorded within each 1m interval
- Engraved aluminium tag showing hole, tray number and interval attached to tray

[3] Austminex hole names use the following prefixes: *BERD* for percussion pre-collars with diamond tails, and *BEDD* for all diamond holes.

- Tray photographed using a digital camera
- Core logged by geologist
- Mineralisation marked up for splitting
- Mineralisation cut using diamond saw
- Core stacked on palettes and transported to open core yard for storage
- Trays containing intersections stored in core shed to maximise security and minimise sulphide deterioration

Core photography

All Austminex diamond core was routinely photographed (wet and dry) using a digital camera. These have been archived on CD.

Geological logging

Before drilling commenced it was decided to log core from Austminex holes using a geological logging system based on that used at the Hellyer-Que River mines, by Aberfoyle Limited. An example blank logging sheet and summary of logging codes used is attached as Appendix III. At the time of writing, only lithology information from this logging has been entered into the Wilga database, whilst rock-type, alteration and structural data have been entered for Currawong.

Chips from percussion pre-collars were also geologically logged but due to the nature of the sample, many of the features evident in drill core could not be observed.

Sampling and assaying

None of the pre-collar percussion chips were used for assay samples.

All massive sulphide and base metal bearing stringer mineralisation was assayed. Nominal 1m sample intervals were used, with boundaries adjusted to coincide with geological contacts. Assaying continued several metres into sub-grade hangingwall and footwall rocks. Some broad low-grade stringer mineralisation was also assayed, using 1.5m or 2m sample intervals.

Core was split using a diamond saw on site. Initially, half core was dispatched for assay and half core retained. To provide material for use in future metallurgical testing, ¾ core from massive sulphide mineralisation from BERD002- onwards was retained and ¼ core sent for assay. Mineralisation surrounding the massive sulphide zones was split to ½ core as for earlier holes.

Amdel Laboratories Ltd. in Adelaide, South Australia, conducted all assaying of Austminex core. Samples were transported by truck from Benambra to Bairnsdale and on to Adelaide. Pulps and rejects were returned the same way for storage at the mine site.

239 samples from Wilga holes and 927 samples from Currawong holes, plus standards, were submitted for assay.
At Amdel, sample preparation comprised:

- jaw crushing of the total sample to 5-10mm
- 200g split pulverised to a nominal 90% passing 106μm
- modified aqua-regia digest
- multi-element ICP scan to determine Cu, Pb, Zn, Fe, As and Ag
- 50g Au fire assay with AAS finish
- air pycnometer SG

Each sample batch included one of four pulverised assay standards, prepared from Wilga massive sulphide ore, by Aminya Laboratories, Burnie.

A database of assay standards was established by submitting the four standards for multiple assays to:

- Aminya Laboratories, Burnie, Tasmania
- Analabs, Cooee, Tasmania.
- Australian Laboratory Services, Brisbane, Queensland

3.2.4 Massive sulphide interpretation method

After resolution of lithological coding issues, Datamine software was used to position the drillhole data in 3D space. Interpretation of the massive sulphide envelope for each lens was then undertaken on 25m-spaced, west-looking, cross sections using Datamine Guide software. The cross sectional perimeters of the massive sulphide envelope were then linked to create a 3D wireframe for each lens.

Most drill holes contained a single zone of mineralisation and correlation between holes was a simple matter. In some holes, (almost all are M Lens holes at Currawong) the intersection comprised more than one band of massive sulphide, separated by variably mineralised and altered host rock. Positioning the footwall and hangingwall in these holes became subjective and was done by connecting the lowermost or uppermost massive sulphide boundary that could be reasonably correlated to adjacent holes. Some internal waste therefore became included in the massive sulphide envelope. The alternative would have been to create a complex envelope of aereally extensive, thin, inter-fingering lenses of massive sulphide and host rock. With the current drill hole spacing this was considered an unrealistic interpretation. In both cases, thin intervals of stringer vein mineralisation, at above economic cut-off grade, were sometimes excluded from the massive sulphide zone and became part of the dilution envelope.

An extreme version of this problem occurred at Currawong, on section 17625E, in drillhole BEDD0031. The hangingwall of M Lens was placed at the top of several metres of strongly chlorite-altered but effectively un-mineralised material. This position was considered to be the “stratigraphic” equivalent of the hangingwall seen

in adjacent hole BEDD0033, on 17575E. The hangingwall position chosen also correlates well spatially with the top of the massive sulphide in other surrounding holes. It was considered more realistic to allow the stratigraphic modelling and kriging (section 4.6) to infer the shape of this internal waste rather than attempt to constrain it manually.

Lateral terminations of the massive sulphide envelope are subjective and based on overall geological interpretation at each deposit. Lenses are sometimes modelled terminating on interpreted faults and sometimes are inferred to lens out. The rate of lensing out is dependent on the last intersected thickness, distance to the nearest barren hole and geology of surrounding holes.

At Wilga, underground drill hole lithology data (UW1-38) was not available at the time of modelling. Assay data was available, so a combination of Cu, Zn (and when present, Fe assays) were used to interpret the location of the massive sulphide boundary, in these holes. Available underground mapping of massive sulphide contacts was also used to modify the cross section perimeters at Wilga.

4 RESOURCE ESTIMATION TECHNIQUE

4.1 MASSIVE SULPHIDE GEOMETRY

For the purposes of this assessment, it was decided that the Benambra Mineral Resources would be defined geologically by the massive sulphide envelope, regardless of grade. The massive sulphide bodies are easily recognisable and correlatable domains that display acceptable geological continuity from drillhole to drillhole. However, a proportion of this massive sulphide resource inventory will be below the calculated economic cutoff grade, and a small fraction of the “waste” (i.e. outside the massive sulphide envelope) will be above the cutoff. Nevertheless, it is industry practice to preferentially adopt a stable geological definition rather than an economic one that would be expected to vary over time.

4.2 ARCHIVED ASSAY DATA VALIDITY CHECKS

The raw assay data files provided by Austminex on CD were loaded into Datamine and routine validity checks were run.

The largest problem proved to be the different conventions used by previous owners regarding ‘missing’ assays. Some missing assays had been incorrectly coded as 0 (zero). These were satisfactorily resolved (without ready access to original assay documents) by careful examination of each drillhole’s assay record in turn. A few from/to mismatches were corrected.

The assay records provided had emanated from Denehurst’s Surpac database and were obviously output after some form of from/to merging process. Adjacent assay intervals often show duplicate assays, as would occur if the original assay intervals were merged with, say, different lithology intervals.Whilst this is not entirely satisfactory, it has not deleteriously affected the compositing process or altered the inherent statistical properties of the 2-metre composites.

Arsenic assays were found to be stored partially as percentages and partially in ppm. The ppm values were converted to percentages.

A few spurious high assays (e.g. 500.8 %Fe) that were obviously incorrect were set to ‘missing’, or set to averages of neighbouring values.

After original assay documents were located in the paper archive, a casual clerk was hired by MODA to input a considerable amount of additional Currawong Fe and As assays and density data, that was missing in the Denehurst database originally provided. The time resources available also permitted detailed checking and correction (including deletion of duplicate records) of all the Currawong assay data, at least for those holes intersecting massive sulphide.

The latest AMDEL assays from the new Austminex drilling were provided in digital format and therefore potential transcription errors were unlikely.

4.11 WILGA MINED-OUT VOIDS

MODA was supplied Denehurst Surpac files for the voids existing when mining terminated in 1996. These were converted to Datamine format by Surpac consultant David Shipp. There were many overlaps present, so it was essential to flag the mined-out block model cells, file by file.

The resulting mined-out model gives a mined massive sulphide tonnage of:

677,000 tonnes @ 3.88 SG, 6.0%Cu, 0.43%Pb, 6.6%Zn, 38 g/t Ag, 0.40 g/t Au, 30.0%Fe.

This tonnage contrasts with that reported mined by Denehurst (with 3% Zn grades assumed by A.B.Molinia to allow for lack of Zn reporting in early Cu zone mining):

955,607 tonnes @ 6.04%Cu, 8.68%Zn

The 40% tonnage anomaly is due to many factors, including:

- Denehurst void surveys do not include overbreak into pillars
- The Denehurst tonnage will include non-massive sulphide dilution
- The very latest mining excavations may not have been surveyed by the time the mine closed

The anomaly will only be satisfactorily explained when the mine is re-opened and survey checks can be made.

6. RESERVE ESTIMATION METHOD

The proposed mining methods are discussed in detail in a separate Mining Volume (Vol.II).

The mining method selected for both Wilga and Currawong is predominantly *Sub-level Open Stoping* with introduced waste backfill. A basic layout of 20m-wide open stopes separated by 10m-wide vertical pillars is planned. The stopes are taken to the full height and cross-sectional width of the economic mineralisation and alternate stopes are backfilled with waste rock to allow pillar recovery without major collapse or surface subsidence. A proper cave layout will allow the pillars to be mass blasted into the unfilled stope voids and drawn down using cave draw conditions under introduced waste fill.

At Wilga other techniques are required to allow mining around the current backfilled mining voids. The method of extraction of this residual and remnant ore is dictated by the ore geometry, ground conditions and shape of the existing mined voids. Both *Post Pillar Cut and Fill* mining and a longitudinal stope using *Avoca* style retreat mining are proposed.
At Currawong the flat lying western up-dip portion of the lens will also require *Post Pillar Cut and Fill* mining (stope MS450A). By starting at the base of the mining zone these areas can be mined efficiently from bottom to top, without leaving any intermediate crown pillars.

The planned stopes and pillars for both Wilga and Currawong, were screen-digitised using Datamine Guide software, as perimeter strings on 10m-spaced sections. The perimeters were linked to produce closed 3D wireframe volumes. For Wilga a 9%Zneq cutoff was used, but for Currawong an 8%Zneq cutoff was used to create more practical mining geometry.

To model likely overbreak dilution, the primary stope perimeter strings were expanded by 2m on exposed walls and backs. These expanded perimeters were also linked into 3D wireframes to represent the final stope shape after ingress of the unplanned dilution.

The dilution percentage for each primary stope therefore varies according to stope width – narrow stopes attract a higher dilution percentage, as would normally be expected. The Datamine Guide software then reports the tonnes and grade of both massive sulphide resource and dilution, for both the planned stope, and, including the unplanned dilution. The actual tonnes and grade of unplanned dilution was therefore calculated by subtraction. Pillars were all assigned a constant 30% dilution at zero-grade with 85% extraction recovery.

Wilga stopes and pillars that contained more than 50% Inferred resource were discarded (JORC Code rules prohibit mine planning over Inferred resource). Therefore, some stopes retained will contain minimal Inferred resource, because it is simply impractical to design stopes exactly on the Indicated-Inferred boundary. At Currawong, the stopes and pillars lying over the Inferred boundary were cut exactly on the boundary.

7. ORE RESERVES TABULATION

Although comprehensive, detailed tables in spreadsheet format have been prepared for this Ore Reserve assessment, only summarised versions are presented in this report. Summary diagrams of the reserves are shown on Figures 33 and 34.

The Measured Resources that fall within planned stopes at Wilga have been downgraded to Probable Reserve status due to current uncertainties in cost estimates for mine closure environmental work.

7.1 WILGA

The Ore Reserves of the Wilga deposit as estimated by MODA are:

TABLE 29
WILGA ORE RESERVES

Stope/ Pillar	Reserve Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%Fe	%Zneq
R1401	Probable	19	5.8	0.13	1.0	21	0.04	16.8	15.6
R1404	Probable	41	2.7	0.32	3.6	27	0.27	24.3	10.4
S1405U	Probable	152	2.8	0.42	5.4	35	0.46	23.5	12.4
S1405L	Probable	272	2.2	0.46	6.8	35	0.43	27.9	12.4
R1409	Probable	35	3.2	0.54	5.5	28	0.29	18.4	13.5
S1414	Probable	443	2.9	0.63	7.3	40	0.52	29.1	14.4
R1416	Probable	42	1.3	0.58	6.8	35	0.42	17.6	10.0
S1417	Probable	11	4.7	0.12	5.2	24	0.54	26.3	16.9
S1418	Probable	17	4.2	0.30	3.7	29	0.53	28.3	14.2
P1419	Probable	111	1.7	0.40	5.1	30	0.43	23.0	9.3
S1420	Probable	206	2.0	0.52	6.0	37	0.61	28.7	11.0
P1421	Probable	83	1.5	0.44	4.9	29	0.53	21.5	8.5
S1422	Probable	122	1.7	0.58	5.6	35	0.72	24.9	9.8
P1424	Probable	43	1.4	0.52	4.6	28	0.59	18.6	7.9
S1425	Probable	37	1.5	0.61	5.3	33	0.65	21.4	9.1
TOTAL	**Probable**	**1636**	**2.4**	**0.51**	**6.1**	**35**	**0.50**	**25.9**	**12.0**
TOTAL	**Stopes**	**1261**	**2.5**	**0.53**	**6.5**	**37**	**0.53**	**27.4**	**12.6**
TOTAL	**Pillars**	**375**	**2.0**	**0.43**	**4.8**	**29**	**0.42**	**21.0**	**9.9**
TOTAL	**(rounded)**	**1600**	**2.4**	**0.51**	**6.1**	**35**	**0.50**	**25.9**	**12.0**

N.B. apparent summation anomalies are due to rounding

7.2 CURRAWONG

The Ore Reserves of the Currawong deposit as estimated by MODA are:

TABLE 30
CURRAWONG ORE RESERVES – A LENS

Stope/ Pillar	Resource Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%As	%Fe	%Zneq
AS460	Probable	93	1.5	1.2	6.2	50	1.9	0.73	29.6	10.0
TOTAL	**Stopes**	**93**	**1.5**	**1.2**	**6.2**	**50**	**1.9**	**0.73**	**29.6**	**10.0**
TOTAL	**Pillars**	**0**								
TOTAL	**Probable**	**93**	**1.5**	**1.2**	**6.2**	**50**	**1.9**	**0.73**	**29.6**	**10.0**

TABLE 31
CURRAWONG ORE RESERVES – B LENS

Stope/ Pillar	Resource Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%As	%Fe	%Zneq
BS500	Probable	62	2.0	0.60	3.6	34	1.0	0.50	35.6	8.6
BS590	Probable	143	1.9	1.5	4.7	45	2.0	1.3	32.9	9.5
BS610	Probable	47	1.6	1.8	4.3	54	2.8	0.97	28.3	8.3
TOTAL	**Stopes**	**253**	**1.9**	**1.3**	**4.3**	**44**	**1.9**	**1.1**	**32.7**	**9.0**
TOTAL	**Pillars**	**0**								
TOTAL	**Probable**	**253**	**1.9**	**1.3**	**4.3**	**44**	**1.9**	**1.1**	**32.7**	**9.0**

TABLE 32
CURRAWONG ORE RESERVES – J LENS

Stope/ Pillar	Resource Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%As	%Fe	%Zneq
JS500	Probable	24	2.7	0.29	2.0	25	0.61	0.36	36.7	8.7
TOTAL	**Stopes**	**24**	**2.7**	**0.29**	**2.0**	**25**	**0.61**	**0.36**	**36.7**	**8.7**
TOTAL	**Pillars**	**0**								
TOTAL	**Probable**	**24**	**2.7**	**0.29**	**2.0**	**25**	**0.61**	**0.36**	**36.7**	**8.7**

TABLE 33
CURRAWONG ORE RESERVES – K LENS

Stope/ Pillar	Resource Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%As	%Fe	%Zneq
KS490	Probable	34	1.3	1.9	7.4	56	2.4	1.6	30.3	10.6
KP505	Probable	39	1.0	1.5	6.1	45	2.1	1.4	22.2	8.7
KS520	Probable	61	1.0	1.5	6.4	51	2.2	1.3	26.3	9.0
KP535	Probable	17	0.88	1.4	5.1	45	1.6	0.61	21.2	7.3
KS550	Probable	43	1.2	1.5	5.9	54	1.5	0.44	26.3	8.8
TOTAL	**Stopes**	**139**	**1.1**	**1.6**	**6.5**	**53**	**2.0**	**1.1**	**27.3**	**9.3**
TOTAL	**Pillars**	**56**	**0.97**	**1.5**	**5.8**	**45**	**1.9**	**1.1**	**21.8**	**8.2**
TOTAL	**Probable**	**195**	**1.1**	**1.6**	**6.3**	**51**	**2.0**	**1.1**	**25.7**	**9.0**

TABLE 34

CURRAWONG ORE RESERVES – M LENS

Stope/ Pillar	Resource Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%As	%Fe	%Zneq
MP445	Probable	63	1.3	0.49	4.5	33	0.45	0.08	20.4	7.9
MS450A	Probable	183	1.8	0.50	3.8	35	0.96	0.10	29.5	8.4
MS460	Probable	213	1.7	0.74	6.1	43	0.68	0.14	28.8	10.4
MP475	Probable	134	1.5	0.66	4.9	38	0.66	0.12	23.9	8.7
MS490	Probable	301	2.3	0.78	5.6	47	0.95	0.11	30.5	11.2
MR500	Probable	61	2.1	0.52	4.0	30	0.46	0.12	18.9	9.2
MP505	Probable	163	2.0	0.57	4.1	33	0.88	0.08	24.5	9.1
MS520	Probable	316	2.6	0.64	5.0	36	1.5	0.12	31.4	11.6
MP535	Probable	190	2.1	0.38	3.5	23	1.3	0.11	25.3	8.7
MS550	Probable	377	2.6	0.44	3.9	29	1.3	0.14	32.2	10.4
MS550U	Probable	108	2.0	0.57	4.5	37	0.71	0.12	29.6	9.5
MP565	Probable	169	2.2	0.38	2.8	24	0.76	0.13	25.1	8.3
MP565U	Probable	45	1.8	0.47	3.9	34	0.44	0.12	25.2	8.5
MS580	Probable	313	2.6	0.67	3.5	35	1.2	0.18	31.0	9.9
MS580U	Probable	112	2.1	0.49	4.2	35	0.59	0.15	26.4	9.3
MP595	Probable	146	2.0	0.51	2.7	28	0.99	0.17	25.4	7.6
MP595U	Probable	70	1.8	0.44	3.2	27	0.53	0.13	20.1	7.7
MS610	Probable	198	2.4	0.74	3.9	36	1.4	0.28	33.6	9.8
MS610U	Probable	167	2.4	0.87	4.1	39	1.3	0.40	29.3	10.2
MP625	Probable	46	1.8	0.72	3.5	28	1.2	0.30	25.8	8.1
MP625U	Probable	82	2.0	0.95	3.6	35	1.7	0.60	24.7	8.7
MS640	Probable	96	1.8	1.6	5.6	48	2.3	0.68	30.1	10.0
MS640U	Probable	154	2.4	0.91	4.0	39	1.4	0.55	32.3	10.0
MP655	Probable	44	1.1	1.9	5.7	53	2.9	0.89	22.5	8.5
MP655U	Probable	43	1.9	0.51	2.6	29	0.83	0.31	25.0	7.4
MS670	Probable	75	1.3	2.7	7.0	70	3.9	1.3	28.8	10.3
MS670V	Probable	21	2.2	0.46	3.5	35	1.5	0.28	33.5	9.0
TOTAL	**Stopes**	**2633**	**2.3**	**0.75**	**4.5**	**38**	**1.3**	**0.24**	**30.7**	**10.2**
TOTAL	**Pillars**	**1257**	**1.9**	**0.58**	**3.7**	**30**	**0.98**	**0.19**	**24.1**	**8.4**
TOTAL	**Probable**	**3890**	**2.2**	**0.69**	**4.3**	**36**	**1.2**	**0.22**	**28.6**	**9.6**

TABLE 35

CURRAWONG ORE RESERVES - TOTAL

Unit	Resource Category	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%As	%Fe	%Zneq
A Lens	Probable	93	1.5	1.2	6.2	50	1.9	0.73	29.6	10.0
B Lens	Probable	253	1.9	1.3	4.3	44	1.9	1.1	32.7	9.0
J Lens	Probable	24	2.7	0.29	2.0	25	0.61	0.36	36.7	8.7
K Lens	Probable	195	1.1	1.6	6.3	51	2.0	1.1	25.7	9.0
M Lens	Probable	3890	2.2	0.69	4.3	36	1.2	0.22	28.6	9.6
TOTAL	**Stopes**	**3142**	**2.2**	**0.85**	**4.6**	**39**	**1.4**	**0.36**	**30.7**	**10.1**
TOTAL	**Pillars**	**1313**	**1.9**	**0.62**	**3.8**	**31**	**1.0**	**0.23**	**24.0**	**8.4**
TOTAL	**(rounded)**	**4500**	**2.1**	**0.78**	**4.4**	**37**	**1.3**	**0.32**	**28.7**	**9.6**

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AUSTMINEX NL

BENAMBRA PROJECT

MINE REOPENING AND DEVELOPMENT

SUMMARY REPORT

OCTOBER 2001

Prepared by

A.B.Molinia BE (Hons) FAusIMM CPMin
(McArthur Ore Deposit Assessments Pty Ltd)

Assisted by

G.J.McArthur PhD FAusIMM CPGeo MMICA MSEG
S.M.Richardson BSc (Hons) MAusIMM (McArthur Ore Deposit Assessments Pty Ltd)
R.Hazeldene Grad.Dip.App.Geol MAusIMM (Mindeco Pty Ltd)

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1 BENAMBRA MINE RE-OPENING STUDY

1.1 Introduction and Previous History

Previous mining was concentrated at Wilga, where the mining rate was up to 0.3 million tonnes per annum, and the broken ore was loaded from stopes into trucks for haulage to the surface stockpile. The mining method was based on a mechanised post-pillar cut and fill technique, with 5m x 5m pillars and up to 10 m wide rooms. Stope fill was obtained from development and from surface stockpiles of material generated during road works.

The current re-opening plan is to mine Wilga ahead of Currawong, at the rate of 0.6 million tonnes per annum (MTA). Wilga will provide the first 2.75 years of production, followed by Currawong for an additional 7.25 years.

The initial years of production mining will be carried out using underground mining contractors. The contractor will be expected to supply all manning, equipment and services to carry out the mining operation at the Benambra Mine Site.

1.2 Safety and Training

1.2.1 General

An integral part of the mining management strategy will be the development and implementation of a comprehensive training and safety plan. This will include systems for:

- Health and safety management
- Training
- Development and implementation of Safe Operating Procedures
- Work place exposure monitoring
- Exposure and risk reduction
- Health monitoring and improvement
- Emergency response
- Public health and safety

1.2.2 Emergency Response Plan

An emergency response plan will be developed for the site, containing action plans for a comprehensive list of potential occurrences. The plan will assign responsibility for actions to designated officers. An “Emergency Duty Card” set will compliment the plan to ensure that each designated officer is provided with a concise list of duties that cover the breadth of any foreseeable scenario.

1.2.3 Contractors Responsibility for Safety

Contractors will be required to have and maintain, a comprehensive safety management plan for their areas of responsibility. This will be consistent with the Company’s own plan. Contractors will be required to comply with the following requirements:

- Conform with the companies’ policy and procedures;
- Conduct pre-employment health assessments of their employees;
- Have all employees attend site induction programmes;
- Train and competency assess personnel, including equipment and plant operators;
- Implement risk assessments and control strategies;
- Have written safe operating procedures for all key tasks;
- Hold regular safety meetings;
- Conduct workplace inspections and equipment checks;
- Provide appropriate safety and protective equipment;
- Maintain a Health and Safety Management Plan;
- Report and investigate all unplanned events; and
- Maintain safety performance statistics.

1.3 Mine Management and Manning

The plan is to employ a competent mining contractor who is capable of providing a full service including planning, supervision and geology. The intention is to minimise the overlap of employees, whilst still ensuring a quality outcome. The mine will provide its own representative to monitor that the work is carried out to good professional standards and that the contractor performs the work in accordance with the plan, statutory requirements and meets or exceeds production and cost targets. A mining workforce totalling 49 has been estimated.

1.4 Mine Access

1.4.1 Wilga

Re-opening of the mine will consist of the following sequence of events:

- Uncover and if necessary re-secure the portal entrance.
- Muck 50 metres of backfilled waste from the decline.
- Rehabilitate the Decline (scale and re-support)
    - Stage 1 - Portal to RL 820, approximately 265 metres of decline.
    - Stage 2 – After re-establishment of main ventilation system, RL 820 to RL 755, approximately 600m of decline.
- Re-establish the Main Underground Ventilation System
    - Stage 1 – Muck out the back-filled main vent rise and re-establish the main surface exhaust fan. (RL 820 to surface).
    - Stage 2 – Re-establish ventilation from RL 755, mine bottom, to RL 820.
- De-water the mine to RL 755. Testing of the mine water level through a surface diamond drill hole that intersects the RL 755 level indicates that the mine contains little, if any, water.
- Re-establish underground mining and development at the rate of 600,000 tonnes per annum.
- Haulage of ore from the portal re-handle area to the Plant Site.
- Transport and placement of fill materials to the underground voids.

1.4.2 Currawong

Access to the Currawong orebody will be via a decline driven at a gradient of 1:7. The decline portal has been located to the west of the orebody at 11650N, 17200E co-ordinates and at RL 1125 (Currawong Grid).

The decline is positioned to start in a fresh outcrop of rock providing a competent face in which to establish the portal. The orebody levels will be a series of take off drives located along the legs of the decline at a regular vertical interval of 25 metres. The orebody will be developed top down with mining starting in the upper levels whilst the lower levels are developed. The decline size is set at 5.0 m wide x 5.5m high to suit an equipment size capable of efficient development and mining at the rate of 600,000 tonne per annum.

Concurrently with the decline advance ventilation shafts will be developed. It is intended to have three shafts, two-intake (R1 and R3 Rises) and one main exhaust (R2 Rise). The R1 intake shaft will be equipped with an escape manway and provide a fresh air base on each working level and an alternate means of egress other than the decline. The R3 Rise is required to provide ventilation to the A-K Lens

1.5 Orebody Characteristics Impacting On Mining

Rock Quality Designation (RQD) data was collected from all Austminex core and is available digitally, together with data from Denehurst Wilga holes. RQD data from all Macquarie Resources holes and Denehurst Currawong holes is thought to exist on paper but appears to be unavailable in digital form.

Using the available digital dataset, the length-weighted mean RQD value at each deposit is shown in Table1. These crude values are based on samples that are not evenly distributed, or corrected for the effect of varying core diameter.

Table 1

DEPOSIT	***ORE – MEAN RQD***	***HOST ROCK – MEAN RQD***
WILGA	82 (n=334)	66 (n=1288)
CURRAWONG	78 (n=451)	67 (n=3693)

High RQD values confirm the overall visual impression from drill core that ground conditions at Wilga and Currawong are generally competent. During the Austminex programmes no drilling problems were encountered due to faulting or broken ground (although both do occur locally). Core breaks relate variously to jointing, cleavage, bedding or faults, depending on lithology and structural setting. In volcanics, phyllosilicate-altered rocks and massive sulphide, core-breaks are generally due to jointing and cleavage. Sediments are dominated by breaks along well-developed bedding and cleavage planes.

At Wilga the immediate hangingwall to mineralisation interface comprises silica-sericite altered dacite and sediments, with generally well developed cleavage and/or bedding planes. The immediate footwall to massive sulphide interface comprises a strongly cleaved “mylonite” zone of variable thickness and low RQD, separating the massive sulphide from a more competent rhyolite footwall. A strong cleavage is also present in parts of the massive sulphide lens, associated with chlorite rich “stringer mineralisation”.

At Currawong, hangingwall conditions are similar to Wilga. The footwall however is generally comprised of bedded and cleaved silica-sericite±chlorite altered sediment.

It is envisaged that diamond drilling on a maximum spacing of 20m will ultimately be required, with some closer spaced drilling in areas of geometric complexity.

1.6 Mining Operation - Wilga

1.6.1 Selection of Mining Methods

The emphasis by Denehurst on high grade and specific ore types required the adoption of a flexible and selective mining method, and therefore a post-pillar mechanised cut and fill method was adopted.

By necessity further mining must fit around the current backfilled mining voids and outside the non-recoverable envelope. The extraction of the residual and remnant ore is dictated by the ore geometry, ground conditions and the mined voids. This results in a number of different mining methods, adapted to the particular geometry and characteristics of the ore block to be mined.

Proposed mining methods for the residual Wilga ore are:

Sublevel Uphole Retreat Stoping
Sublevel Downhole Retreat with Backfill (Avoca Style)
Post Pillar Cut & Fill
Primary Sublevel Open Stopes (with Backfilled Waste)
Pillar Recovery by Mass Blast into Primary Stopes.

Where applicable sub level stoping is usually preferred to cut and fill mining for its:

Safety aspects
Lower mining costs
Higher productivity
Higher ore body recovery

1.6.2 Design Criteria and Cut-off Grade

Economic modelling was used to determine a cut-off grade which yields a rate of return above 10%. This resulted in a cut-off grade of 9% Zn equivalence based on the formula Zn%+2.5xCu%. Gold and silver contribute less than 5% to the revenue and therefore were excluded from the formula.

This cut-off, when applied to the Wilga deposit, results in regular shapes and blocks of ore suited to productive mining methods. The ore generally decreases in grade towards the hanging-wall and this lower grade material is excluded from the stoping blocks. Attempts were made to further high grade by using higher cut-off grades. These attempts resulted in shapes that lacked continuity between sections, and did not provide regular mining blocks that could be mined at reasonable cost.

In effect high grading above 9% Zn equivalence reduces the mining tonnage and number of work places such that a production rate of 600,000 tonnes per annum cannot be achieved. Modelling has indicted that the gain in grade by mining at lower

rates does not compensate for the higher costs associated with the reduced rate, consequently it is a less preferred option.

Using the above, the resource within the planned mining blocks is 1.59 million tonnes @ 2.7% Cu, 7.0% Zn, 40 g/t Ag, 0.57 g/t Au (13.75 Zn eq).

1.7 Mining Operation – Currawong

1.7.1 Selection of Mining Methods

Unlike Wilga, this orebody is undeveloped and its width, geometry and size make it amenable to a number of mining methods. The overall dip is not particularly favourable to open stoping, but the overall geometry and width is such that open stopes can be used. The footwall ore will be mucked from footwall draw points on the access drill levels and the hanging wall ore can report down to the lower main extraction levels.

Previous mining proposals by Denehurst have favoured “Post Pillar Cut and Fill” mining. This method has some attraction in that it is selective and can be adapted to flatter dips. But unless mining starts at the base of the deposit the method requires remnant crown/sill pillars between the levels, hence as previously described for Wilga the overall recovery never achieves the quoted 85%.

Post Pillar mining requires extensive working under freshly exposed backs, and can be categorized as slightly more risky than other methods. These hazards can be managed but alternate methods such as sublevel stoping substantially reduce this exposure.

The Currawong orebody will allow a reasonably efficient sub level open stoping method. The planned method is similar to that proposed for Wilga. Primary vertical open stopes of 20 metre widths separated by 10 metre wide pillars.

1.7.2 Design Criteria and Cut-off Grade

As for Wilga a cut-off grade of 9% Zn equivalence was initially applied to the Currawong deposit, but unfortunately this did not result in regular shapes showing continuity from section to section. These resultant narrow shapes were not suited to low cost mining and the lack of continuity would make any type of mining extremely difficult and high cost.

Due to this lack of continuity and regular mining shapes, a cut-off of 8% Zn equivalence was modelled. This resulted in regular shapes by including medium grade ore between and adjacent to the higher grade areas. The average grade of material within the 8% Zn cut-off averaged greater than 9% Zn equivalence, and the resulting shapes were continuous between sections and allowed the use of relative low cost bulk mining methods. Therefore the 8% Zn equivalence was adopted as the designed cut-off grade.

The mineral resource within the planned mining stopes is 4.26 million tonnes @ 2.4% Cu, 5.0% Zn, 42 g/t Ag and 1.4 g/t Au (11% Zn eq).

1.7.3 Main Mining Method

The majority of the deposit will be mined in a similar manner to Wilga Zone C. The initial 20metre primary stopes are to be separated by a 40metre pillar-stope-pillar block. Prior to mining the pillar-stope-pillar combination the initial stopes will need to be tight filled to the backs. This means that the ten metre pillars are only fully exposed on one side at any time during mining, and back support is never fully reliant on pillars but on a combination of pillars and filled stopes (pillar/filled stope/pillar).

Once the alternate stopes separated by the “pillar-stope-pillar” block have been mined and filled, extraction of the intermediate stopes within the pillar/stope/pillar set can occur. Once extraction of this stope is complete it will become the void for the mass blast of the two pillars. The draw point arrangement is planned to give a suitable layout for a controlled cave draw of the broken ore. The blasted stope essentially fills the void and therefore no surface subsidence can occur until the pillar ore is drawn down. As described in the Wilga section this is similar to a core and shell pillar recovery technique.

To prevent surface subsidence, rises will be broken through to the surface, above the centre of the mass blasted material. Waste fill will be introduced via the rises to control the void and prevent major collapse or surface subsidence.

Dilution/Recovery

Dilution recovery for the primary stopes and pillar will be similar to Wilga. On average waste dilution in these primary stopes is estimated at 12% for a total extraction recovery of 95%. In the pillars, 85% of the blasted mass recovered with 30% dilution at zero grade has been adopted.

1.8 Mine Infrastructure and Services

1.8.1 Wilga

1.8.1.1 Road & Earthworks

As most of the existing features for the mining of Wilga are in place and in good working order, it is envisaged that only minor earthworks will be undertaken prior to the recommencement of mining activities at the Benambra site. The main haul road from Waxlip Spur to the Wilga underground mine, is 5metres wide (minimum) and has regular passing bays. The road is in good condition and requires only grading to bring it back to service as a heavy vehicle haul road.

1.8.1.2 Services

At Wilga the underground mining contractor will be locating most of his facilities at the mine portal in a layout and manner similar to that approved for the previous operation.

Vehicles will undergo servicing at the Wilga Portal Workshop. Provision will be made to trap and collect all waste oils and materials for either recycling or sending back to an approved disposal facility.

1.8.1.3 Mine Equipment

The following is a likely list of equipment to be provided by the mining contractor.

Drilling and Support

Development will use a twin boom electric hydraulic jumbo (Atlas Copco H145 or equivalent), 1-off.

Production will require an electric hydraulic long hole drill rig capable of drilling up to 45metre 104 mm diameter blast holes (Atlas Copco Simba or equivalent), 1-off.

Support will use a dedicated support unit such as a Tamrock robolter, 1-off.

Loading & Haulage

Loading equipment will consist of 5 mx3 Load Haul Dump units (Caterpillar Elphinstone R1700's or equivalent), 2-off for development & production and 1-off for backfill.

Haulage up the 1 in 7 decline will require a nominal 40 tonne 4 wheel drive underground truck such as the Toro 35D, or Elphinstone D40, 3-off for 0.6 million tonnes per annum.

Service Vehicles.

A heavy-duty service vehicle such as an Elphinstone IT28 will be required for service work and as a charging and/or support unit, 1-off.

Other

Main ventilation fan (150 mx3/sec) electric powered 150 kW.
Auxiliary Ventilation fans (2 stage x 45 kW per stage)

1.8.1.4 Ventilation

It is proposed that the ventilation circuit used during initial mining operations between 1992 and 1996 be used when mining re-commences. This involves using the 60m shaft located at the southern extremity of the adit as the return airway.

Fresh air will enter the mine via the main decline and be split to flow through the various levels of drives. Effective stope ventilation is achieved throughout the mining areas through panel crosscuts to the ventilation drive and exhaust shaft.

An intake airway/escapeway (Figure 4) has been designed for the South Eastern end of the orebody. This will provide fresh air intake direct to the bottom of the mine and an escape route located in a fresh air intake.

Exhaust ventilation will be provided by the main fan mounted on the 60m shaft located at the western end of the mine adjacent to the adit. Auxiliary ventilation for development will be provided by forced ventilation through standard ventilation duct.

1.8.1.5 Emergency Escapeway

The escape man-way has not been designed in detail but consists of ladder-ways designed to appropriate standards in 60 degree rises connecting the working levels and then from the RL 850 level through to the surface.

1.8.1.6 Underground Communication System

A modern leaky feeder system will be used for mine communication and traffic control. Procedures will be developed and all employees will need to be trained in the system before proceeding underground.

1.8.1.7 Explosive Storage and Usage

The contractor will be required to supply explosive magazines meeting the relevant legislative requirements and obtain the appropriate approvals for the installations. At Wilga they will be located at a suitable surface site, probably on the track past the portal, or underground. There will be separate containers for Detonators, Packaged Explosives, and Packaged ANFO. Containers will be separate by sufficient distance or protected by mounds or barriers such that no unforeseen fire or explosion can be propagated to another container. The appropriate restricted zone will be applied to keep personnel outside any danger area. It is not intended to manufacture explosives on site. But if a contractor wishes to mix his own Anfo, then he will need to obtain the appropriate approvals and construct the facilities to the appropriate standards.

Vehicles for the underground transport of explosives will be appropriately marked and provided with separate containers to prevent transmission of explosions between explosive groups.

Due to the possibility of sulphide dust explosions, all firing will be conducted from the surface. Physical checks (tag boards) will be combined with underground inspection and physical barriers to insure that the mine is cleared and secured before blasting.

Comprehensive procedures will be required for post blast re-entry checks, particularly in the light of the possibility of sulphide dust explosions.

1.8.1.8 Mine Dewatering

Pumping

Testing of the mine water level through a surface diamond drill hole that intersects the RL755 level (base of mine) indicates that the mine contains little, if any, water.

Pump capacity of approximately 5 l/sec will be required to cope with the estimated natural seepage of approximately 3 l/sec and the introduced drilling and service water of approximately 2 litre/sec.

Water collection underground will be simple short decline sumps with flygt pumps. Slimes from these sumps will be mucked regularly into back filled stopes. The small quantities will have little impact on stope fill. The main underground pumps will be

located on the RL755 level and will pump to the surface at RL872 a vertical distance of 117 metres

A settling tank or sump and treatment facility will be required to allow the water to be re-cycled underground or used for dust suppression on the haul roads. The slimes from the settling sumps will be removed by a suction pump into suitable containers for transport either to the mill or the tailings treatment facility, or back underground into abandoned areas of the mine where they will have no impact on stability or mine drainage.

1.8.1.9 Mine Fill

Waste rock material will be required for underground fill, both to provide support and a working platform for cut and fill mining, and to fill voids and provide support for open stoping. Cemented rock fill or cemented paste fill is not economically viable for the Benambra Mine.

At least four different sources of waste rock material will be used.

- Development waste - this is the preferred material for underground fill, as it is already pre-mined and does not require any additional surface land disturbance.
- Excess waste material from surface earth works, namely re-establishing the Wilga portal site and the light vehicle access road to Currawong.
- Quarried Material - the current plan is to incorporate a surface quarry into the tailings storage area such that the quarry will provide additional tailings storage capacity and minimise the height and extent of the final tailings dam.
- Mill tailings – Unconsolidated tailings cannot be used for underground fill in areas adjacent to active underground workings. But once a particular area of the mine is worked out and there is no danger of an uncontrolled mud rush into any active workings, it may be possible to top up the residual voids with tailings fill. This will not take a large volume of tailings but it will be useful in filling residual voids where it is difficult to place rock fill. This is unlikely to be an economic option for Wilga, as the cost of the tailings placement pipeline would exceed the benefits of disposing of a small quantity underground.

Fill placement will be by direct trucking to the underground voids or by surface fill passes. Where it is possible and economic (C Block), waste material will be directly placed into stopes through waste passes. Where this is not possible the waste will be re-handled into the stopes.

A waste pass system has been designed for “C Block”. This involves trucking the waste approximately 70 metres down the decline and 50 to 90 metres along the 850 level, to ore passes designed to directly fill the C Block stopes and pillars. An ejector

tray truck will be used to tip directly into one of three passes designed for this purpose.

1.8.2 Currawong

1.8.2.1 Road & Earthworks

Two roads are planned for the commencement of mining activities at the Currawong deposit, the light vehicle track from McCallums South and the haul road from the portal to the Waxlip Spur Plant.

The existing access track will need to be re-activated for access to the portal. This requires the removal of the material stacked on the road as part of the site re-vegetation programme. This material will be stockpiled for future re-vegetation, and any material not suited for this purpose will be used for underground fill at Wilga.

The Haul road (figure 6) will be suitable for heavy vehicle transport of ore to the Mill located at the Waxlip Spur site. The road (approximately 2 km in length) will be constructed using conventional ripping and dozing. Small areas may require drill and blast.

1.8.2.2 Services

The Services currently in place at the Waxlip Spur Plant will be fully utilised for the Currawong mine. There will be minimal facilities available at the mine protal.

1.8.2.3 Mine Equipment

Equipment will be similar to that required for Wilga.

1.8.2.4 Ventilation

Based on a similar equipment fleet the ventilation requirements for Currawong will be similar to Wilga.

The ventilation system will consist of the main decline and two independent vent rise systems connecting to the surface. The eastern rise system will be the main exhaust system for the mine. With the western system providing an alternate fresh air intake and escape manway. This allows fresh air to be directed to one end of a working level and flow through that level to the exhaust system at the other end. In addition fresh air will enter the mine via the main decline and be split to flow through the various levels of the mine. The rises will be constructed on an on-going basis, as

each level is established 60 degree rises will be excavated to connect to the rise system on the level above.

1.8.2.5 Emergency Escapeway

The western rise system will be down casting and serve as the alternate escapeway to the main decline. The escape manway has not been designed in detail but will consist of ladder-ways, to the appropriate standards, in 60 degree rises connecting the working levels. The section from the upper level to surface may be vertical raise-bored rises, equipped with modular ladder ways as used at other mines around Australia.

On each level a steel fireproof ventilation door will be provided so that in an emergency the rise can be sealed from the level and serve as a fresh air refuge. The mechanisms and workings of this system will be defined in the emergency response plan.

1.8.2.6 Explosives

As per Wilga, explosive storage facilities will be installed to the required standard. Initially during the development phase these facilities will be surface located, on one of the tracks around the site. The facilities will be isolated from regular traffic and contain a buffer zone to prevent damage to persons and facilities from any unforseen explosion.

1.8.2.7 Mine Dewatering

Mine water, including drilling and wash down water will gravitate via the footwall drives to the appropriate collection sumps at intervals along the decline.

Previous predictive work by Golder & Associates indicated inflow rates of 12 L/s at Wilga. This was revised to 3.2 to 10.0 L/s in August 1988. In effect water in flow to Wilga was less than 3 L/s. Therefore at Currawong it can be expected that, after the initial inflows from creating the stope voids, water inflows of less than 10 L/s will become the norm. A Kinhill report of 1987 suggests that the mine will produce 1 to 6.5 L/s of ground water, after the initial higher inflows associated with the drainage of water bearing structures.

These types of inflow can be handled by a simple system of staged submersible pumps, capable of pumping 2% coarse solids to the next sump through 150 mm victaulic piping, at a minimum rate of 15 L/s, to prevent sedimentation. Float controls will regulate the flow. The water will be pumped to a portal settling and polishing tank where it will be either recycled for underground or pumped to the Waxlip Spur ore treatment plant.

This initial development pumping will be completed using off-the-shelf submersible units, with smaller units as decline face pumps.

The long term system will be designed by pumping specialists based on the flow rates encountered, and may involve pumping by a bore hole and pipe line direct to the mill for incorporation into the integrated water management system.

1.8.2.8 Mine Fill

The main fill placement system will be by surface fill passes. Where it is possible and economic, waste material will be directly placed into stopes through surface passes. Where this is not possible the waste will be re-handled into the stopes.

A system of 5 passes (figure 6) has been designed to feed directly into the M Lens stope/pillar mass blast voids. These passes will also be used to fill the primary stopes. The passes will be constructed to first service the primary stopes, with the waste from the base of the passes trammed, by LHD, directly into the upper levels of the primary voids. In the long term the rises will provide direct tipping into the stope/pillar mass blasts, except for the lower MS640 stope, which will be broken into the upper MS640 stope. This stope is connected to a surface rise that allows topping up of the combined voids.

Fill for the cut and fill room and pillar mining will come from development activities, therefore the production rate from this area will be matched to the underground waste produced.

The sources of fill have been discussed previously, with the main source for Currawong being material mined from an open cut behind the tailings dam.

1.9 Development Schedule

Wilga

The development schedule is based on achieving a production rate of 600,000 tonnes per annum. Development at Wilga totals 4200 metres for 1.6 million tonnes (390 tonnes/metre). This is a better yield than Currawong as extensive use is made of current development. Pre-production development of 580 metres is required to make sure that sufficient ore is available to start and maintain the required production rate.

Currawong

Total development of around 15,100 metres is required for the Currawong deposit.

This includes 2000 metres of decline development, 1400 metre of ventilation rises and 1000 metre of fill passes. Development is scheduled at a peak of 2800 metres per year, dropping to less than 2000 metres per year (production year 6). This rate can be comfortably achieved by a single development jumbo, with residual capacity to mine an additional 1000 metres per year of room and pillar stoping. Which is required for the AS450A stope.

Table 2 – Development Schedule

Development Schedule												
Year	**2001/02**	**2002/03**	**2003/04**	**2004/05**	**2005/06**	**2006/07**	**2007/08**	**2008/09**	**2009/10**	**2010/11**	**2011/12**	Total
Wilga												
Capital	580											580
Operating		1750	1750	120								3620
Subtotal	580	1750	1750	120								4200
Currawong												
Decline			300	600	600	500						2000
Level				1700	1500	1500	1400	1400	1300	1200	700	10700
Vert - Vent				380	180	160	160	360	160			1400
Vert - Fill					200	200	200	200	200			1000
Subtotal			300	2680	2480	2360	1760	1960	1660	1200	700	15100
Capital			300	2000	1100	1010	640	630	425	240		
Operating			0	680	1350	1350	1120	1330	1235	960	700	
Total Development Metres	580	1750	2050	2800	2480	2360	1760	1960	1660	1200	700	19300
Operating Development												
	Metres	1750	1750	800	1350	1350	1120	1330	1235	960	700	
Development Cost $/metre 2150	$000,s	3762.50	3762.50	1720.00	2902.50	2902.50	2408.00	2859.50	2655.25	2064.00	1505.00	26542
Capital Development												
	Metres		300	2000	1100	1010	640	630	425	240		
Development Cost $/metre 2500	$000,s		750.00	5000.00	2750.00	2525.00	1600.00	1575.00	1062.50	600.00		15863

1.10 Mine Production Schedule

The mine production rate has been estimated at 600,000 tonnes per year, as this is probably the highest production rate that could be sustained from the Currawong deposit considering its size of 4.5 million tonnes and width of 10 to 30 metres. This gives a 10-year mine life based on the current reserve of 6.1 million tonnes.

Wilga is the highest grade and most profitable, it is also easier to bring into full production, as there is already an existing decline. Therefore it is logical to bring Wilga into production first and follow with Currawong.

As much as possible the mining has been scheduled to mine the higher grade in the early years. This ability is more restricted at Currawong, where the grade difference through the orebody is not that pronounced. Sensible development scheduling dictates that mining commences as soon as possible rather than wait until the mine is fully developed. This means that the upper levels are brought into production before the lower levels of the mine containing slightly higher grades.

Table 3 – Production Schedule

Benambra Mine - Production Schedule												
Years		2002/03	2003/04	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12	Total
Wilga												
tonnes		550000	600000	480000								1630000
- %Cu		2.21	2.59	2.22								2.4
- %Pb		0.54	0.51	0.49								0.5
- %Zn		6.65	6.17	5.35								6.1
- g/t Ag		36	36	33								35
- g/t Au		0.53	0.50	0.48								0.5
Currawong												
tonnes				120000	600000	600000	600000	620000	640000	640000	640000	4460000
- %Cu				2.44	2.04	2.23	2.45	2.43	1.66	2.06	1.67	2.1
- %Pb				0.87	0.92	0.72	0.67	0.67	0.94	0.67	0.83	0.8
- %Zn				4.13	4.47	4.71	4.07	4.77	4.52	3.50	4.71	4.4
- g/t Ag				39	41	41	33	37	39	30	39	37
- g/t Au				1.32	1.37	1.09	1.41	1.24	1.30	1.27	1.13	1.3
Metallurgical Recoveries												
% Zn to Zn Conc		85	85	84	80	80	80	80	80	80	80	
% Cu to Copper Cor		83	83	83	83	83	83	83	83	83	83	
% Ag to Copper Cor		31	36	33	24	26	35	31	21	33	21	
% Au to Copper Cor		15	14	15	20	16	21	18	19	19	16	
Concentrate Production												
Zinc	- tonnes	62183	62947	51302	42078	44364	38345	46379	45395	35157	47283	475434
	- % Zn	50.00	50.00	50.20	51.00	51.00	51.00	51.00	51.00	51.00	51.00	
Copper	- tonnes	40432	51535	44672	39088	42757	46990	48093	33862	42008	34164	423602
	- % Cu	25	25	25	26	26	26	26	26	26	26	
	- g/t Ag	150	150	150	150	150	150	150	150	150	150	
	- g/t Au	1.1	0.8	1.3	4.3	2.4	3.8	2.9	4.8	3.7	3.4	

1.11 Mining Capital and Operating Costs

Due to the need to minimise capital the preferred approach is to utilise mining contractors for the Benambra Mine.

Mining costs are based on budget mining unit costs provided by contractors in response to a “Request for Budget Underground Mining Costs” issued by Austminex to a number of contracting companies in November 2000. Three responses to the request were obtained and these form the bases of the mining cost estimates.

Where appropriate these rates have been compared to mining costs at other operations and were adjusted where the contract costs did not appear to be in line with costs achieved elsewhere in similar circumstances.

1.11.1 Wilga Capital Start-up Costs

Costs included in re-starting the mine consist of:

- Contractor Mobilisation
- Uncover and re-secure the portal entrance.
- Muck 50 metres of backfilled waste from the decline.
- Rehabilitate the Decline (scale and re-support)
    - Stage 1 - Portal to RL 820 approximately 265 metres of decline.
    - Stage 2 – After re-establishment of main ventilation system, RL 820 to RL 755 approximately 600m of decline.
- Re-establish the Main Underground Ventilation System
    - Stage 1 – Muck out the back-filled main vent rise and re-establish the main surface exhaust fan. (RL 820 to surface).
    - Stage 2 – Re-establish ventilation from RL 755 mine bottom to RL 820.
- De-water the mine to RL 755.
    - Testing of the mine water level through a surface diamond drill hole that intersects the RL 755 level indicates that the mine contains little, if any, water.

The total estimate for re-opening the Wilga mine is $1.2 Million.

1.11.2 Wilga Pre-production Capital Development

In order to produce at the rate of 600,000 tonnes per year it will be necessary to do a base level of pre-production development, sufficient to have at least two main stoping areas in production at the time of mill start up. It will also be necessary to have a small stockpile in place for mill commissioning work and as insurance that the mine will not delay the project start up.

The mine schedule requires a total of 580 metres of development before mill start up. This consists of 200 metres of declining plus 120 metres of level development to develop the Zone C Stopes and some 110 metres of level development to access the Zone B post pillar cut and fill stopes. In addition to this access development to the Zone B stopes it is planned to carry out 150 metres of stope development within the Zone B ore body, this will result in approximately 9000 tonnes of ore being stockpiled prior to mill start up.

The cost estimate for this pre-production development is:

Decline 200 metres @ $2500/metre	= $ 500,000
Level development 380 metres @ $2150/metre	= $ 800,000
Total Cost	=$1,300,000

1.11.3 On-going Capital Development and Operating Development Costs

1.11.3.1 Wilga

The total development estimate for the Wilga mine is 4200metres and of this, 580 metres are scheduled as pre-production development. The residual 3620metres is planned to be developed as part of the ongoing mine operation. Because of Wilga's short life (2.5 years) this development will be treated as operating.

Total development costs are:

Metres 3620 @ $2150/metre = $7.78 million.

This equals $4.78 per tonne for the 1.63 million tonnes to be mined.

1.11.3.2 Currawong

The development schedule for Currawong is shown in table 6.
Total development is 15,100 metres for a total of 4.5 million tonnes.

The split between capital and operating development is determined according to the following definitions.

Capital development is defined as:

- Main decline access
- Main ventilation and escape way rises and associated development
- Main Backfill Rises
- Main level access cross cuts and drives which are expected to be in use for a number of years.

Operating development includes all draw points and drill drives that are directly associated with the immediate production of ore, plus some level drives and accesses to ore blocks that will be mined within 12 months of being developed.

The net effect of this definition is that some 6375 metres at $15.94 million ($3.59/tonne) is considered capital development and the residual 8725 metres at $18.76 million ($4.22/tonne) is included in operating costs.

1.11.4 Operating Costs

Operating costs have been estimated from budget estimates received from three independent contractors (Barminco, Byrnecut and UME) in response to a "Request for Budget Underground Mining Costs" circulated in November 2000. The contractors submitted estimates based on a detail plan for the reopening, development and mining of the Wilga deposit. Costs have been averaged where appropriate and adjusted where they appear not to agree with known industry costs.

The project costs have also been distributed to a number of other contractors, namely: Mancala, Clough, Theiss, and Roche. The responses indicate that the overall costs are competitive but could be achieved under favourable circumstances. Clough and Mancala suggested that due to the fine profit margins they would prefer to work under alliances based on a costs plus formula, structured to take into account performance and quality.

1.11.4.1 Summary of Wilga unit operating costs:

	$/tonne mined
Contractors Site Overheads, Services and Power	3.00
Drilling and Blasting Cost	6.80
Load & Haul (Production)	4.06
Back load and Place Backfill	1.45
Stope Support	1.13
Technical & Supervision	1.04
Geology & Delineation Drilling	0.84
Total Underground Production Cost	18.32
Operating Development	4.78
Total Underground Operating Cost	23.10
Surface Haulage to Mill	3.50
Total Mining Costs to Mill	26.60

Surface haulage costs are based on estimates from Bonney Fox using second hand trucks available at the time of the quotes.

1.11.4.2 Summary of Currawong unit costs:

	$/tonne
Contractors Site Overheads, Services and Power	3.00
Drilling and Blasting Costs	4.70
Load & Haul (Production)	6.00
Back load and Place Backfill	1.11
Stope Support	0.93
Technical & Supervision	1.00
Geology & Delineation Drilling	1.00
Total Underground Production Cost	17.74
Operating Development	4.22
Total Underground Operating Cost	21.91
Surface Haulage to Mill	2.00
Total Mining Costs to Mill	23.96

Figure No 1 – Wilga Longitudinal Projection

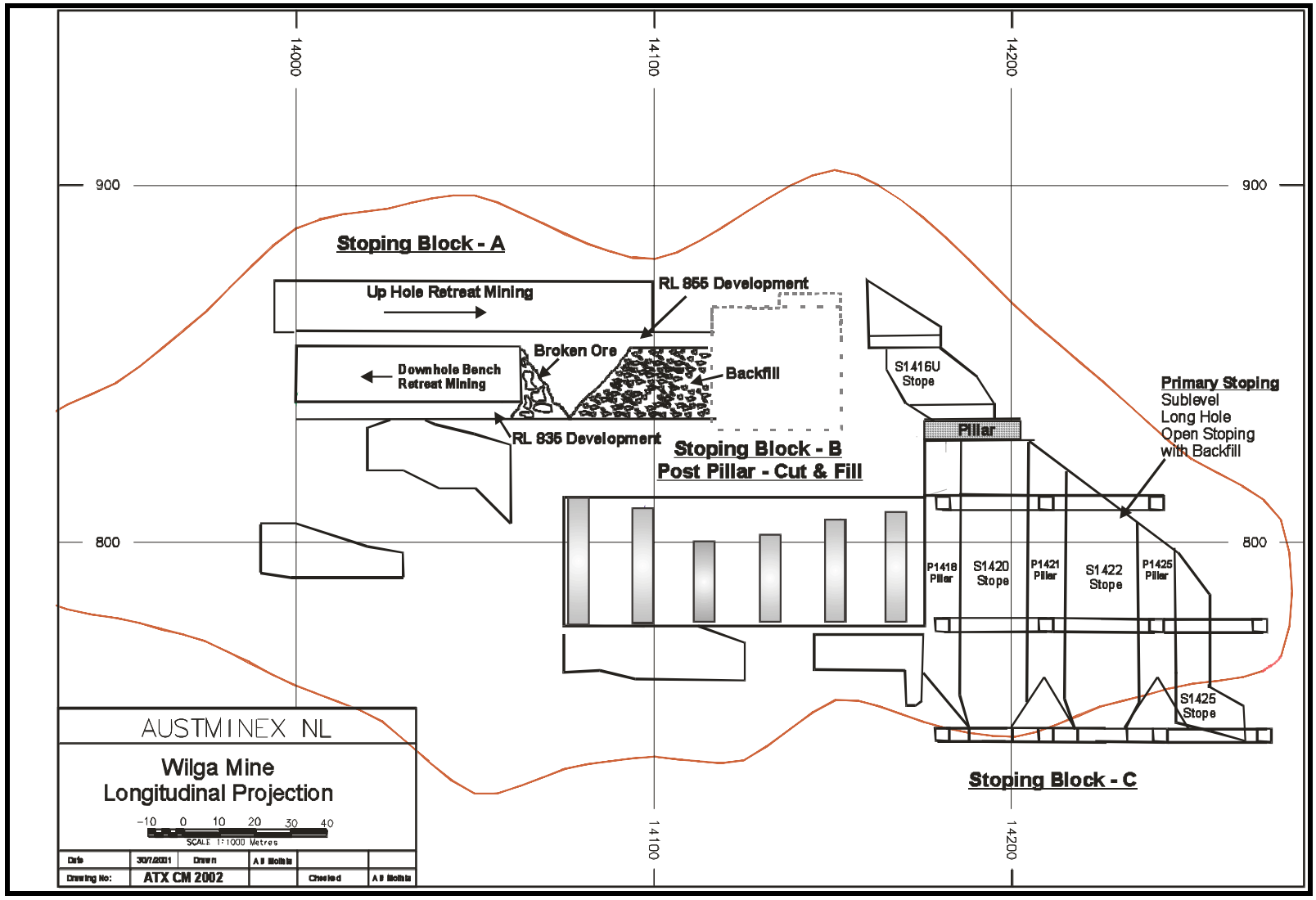


Figure No 2 – Wilga Stoping Block A – Conceptual Mine Design

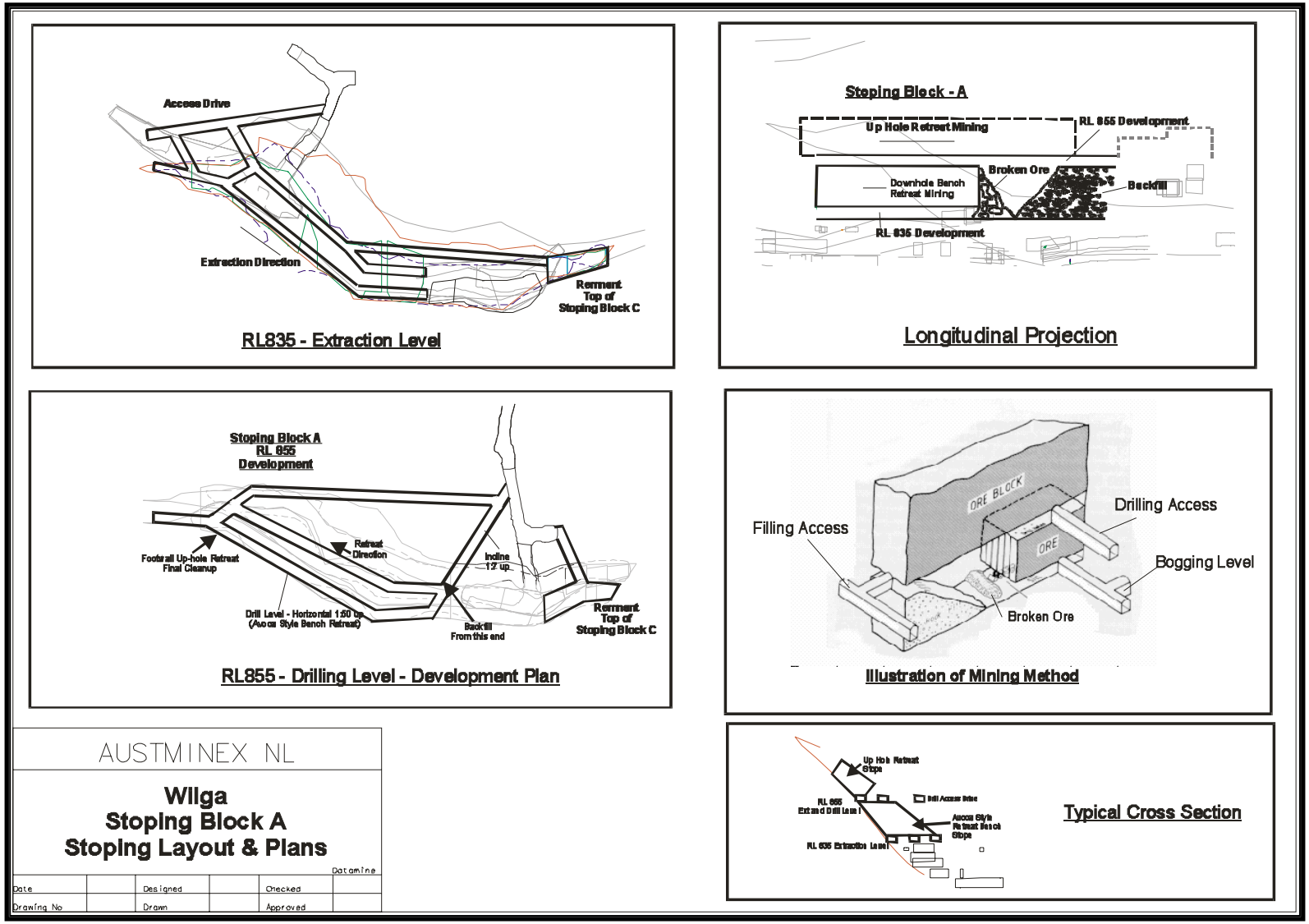


Figure No 3 – Wilga Stoping Block B – Conceptual Mine Design

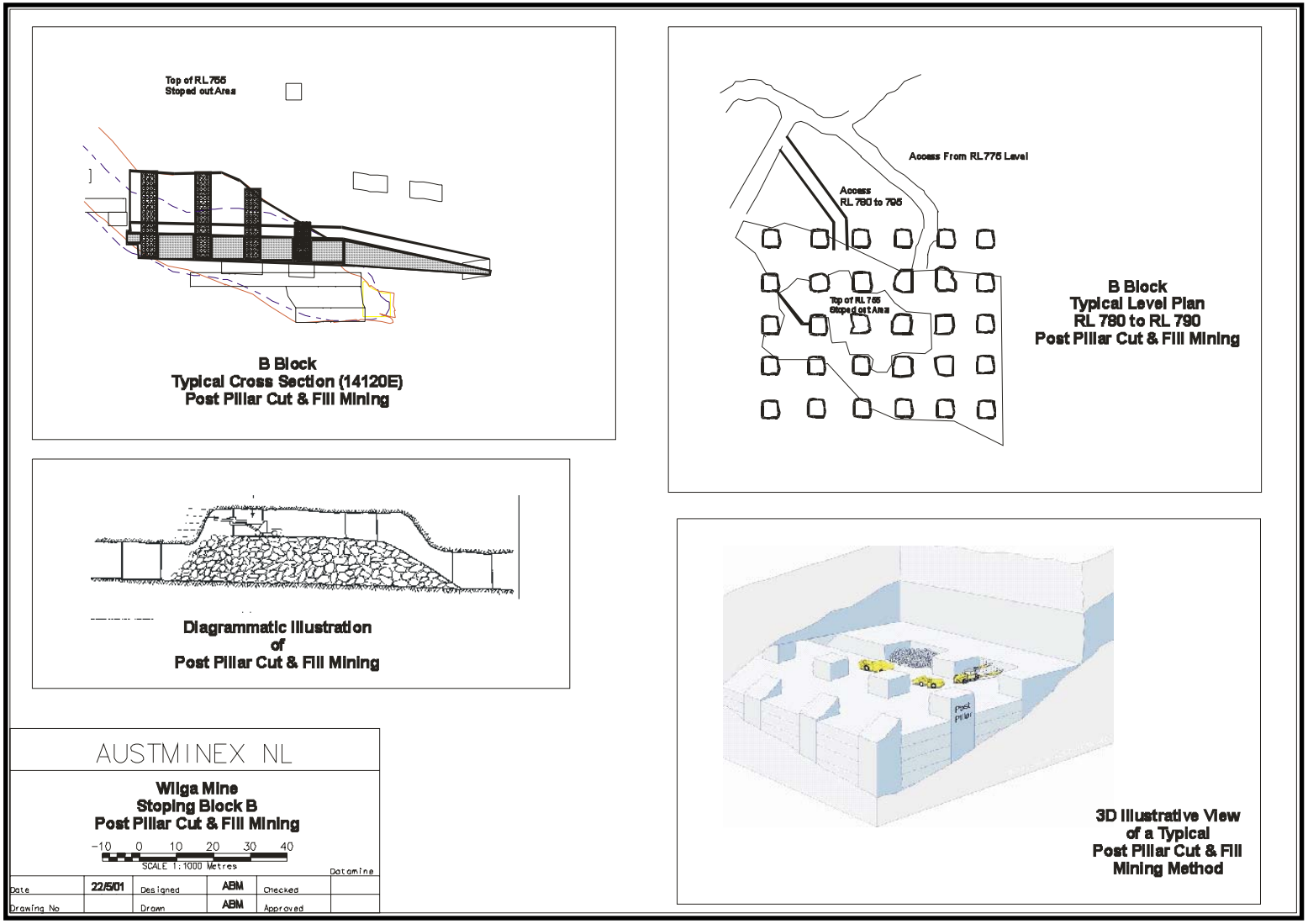


Figure No: 4 - Wilga Stoping Block C – Conceptual Mining Plan

Sublevel Open Stopes with Waste Fill and Pillar recovery by Mass Blast and draw under Waste Fill

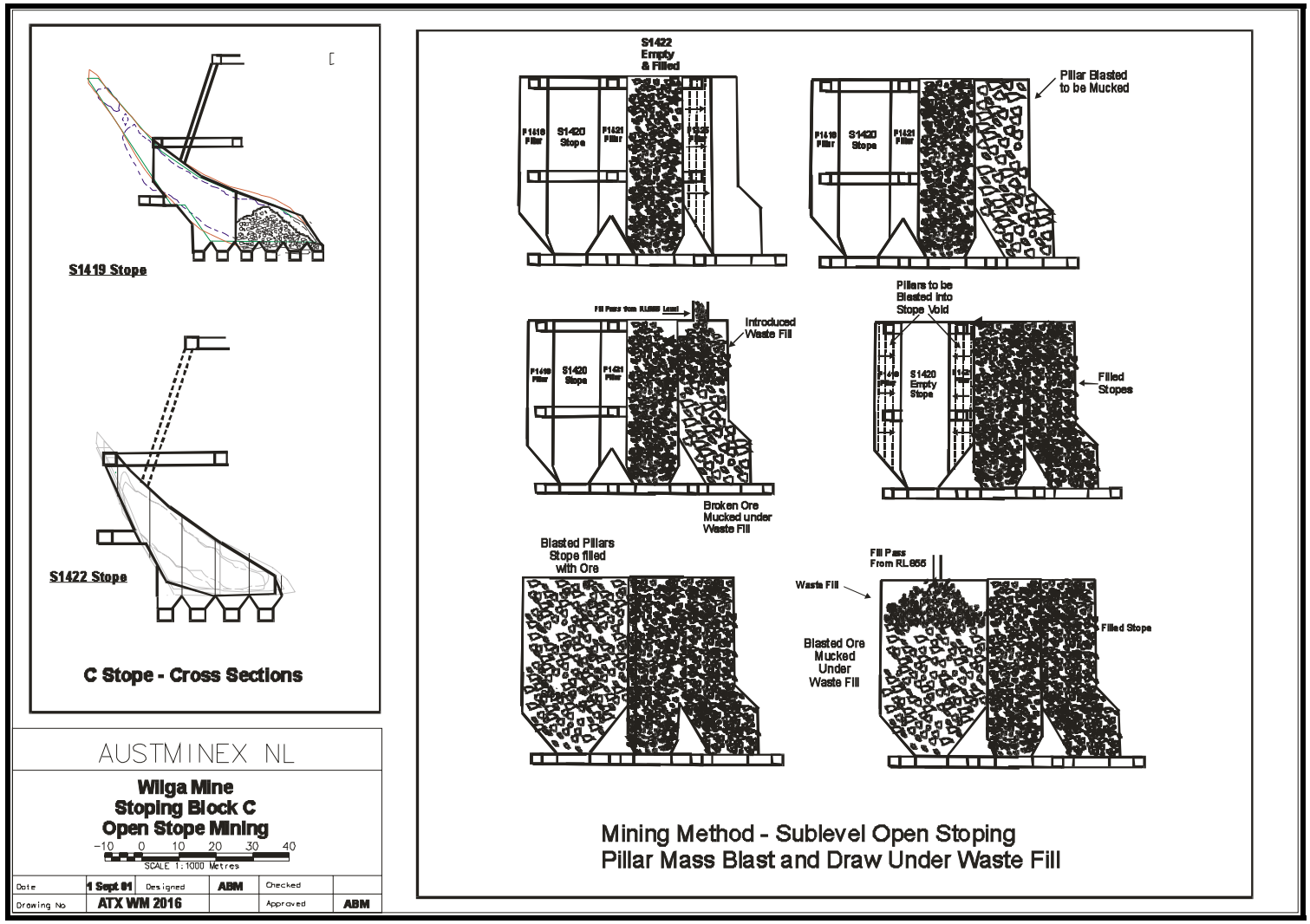


Figure No: 5 - Wilga Mine Ventilation Diagram

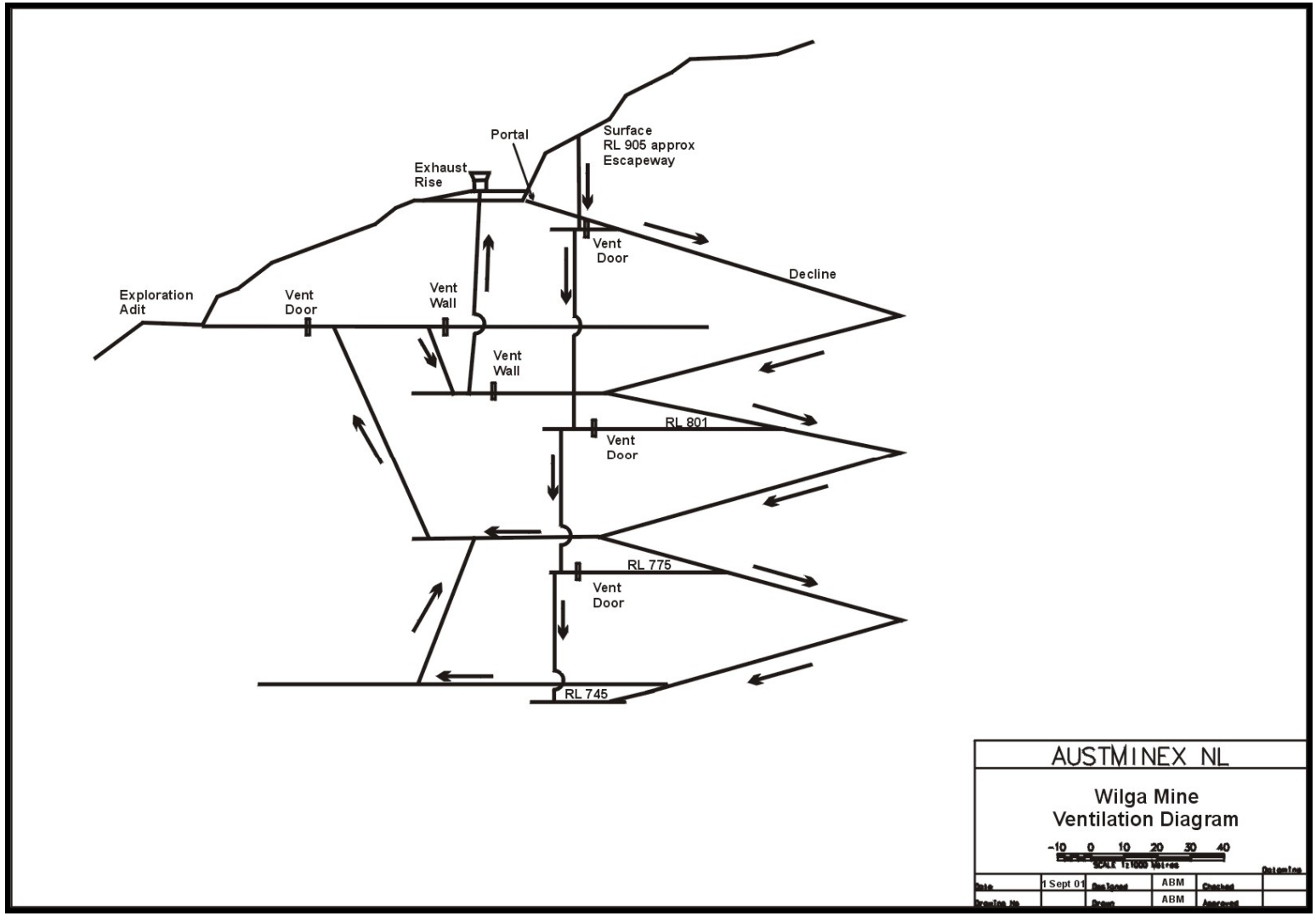


Figure No 6 - Currawong Portal and Decline Plan

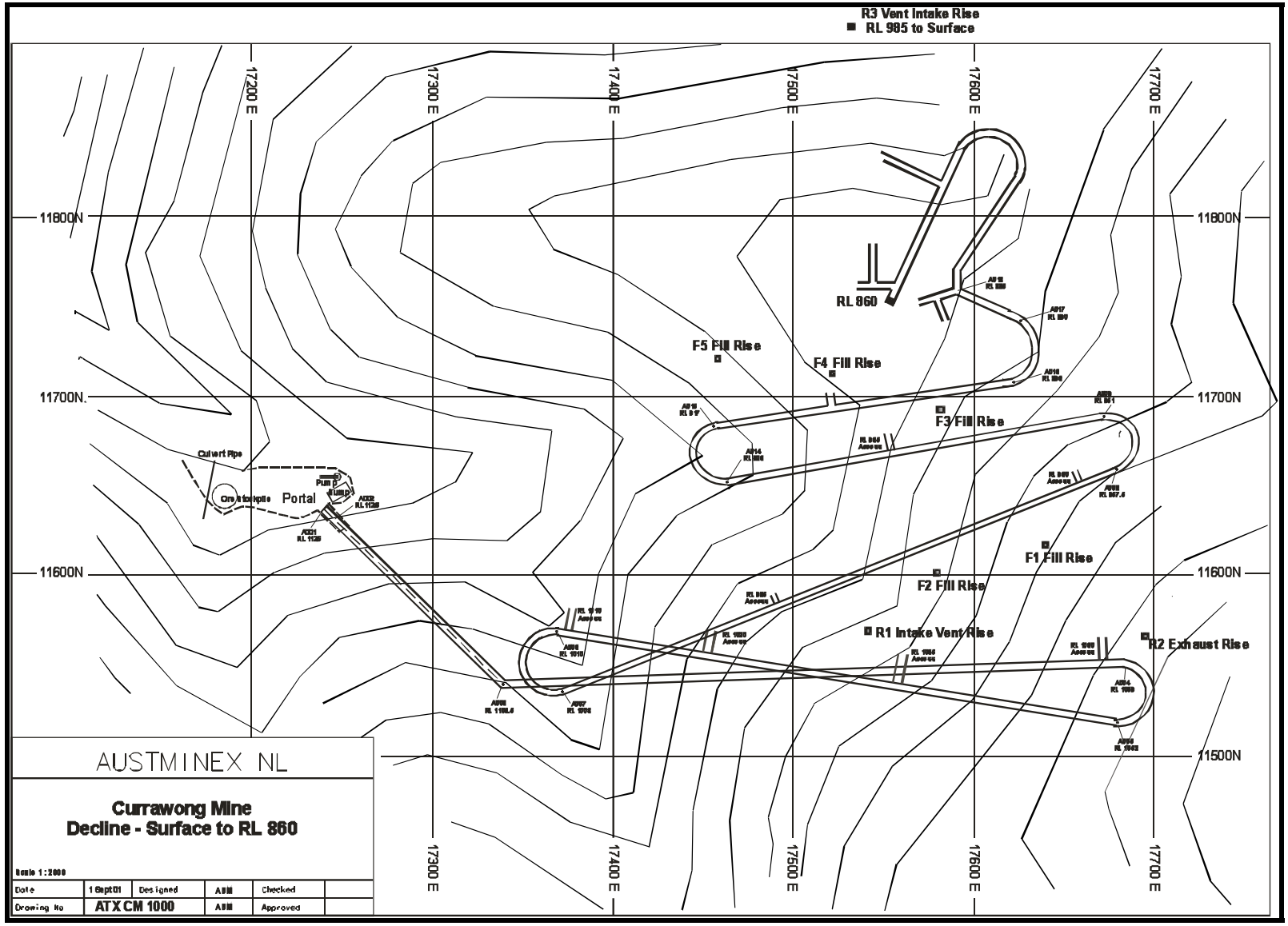
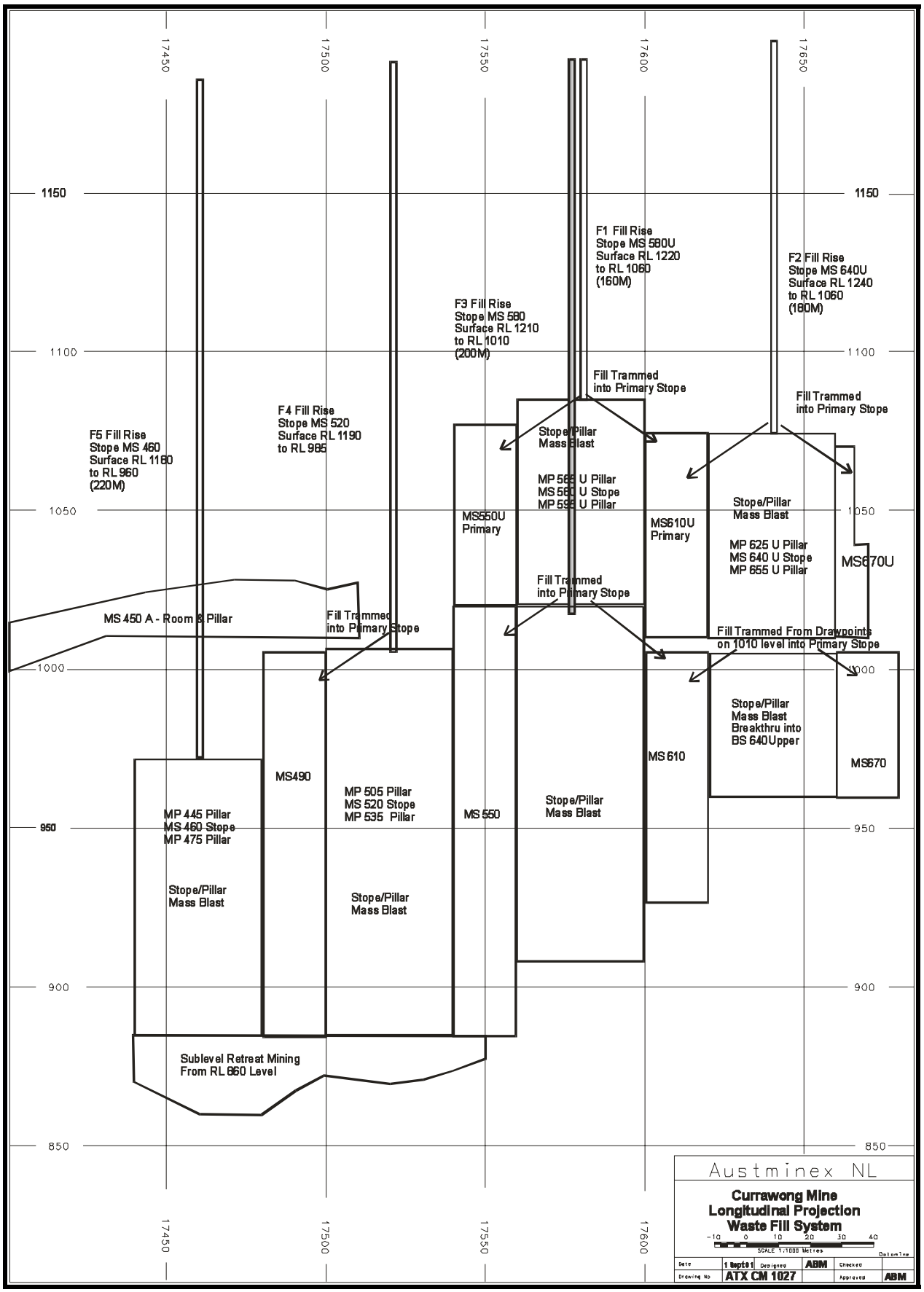


Figure No 7 – Currawong Longitudinal Projection and Waste Fill System



AUSTMINEX NL

BENAMBRA PROJECT

METALLURGICAL FLOWSHEET DEVELOPMENT

SUMMARY REPORT

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*Note: This index applies to this summary report (Phase II report plus excerpts from Phase I report). The index for the Phase 1 testwork report is also attached to indicate the source of further detailed information. The full metallurgical report comprises: Volume 1 – Phase I testwork, Volume 2 – Phase II testwork and Volume 3 – test sheets

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VOLUME 1 – PHASE I
TESTWORK REPORT

1 EXECUTIVE SUMMARY

A preliminary metallurgical test programme was conducted between November 2000 and February 2001on a range of samples from the Wilga and Currawong ore bodies. The samples tested included underground ore samples and diamond drill core from the previous operation and recent diamond drill core. Global composites representing the two ore bodies at about mining reserve grade were also prepared from the recent drill core and tested. The results of this work were presented in the report Metallurgical Flowsheet Development Phase I and Cost Estimation.

The metallurgical test programme has been extended to further improve the concentrate grade and recovery achieved and to increase the understanding of the variability of the response of the ore to the developed flowsheet. In addition, samples were produced for determination of engineering properties for use in preliminary plant design and equipment specification.

Because of this test programme, it has been demonstrated that an improved and robust flowsheet is possible over that which was used previously. This flowsheet does not use any unusual, exotic or environmentally toxic chemicals and, mostly, uses only well proven mineral processing technology. Other than the changes to the range and quantity of flotation reagents used, the most significant feature of the flowsheet is the incorporation of an ultra-fine grinding stage after the production of a primary rougher concentrate. Ultra-fine grinding is applied in both copper and zinc flotation stages.

The predicted average metallurgical performance from the two known ore bodies based on this test programme are shown in the following table.

Product	**Copper Concentrate**		**Zinc Concentrate**	
Parameter	**Wilga**	**Currawong**	**Wilga**	**Currawong**
Concentrate Grade	25%Cu	26%Cu	50%Zn	51%Zn
Metal Recovery	83%	83%	85%	80%

It is expected that concentrate grade higher than the target values will be produced from ore with higher head grade than the resource average, possibly with higher metal recovery. Because of the need to maintain a marketable concentrate grade, it is expected that lower recovery of metal to a concentrate of the target grade will be achieved from lower grade ore.

Locked cycle flotation tests have indicated that the open cycle test results can be reproduced quite accurately with no deleterious effects due to circulating loads. A simple test, using water recovered from filtering of other flotation test products as make-up, indicated that there were no immediate effects of water recycle. This does not guarantee that effects due to water recycle, beneficial or otherwise, will not occur in an operating plant.

One potential problem, in the ultra-fine grinding stage, is the apparent deleterious effect on grinding mill performance of the presence of talc in the primary rougher concentrate. It has been demonstrated that samples of concentrate low in magnesia and silica (an indicator of talc content) exhibit much lower grinding energy requirements. It has also been demonstrated, on a sample with 'high' talc content, that the talc content of the primary concentrate can be controlled by the use of appropriate quantities of talc depressant.

Although most of the metallurgical test programme has been done with an ultra-fine grind size of 10μm, it has been demonstrated that some relaxation of this criterion will be possible in practice. This has allowed less stringent design criteria to be applied for ultra-fine grinding mill selection. This should result in lower capital and operating costs for this plant area than might otherwise have been the case.

Other engineering properties of the ore, concentrate and tailings have been determined. These fall in an expected and normal range and no operational difficulties are anticipated due to the fine product distribution that results from the ultra-fine grinding stage. One property that has not been quantified is the propensity of the concentrates to generate ultra-stable froth structures. Experience, at other operating flotation plants involving ultra-fine grinding, suggests that this only becomes a major problem at a significantly finer product size than is proposed here. An allowance, by means of elevated 'froth factors', has been incorporated into the pump selection and pipeline sizing calculations.

2 INTRODUCTION

In October 2000, Austminex NL started a study of the feasibility of re-opening the Benambra mine located near Omeo in north-eastern Victoria. This study has involved extensive laboratory testing to develop a processing flowsheet that is both technically and economically viable.

The first phase of metallurgical testing between November 2000 and February 2001 had demonstrated that regrinding of primary rougher concentrate was of significant benefit to both copper and zinc flotation performance. The second phase of metallurgical testing was directed to investigate further the possibilities of improving results through both chemical and physical means.

This report discusses and summarises the results of the second test programme, which extended from March 2001 to July 2001. Individual test details and results are contained in a separate volume, Benambra Project Metallurgical Test Sheets – November 2000 to July 2001.

3 METALLURGICAL TEST SAMPLES

3.1 Wilga Ore Body

Details of samples used for metallurgical testing of ore from the Wilga ore body were unchanged from those reported previously. Tests were conducted on Wilga global composite, BERD05, BERD07 and BERD09A.

A drum of ore, from the 755m level, that had been collected shortly before the mine closure in 1996, was partly used to carry out grinding characterisation testing at Amdel. The drum and the remaining sample was returned to the mine site at Benambra after completion of this test work.

3.2 Currawong Ore Body

Details of samples used for metallurgical testing of ore from the Currawong ore body were unchanged from those reported previously. Tests were conducted on Currawong global composite, BERD10, BERD12B and CURB135.

4 FLOWSHEET DEVELOPMENT

The flowsheet development programme conducted in Phase II was directed to achieve the following aims:

- To investigate possibilities for improving copper recovery and selectivity by use of variations to the reagent schedule.
- To investigate and optimise the use of finer concentrate regrinding for copper concentrate upgrading.
- To investigate and optimise the use of finer concentrate regrinding for zinc concentrate upgrading
- To carry out locked cycle flotation tests to demonstrate any effects of recycle streams on flotation performance.
- To carry out variability testing of the preferred flowsheet and reagent schedule on a range of ore samples.
- To produce samples for the determination of engineering properties of concentrates (and tailings where appropriate) for
    - ultra-fine grinding,
    - thickening and
    - filtering.

- To determine other engineering properties as appropriate for flowsheet design and equipment selection

As with the first phase of testing, the metallurgical test programme was directed to achieve stable chemical conditions that would be generally applicable to all variations in ore type that are likely to be received into the plant. Any additions to the reagent schedule were to be justified based on significant improvements in metallurgical performance. Any additions to the equipment schedule for the circuit were to be justified based on both significant improvements to the metallurgical performance as well as to the financial benefit of the project.

4.1 General Flotation Test Programme

At the completion of the previous stage of testing, it had been determined that some degree of ultra-fine grinding would be required to achieve a significant improvement in the metallurgical performance from Benambra ore. There was hope that there might be some scope for improving performance more economically by means of changes to the chemical regime rather than by pursuing further degrees of ultra-fine grinding. The test programme was planned to focus first on possible improvements to copper flotation. When satisfactory results had been achieved for copper flotation, the test programme was to be extended to the requirements for zinc flotation. The majority of testing was planned for the Wilga global composite with only confirmatory testing envisaged for the Currawong global composite. Following the completion of the global composite test programme, variability testing of individual diamond drill hole intersections was to be conducted.

4.1.1 Copper Flotation

Three main factors were investigated in relation to copper flotation. The first was to improve selectivity between chalcopyrite and sphalerite and/or pyrite, the second was to improve the liberation of chalcopyrite by finer regrinding of primary rougher concentrate and the third was the effect of primary mesh of grind.

4.1.1.1 Selectivity

The reagents selected for tests relating to sulphide mineral selectivity were collector, sodium meta-bisulphite and potassium ferricyanide. This programme was ultimately limited in favour of other, more favourable areas of investigation.

4.1.1.2 Liberation

The regrind size for primary copper rougher concentrate was tested at p_{80} 10μm and p_{80} 7μm. The majority of the testing was conducted at the former size distribution with only a single test at the latter. Additional tests investigated the sensitivity of copper ultra-fine flotation to the regrind size at a range of product sizes.

4.1.1.3 Primary Mesh of Grind

A series of tests was conducted, varying the primary mesh of grind to determine whether the primary grind size could be coarsened without significant sacrifice of the ability to produce satisfactory concentrate grade and recovery.

4.1.2 Zinc Flotation

The main areas of investigation for zinc flotation once more related to selectivity against pyrite and to mineral liberation.

4.1.2.1 Selectivity

Addition rates for the main reagents (lime, copper sulphate and collector) were the only factors assessed in this phase of work.

4.1.2.2 Liberation

The regrind size for primary rougher concentrate was tested at p_{80} 15μm and p_{80} 10μm. The majority of the testing was conducted at the latter size distribution with only a few tests at the former. Additional tests investigated the sensitivity of zinc ultra-fine flotation to the regrind size at a range of product sizes.

4.2 Wilga Global Composite

The majority of the general metallurgical test programme for process development was conducted on the Wilga global composite. This represented the ore that would be treated initially in the re-started operation. There was also a larger quantity of the composite available at the laboratory to carry the greater workload that was expected for primary process development. It was expected that testing on the Currawong global composite and variability samples would be much less extensive and more of a confirmatory nature.

4.2.1 Test Results – Copper Flotation

The initial stage of the test programme was directed to achieving a significant improvement in the copper metallurgical performance over that achieved in the work reported previously. Minor adjustments to the copper flotation test conditions were made during the subsequent zinc flotation process development test programme.

4.2.1.1 Reagent Effects

Three flotation tests were conducted to test whether significant changes to the metallurgical performance could be brought about by changes to the flotation chemistry. The changes tested were a significant increase in the addition of sodium meta-bisulphite (MBS), the latter with a change to a stronger collector and the use of potassium ferricyanide in the ultra-fine flotation stages instead

of MBS. All tests involved regrinding primary rougher concentrate to p_{80} 15μm. The results of these tests are compared with the 'standard' test result (Test WGC07) in the following table.

TEST	COPPER		ZINC	
	Grade %Cu	Recovery %	Grade %Zn	Recovery %
		Cu/Zn		
WGC07	18.6	86.2/9.2	na	na
WGC08	17.6	83.6/9.5	nd	nd
WGC09	17.2	86.7/13.4	nd	nd
WGC10	18.1	86.6/13.8	nd	nd

These results, although far from comprehensive, indicate that the standard conditions produced the most acceptable result. There appeared to be little prospect of significant improvement by significant changes to the reagent schedule and significant potential for inferior performance with relatively minor changes. For these reasons, this line of investigation was discontinued in favour of investigating the effects of finer regrinding.

4.2.1.2 Effect of Ultra-fine Grind Size

Two tests were conducted to investigate the copper flotation response at p_{80} 10μm and p_{80} 7μm. The addition of collector to the first stage of ultra-fine flotation after regrinding was increased to 20g/t and 25g/t respectively to provide for the additional surface area generated by regrinding to the finer size. In the light of subsequent testing, these increases in collector addition might not have been sufficient. The results of these tests are summarised in the following table.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn
WGC11	27.3	77.0/3.7
WGC12	23.7	80.1/7.2

The concentrate grade achieved in Test WGC11 was much improved over anything achieved previously, although the recovery was relatively low. The test result with the finer grind size (Test WGC12) gave a result about equivalent to single stage ultra-fine flotation in Test WGC11. Given the significant improvement in results with a 10μm grind together with the high cost of achieving the finer grind and the possibly small improvement in liberation that would be obtained, it was decided not to pursue the finer grind size further.

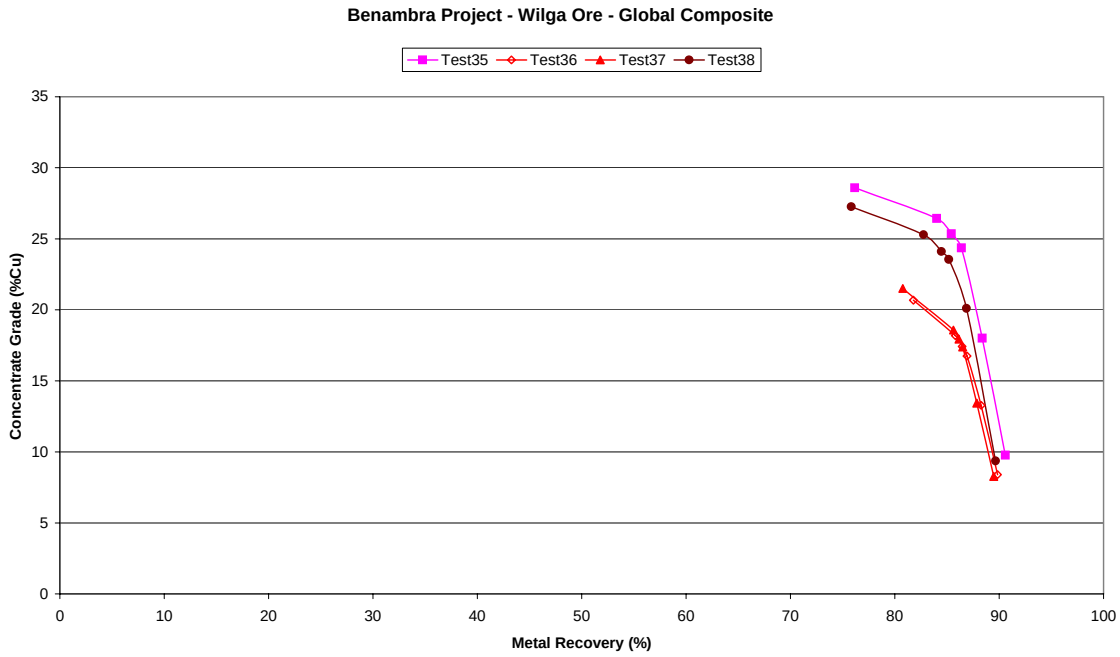
Two further tests with the 10μm grind were conducted, increasing collector addition to the ultra-fine rougher and then, in conjunction, increasing MBS addition to both stages of ultra-fine flotation. The results of these tests are summarised in the following table.

TEST	**COPPER**	
	Grade %Cu	**Recovery %**
		Cu/Zn
WGC13	20.0	86.9/10.2
WGC14	25.0	86.0/5.8

The increased collector had the desired effect of reducing the losses through the ultra-fine flotation stages although selectivity was significantly reduced. Increasing the addition of MBS had the effect of restoring selectivity without affecting the recovery through the ultra-fine flotation stages.

At the end of the test programme on the Wilga global composite, a series of tests was done to revisit the effect of ultra-fine grind size with a view to defining more closely the duty required of the proposed ultra-fine grinding mills. Tests were done with ultra-fine product sizings of 10, 13 and 15μm. In the last test in the series at 13μm, a reduction in collector addition to the ultra-fine flotation stage was made.

TEST	ULTRA-FINE	COPPER	
	GRIND SIZE	Grade %Cu	Recovery %
	µm		Cu/Zn
WGC35	10	24.4	86.4/6.3
WGC36	15	16.7	86.9/10.1
WGC37	13	17.4	86.5/11.6
WGC38	13	23.5	85.2/6.4



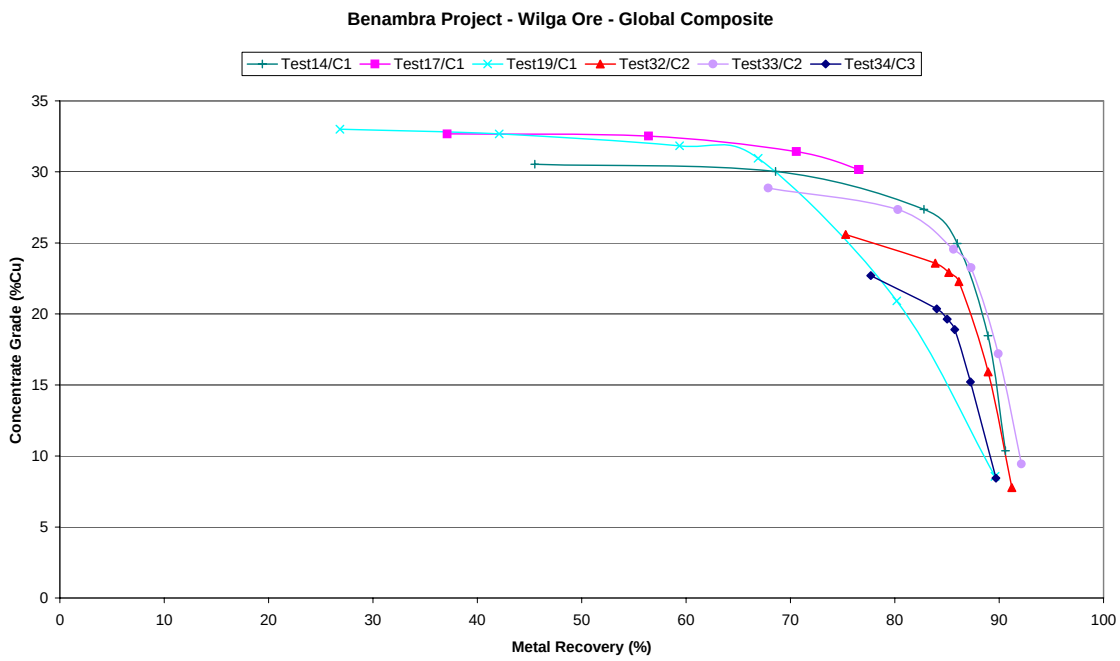
These results indicate that a significant proportion of the benefit of regrinding beyond 15µm can be achieved at a product size of 13µm. This has significant implications for the sizing and operation of the ultra-fine grinding circuit in any proposed operation. There is potential for reduced capital costs and reduced operating costs arising from the reduced duty.

4.2.1.3 'Standard' Test Reproducibility

The conditions used in Test WGC14 became an effective standard and, during the course of the test programme, a number of tests were conducted that could be considered to be duplicate tests. Unfortunately, there were some significant variations in response and a number of other tests were done that involved relatively minor variations from this standard. These test results are reported in the next section.

TEST	COPPER	
	Grade %Cu	**Recovery %**
		Cu/Zn
WGC14	25.0	86.0/5.8
WGC17	30.2	76.6/1.6
WGC19	31.0	66.9/1.5
WGC32	22.3	86.1/5.7
WGC33	23.3	87.3/7.8
WGC34	18.9	85.8/7.7

Despite the variations in the result at the completion of the test, the tests actually conform to a regular pattern of grade and recovery as illustrated in the following graph. These results were produced from three different 'versions' of the Wilga global composite that were produced during the course of the test programme. It should also be noted that the distribution of the timing of fractions taken in the ultra-fine cleaning stage was changed between test WGC19 and WGC32 although the total flotation time was not changed.



It is not clear what the cause of the variation in outcome of the tests is, but one possibility lies in the complexities of processing the primary rougher concentrate in preparation for and following the regrinding process.

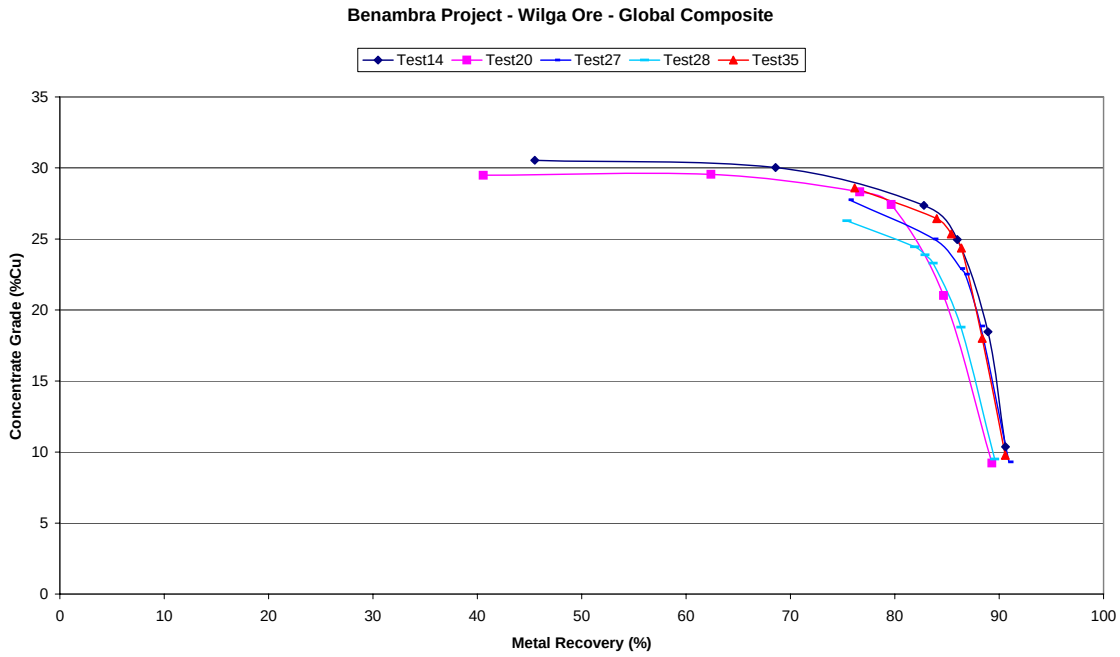
4.2.1.4 Modified 'Standard' Test Conditions

As a response to variability in test results, minor changes were made to the 'standard' test conditions to try to correct for the variations. The following groups of tests were the result of these changes.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn
WGC14	25.0	86.0/5.8
WGC20	27.4	79.7/3.5
WGC27	22.5	86.7/7.6
WGC28	23.3	83.7/7.4
WGC35	24.4	86.4/6.3

The four tests in this group involved increased collector (40g/t) addition to the ultra-fine roughing flotation stage. The middle two tests also had an additional

10g/t of collector added to the ultra-fine cleaner. This might have contributed to the lower concentrate grade reported for these two tests. Although the end result of the test indicated considerable variation, as before, the grade recovery curves were reasonably consistent with recovery to a 25% concentrate ranging between 80% and 86%.

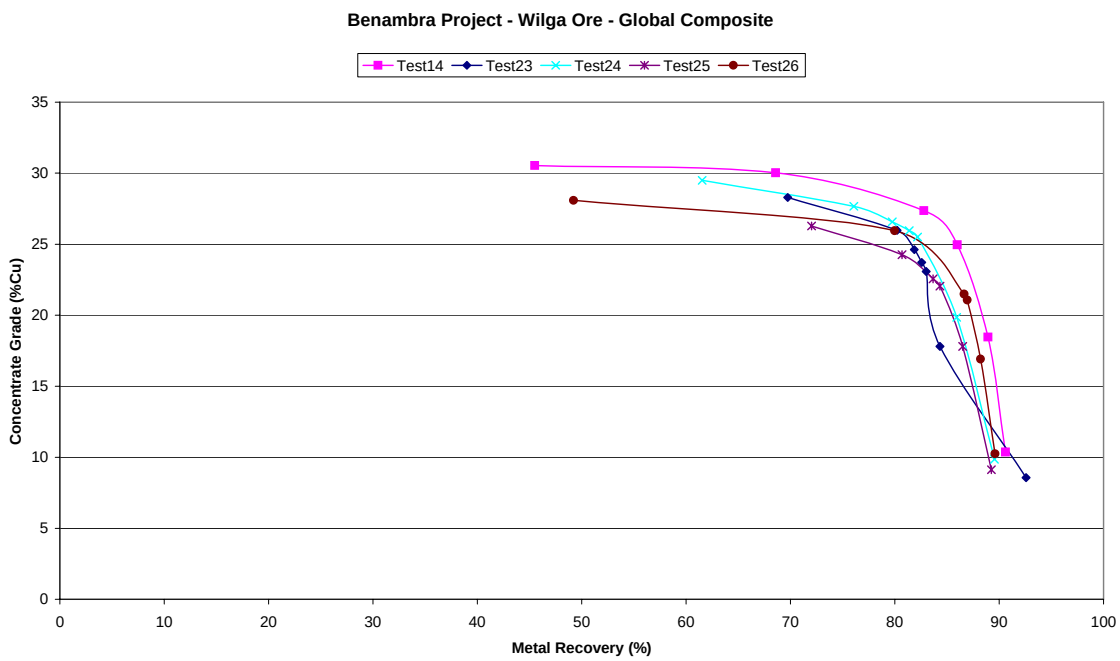


These results suggest that there was little difference between performance at the modified conditions compared to the 'standard'. Once more, there was a change in the test fraction timing between the earlier and the later tests.

Another group of tests that can be considered as a set of modified 'standard' tests involved the use of reduced MBS addition to the ultra-fine roughing and cleaning stages.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn
WGC14	25.0	86.0/5.8
WGC23	23.1	83.0/7.0
WGC24	25.5	82.2/5.4
WGC25	22.1	84.3/6.6
WGC26	21.1	86.9/7.7

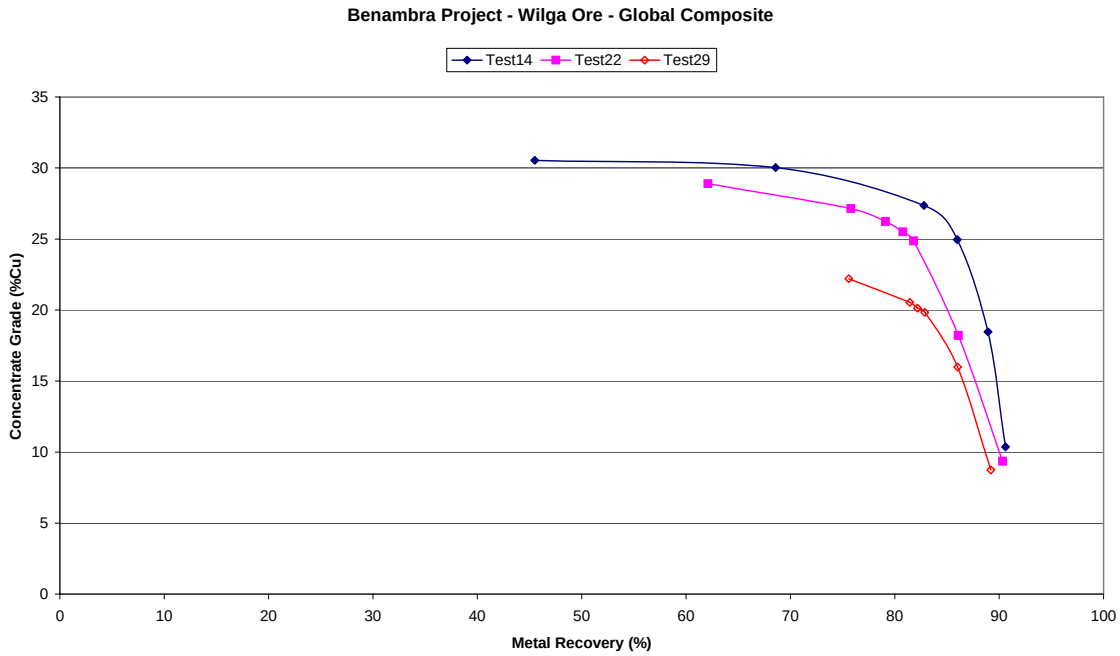
Once again, the test end point shows considerable variation. The grade recovery curves are relatively consistent with some possible sampling/assay variability causing the abnormal appearance of the curve for test WGC23. It would appear that the ultra-fine roughing and cleaning performance was slightly adversely affected by the reduction in MBS addition when compared to the 'standard' with the recovery at 25% concentrate grade falling at the low end of the general range.



During the execution of this programme, two other tests involved minor variations to the standard conditions but these were not repeated in any other test.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn
WGC14	25.0	86.0/5.8
WGC22	24.9	81.8/4.4
WGC29	19.8	82.9/8.8

The changes in reagent addition involved increased collector (45g/t) in ultra-fine roughing and increased MBS in primary roughing. The test results showed that little or no improvement in performance might be achieved by such changes. The poor performance exhibited in test WGC29 is similar to that which resulted in test WGC34 and there is some possibility that this result is in some way aberrant.



4.2.1.5 Effect of Primary Grind Size

A series of tests was conducted to investigate the effect of primary grind size on copper flotation performance. The initial tests were directed to assess the effect of coarsening the primary grind to p_{80} 45µm and to p_{80} 53µm on primary rougher concentrate grade and recovery. An additional test at p_{80} 45µm was conducted using RTD12 as collector in place of the normal collector A3894.

TEST	COPPER	
	Grade %Cu	**Recovery %**
		Cu/Zn
WGC14	10.4	90.6/17.2
WGC15	9.7	85.4/16.4
WGC16	12.4	54.1/8.2
WGC18	6.4	92.6/26.6

The effect of coarsening the primary grind at the standard reagent addition was to reduce concentrate grade and recovery although the result at the coarsest grind size appeared to be anomalous with higher concentrate grade and significantly reduced recovery. The final test in this series produced increased recovery but with significantly lower concentrate grade and correspondingly higher zinc recovery to the primary rougher concentrate.

Because of the (possibly anomalous) result at p_{80} 53μm, the next group of tests was conducted at p_{80} 45μm. The tests extended to ultra-fine roughing and cleaning after regrinding to 10μm.

TEST	COPPER	
	Grade %Cu	**Recovery %**
		Cu/Zn
WGC14	25.0	86.0/5.8
WGC21	23.6	81.6/5.7
WGC30	22.5	84.5/8.1
WGC31	24.4	78.6/5.2

In line with the results from the roughing tests, final concentrate grade and recovery were lower from the coarser primary grind. Changing the collector type and addition did not produce the hoped for improvement. The final test at p_{80} 53μm confirmed the trend to lower recovery although the result was significantly better than would have indicated from the primary rougher test

earlier. This tended to confirm that the result of the earlier test was, in some way, anomalous.

The test results suggest that coarser primary grind size will result in higher primary rougher concentrate mass (lower grade at similar recovery), resulting in higher energy costs for regrinding. The target concentrate grade will be more difficult to achieve from the lower primary rougher concentrate grade without loss in recovery.

This option is probably better pursued during operation as a tuning exercise rather than as a significant design criteria.

4.2.2 Test Results - Zinc Flotation

When acceptable results had been achieved for copper flotation, testing of conditions required for zinc flotation began. As largely as possible, the conditions for 'standard' copper flotation were used as a pre-cursor to zinc flotation. The first stage of the programme was to assess the sensitivity of flotation response to variations in the conditions applied for primary rougher flotation. Additional testing addressed the conditions required for ultra-fine roughing and cleaning including regrind particle size.

4.2.2.1 Reagent Effects – Primary Rougher Flotation

Based on experience, it was considered that the reagent additions used in earlier zinc flotation testing on the Wilga global composite could have been high for the head grade of ore tested. It was also considered necessary to investigate a range of flotation conditions to try to minimise the use of lime in the zinc flotation circuit because of possible operating problems arising from the recycling of process water. This effect was a topic of considerable discussion during the previous operation of the plant by Denehurst.

4.2.2.1.1 Variation in Activator and Collector

The initial group of tests investigated the effects of varying the addition rate of copper sulphate and collector to the primary rougher stage of zinc flotation. The primary rougher flotation pH was maintained at 10.5 with lime. A 50:50 blend of sodium iso-butyl xanthate (SIBX) and Cytec A3047 was used as collector throughout. All tests except WGC07 followed 10μm regrind in copper flotation with some variation in copper flotation chemistry as well.

TEST	REAGENTS	ZINC	
	$CuSO_4$/Collector	Concentrate Grade %Zn	Recovery %
	g/t		ToZn/ToCu/Total
WGC07	1000/80	20.4	88.7/9.2/97.9
WGC17	500/80	24.1	95.7/1.6/97.3
WGC19	250/80	26.2	93.3/1.5/94.8
WGC20	250/50	26.9	91.6/3.5/95.1
WGC26	500/50	21.3	89.3/7.7/97.0

These results indicate that the recovery of zinc to both copper and zinc concentrate is essentially constant. There is a slight reduction in total recovery at the low copper sulphate addition. The main variation in zinc recovery to zinc concentrate relates to the variation in zinc recovery to copper concentrate. This reinforces the importance of maintaining high selectivity in the copper flotation stage. Zinc concentrate grade appears to be favoured by lower reagent addition rates although the higher grades also correspond to the tests with greater selectivity in the copper section. This suggests a 'flow through' effect from the copper section in maintaining selectivity, possibly against pyrite as well.

4.2.2.1.2 Variation in Slurry pH

Additional tests were conducted to investigate the effects of primary rougher flotation pH on zinc flotation performance. The other reagent conditions were maintained at the levels set in test WGC26. This test was selected over others in the series because of the abnormally low copper and zinc recovery to the copper concentrate that had occurred in these tests compared to the 'normal'.

TEST	CONDITIONS	ZINC	
	pH	Concentrate Grade %Zn	Recovery %
			ToZn/ToCu/Total
WGC26	10.5	21.3	89.3/7.7/97.0
WGC25	9.5	28.2	88.3/6.6/94.9
WGC32	8.5	22.8	90.5/5.7/96.2
WGC33	8.5	23.5	88.7/7.8/96.5

These results confirm that the total zinc recovery is remarkably consistent and that it is quite insensitive to the slurry pH used for flotation. The zinc concentrate grade also showed no marked trends as the pH was varied. It is not clear whether the result achieved in test WGC25 was significant as no repeat test was conducted.

4.2.2.1.3 'Standard' Test Reproducibility

The test conditions from test WGC26 were set as the 'standard' conditions for zinc primary rougher flotation and a number of tests were conducted under these conditions. The results of these tests are summarised in the following table.

TEST	CONDITIONS	ZINC	
	pH	Concentrate Grade %Zn	Recovery %
			ToZn/ToCu/Total
WGC26	10.5	21.3	89.3/7.7/97.0
WGC27	10.5	25.3	88.3/7.6/95.9
WGC28	10.5	25.3	88.0/7.4/95.4
WGC29	10.5	28.3	85.9/8.8/94.7
WGC34	10.5	24.0	89.1/7.7/96.8
WGC35	10.5	24.6	90.3/6.3/96.6

These tests show generally good reproducibility taking into account the dependencies relating to performance in the copper section. The higher concentrate grade reported in test WGC29 corresponded to a lower overall recovery but there were no obvious other factors likely to have affected the result.

4.2.2.2 Primary Concentrate Upgrading

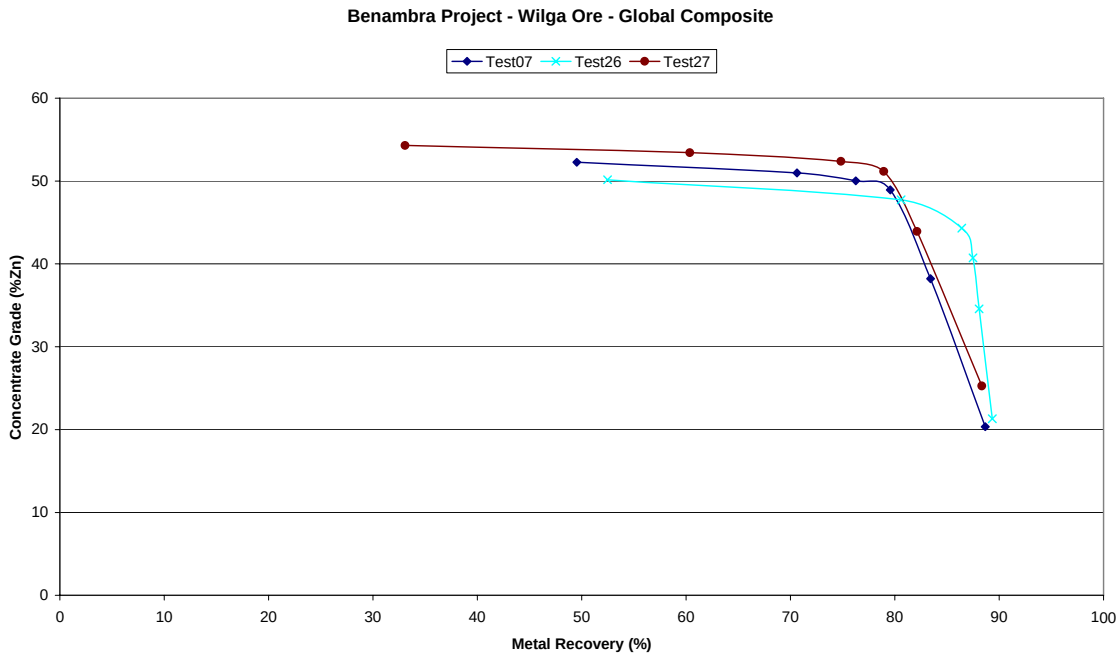
All of the tests done for zinc primary rougher concentrate upgrading were conducted following regrinding to some degree. The main items tested were for effects of concentrate grind size and limited testing of variations to chemical conditions in this process stage.

4.2.2.2.1 Effect of Ultra-fine Grind Size

Two tests were conducted with the primary rougher concentrate reground to p_{80} 15μm prior to ultra-fine flotation. These tests followed primary rougher flotation with quite different chemical conditions. In the case of the second test, the copper primary rougher concentrate had been reground to p_{80} 10μm. There was no strictly comparable test for either of these copper flotation conditions that was followed by zinc regrinding to 10μm. However, based on the relative lack of sensitivity of copper performance to the variations in conditions involved in the copper flotation stage, the results of the first test conducted with the zinc regrind to 10μm is considered to be relevant.

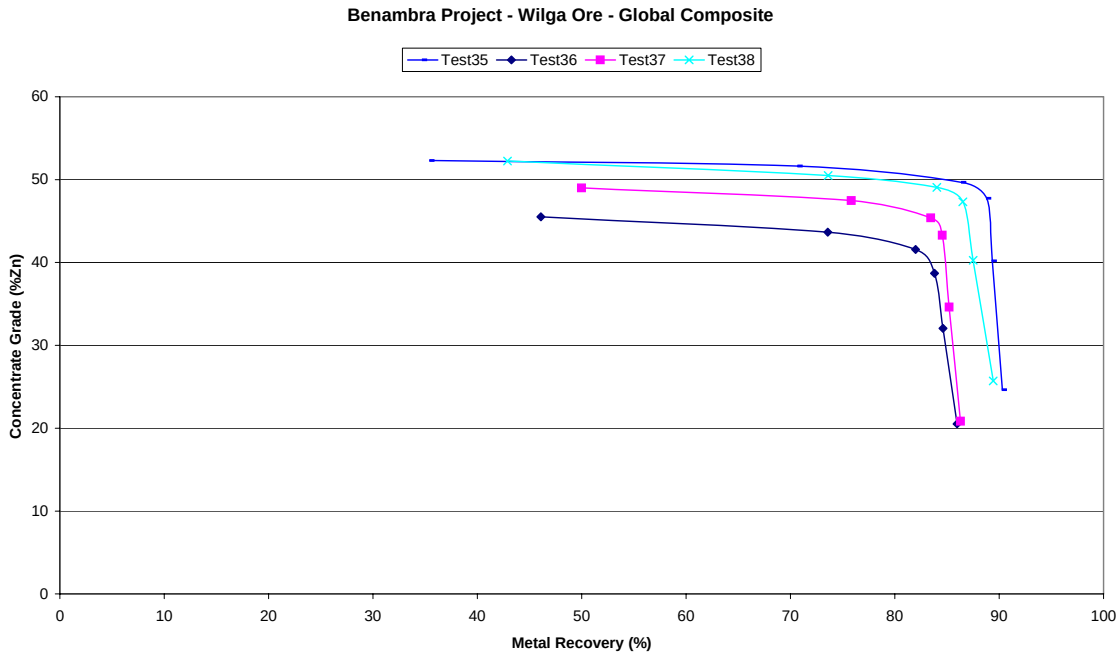
TEST	ZINC	
	Grade %Zn	Recovery %
		Zn/Pb/Cu
WGC07	48.9	79.6/8.2/2.9
WGC26	40.7	87.5/13.6/6.0
WGC27	51.1	78.9/11.4/5.2

The effect of finer regrinding appears to have been to increase the concentrate grade by about 2%Zn at similar zinc recovery. The low concentrate grade in test WGC26 is largely a reflection of the significantly higher recovery reported. At a comparable recovery to the other two tests, the concentrate grade in test WGC26 was approximately 48%Zn.



As for the copper flotation stage a series of tests was conducted to investigate whether the requirement for an ultra-fine regrind size of 10μm could be relaxed. The grind size pattern followed that of the copper stage.

TEST	ULTRA-FINE	Zinc	
	GRIND SIZE	Grade %Zn	Recovery %
	µm		Zn/Pb/Cu
WGC35	10	47.7	88.8/15.8/7.2
WGC36	15	44.0	85.5/16.2/5.6
WGC37	13	43.3	84.5/18.7/7.0
WGC38	13	47.3	86.5/14.3/6.8



Although the results are not as clear cut as those for copper, the results have been affected by the variations in zinc recovery reported in the copper flotation stage. Despite this, it would appear to be safe to conclude that the possibility exists of being able to accept an ultra-fine grind product as coarse as 13µm and still be able to realise almost the same benefits as would be achieved by grinding to the finer product size.

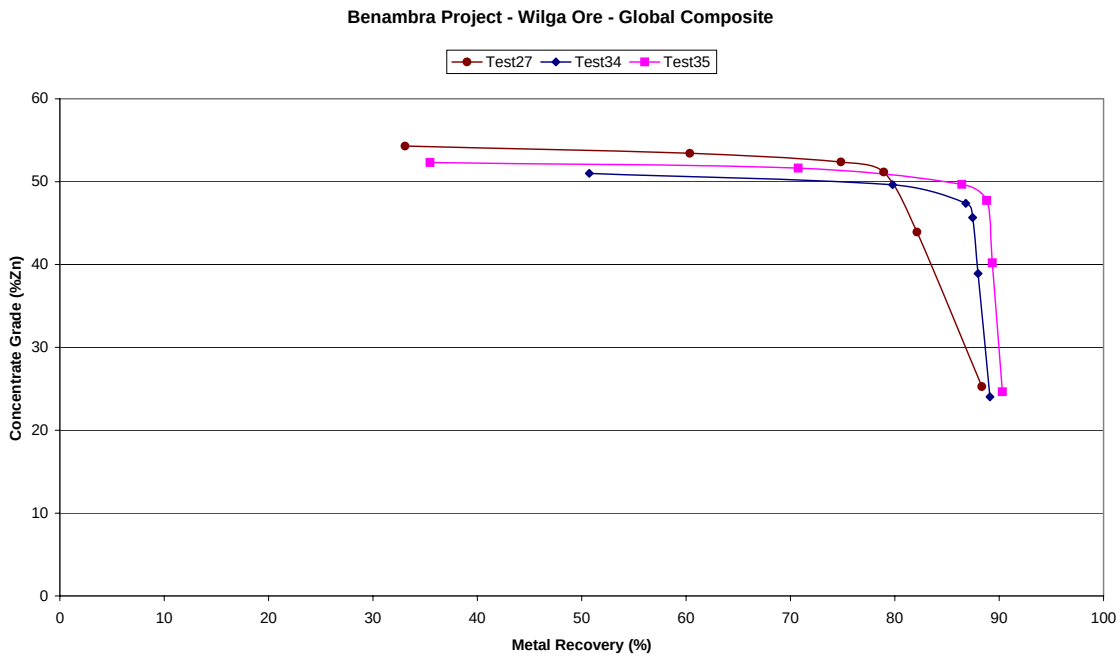
4.2.2.2.2 <u>'Standard' Test Reproducibility</u>

The test conditions from test WGC27 were set as the 'standard' conditions for zinc ultra-fine flotation and a number of tests were conducted under these conditions in the course of the ongoing test programme. There were variations in the preceding copper flotation conditions and flotation results that will have had some influence on the zinc flotation outcome in particular with

respect to the recovery achieved. The results of these tests are summarised in the following table.

TEST	ZINC	
	Grade %Zn	Recovery %
		Zn/Pb/Cu
WGC27	51.1	78.9/11.4/5.2
WGC34	45.7	87.5/15.3/6.6
WGC35	47.7	88.8/15.8/7.2

Although the test endpoint in these tests is significantly different, the similarity of the grade recovery curves is evident in the following chart. It is not immediately evident what caused the difference in end point of the tests but as discussed previously the regrinding step is quite long and complex and could give rise to some variations.

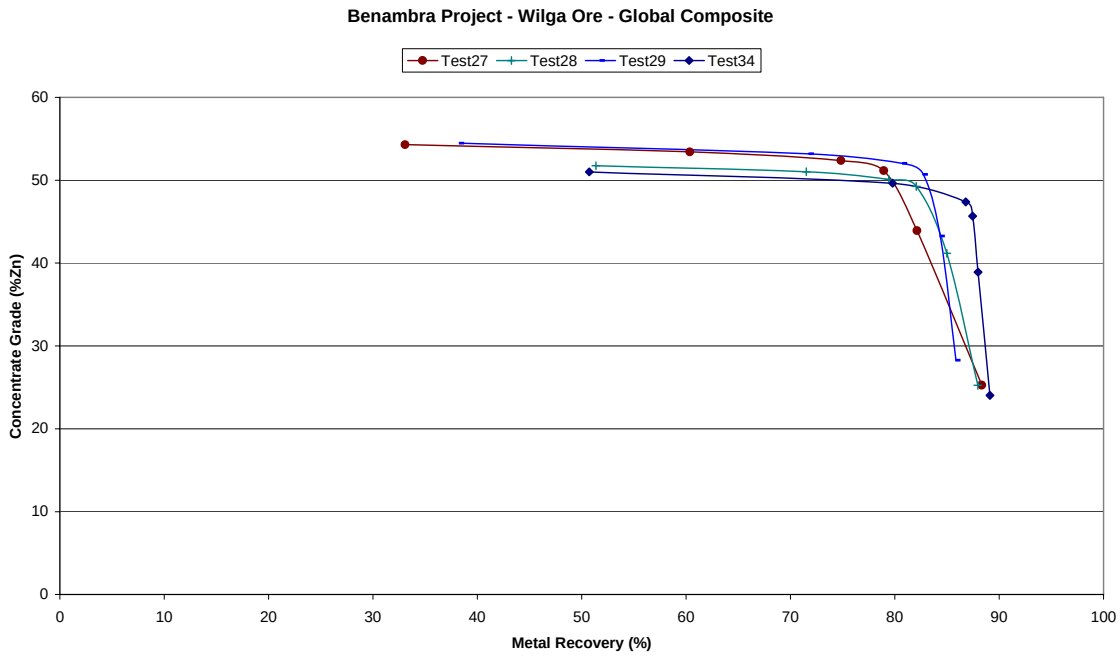


4.2.2.2.3 Modified 'Standard' Test Conditions

As with the copper test programme, some minor changes were made to the standard test conditions in response to unexplained variations in test results. These tests involved a small addition of collector to the ultra-fine roughing stage.

TEST	ZINC	
	Grade %Zn	Recovery %
		Zn/Pb/Cu
WGC27	51.1	78.9/11.4/5.2
WGC28	49.3	82.1/12.5/6.5
WGC29	50.7	82.7/12.4/6.6

As was the case with the copper flotation, such changes to the reagent schedule had relatively little effect on the flotation performance. Some of the differences in performance can be attributed to variations in the result from copper flotation.



4.3 Currawong Global Composite

To a large degree, the tests done on the Currawong global composite were based on what had been applied to the Wilga global composite with only minor changes to conditions to trim the results achieved. The Currawong global composite has a greater proportion of a naturally floating mineral ('talc') and relies on the performance of the depressant sodium carboxymethylcellulose (CMC) for satisfactory results. The addition of CMC was used throughout the test programme on the Wilga global composite for consistency rather than due to a positively demonstrated need for the reagent with this composite and with no apparent deleterious effect. No optimisation

of the CMC addition required for either of the global composites has been conducted.

4.3.1 Test Results – Copper Flotation

In general, the flotation results achieved from the Currawong global composite exceeded those achieved from the Wilga Global composite, particularly with respect to the concentrate grade achieved.

4.3.1.1 Effect of Ultra-fine Grind Size

Two tests were conducted in which the copper primary rougher concentrate was subjected to grinding to p_{80} 10μm. In the second test, the collector addition to ultra-fine roughing was increased from 30g/t to 40g/t.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn/Pb
CGC07	24.1	78.2/5.1/47.9
CGC08	31.0	70.9/0.9/34.1
CGC09	27.5	81.1/2.0/42.5

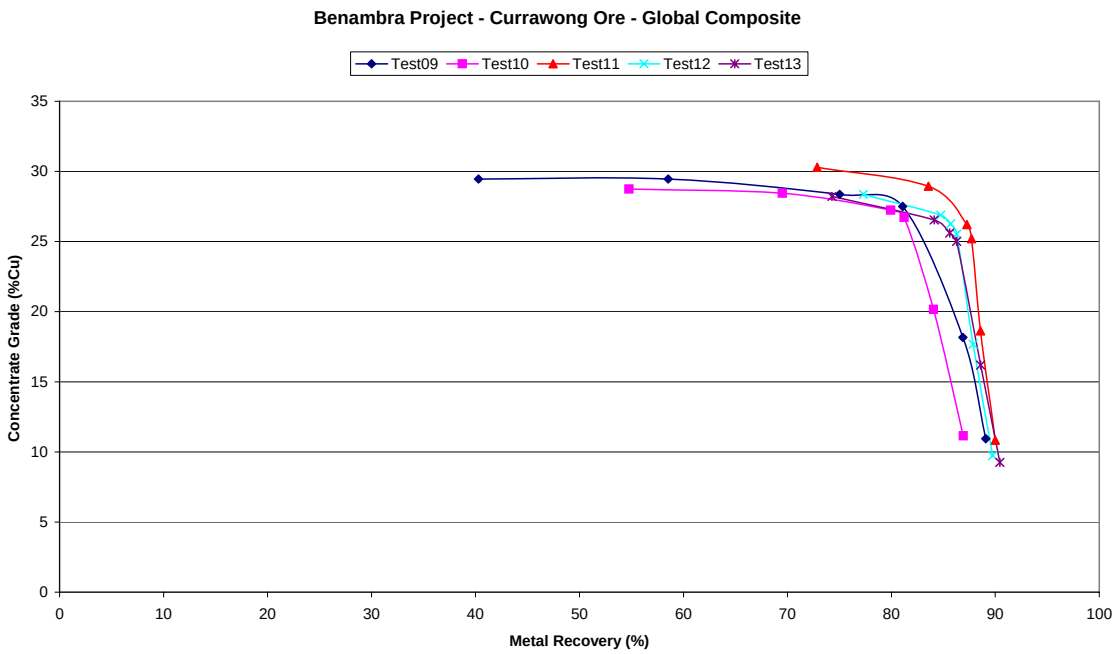
These results confirmed the generally better copper metallurgical performance from the Currawong ore and the significant improvement achieved by applying finer grinding to the primary rougher concentrate.

4.3.1.2 'Standard' Test Reproducibility

The conditions used for test CGC09 were accepted as 'standard' for copper flotation and most subsequent tests were conducted using these test conditions. Three tests were included that involved the addition of an extra 10g/t of collector to the ultra-fine cleaner stage of flotation. This change appeared to have had an insignificant effect on the grade recovery curve.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn/Pb
CGC09	27.5	81.1/2.0/42.5
CGC10	26.7	81.2/2.6/49.5
CGC11	25.2	87.7/4.8/53.6
CGC12	25.5	86.3/4.2/48.3
CGC13	25.0	86.3/4.4/56.3

The additional collector to the ultra-fine cleaners (test CGC11-CGC13) resulted in higher recovery but with lower concentrate grade. The general consistency of the grade recovery curve through these tests can be seen in the chart below.

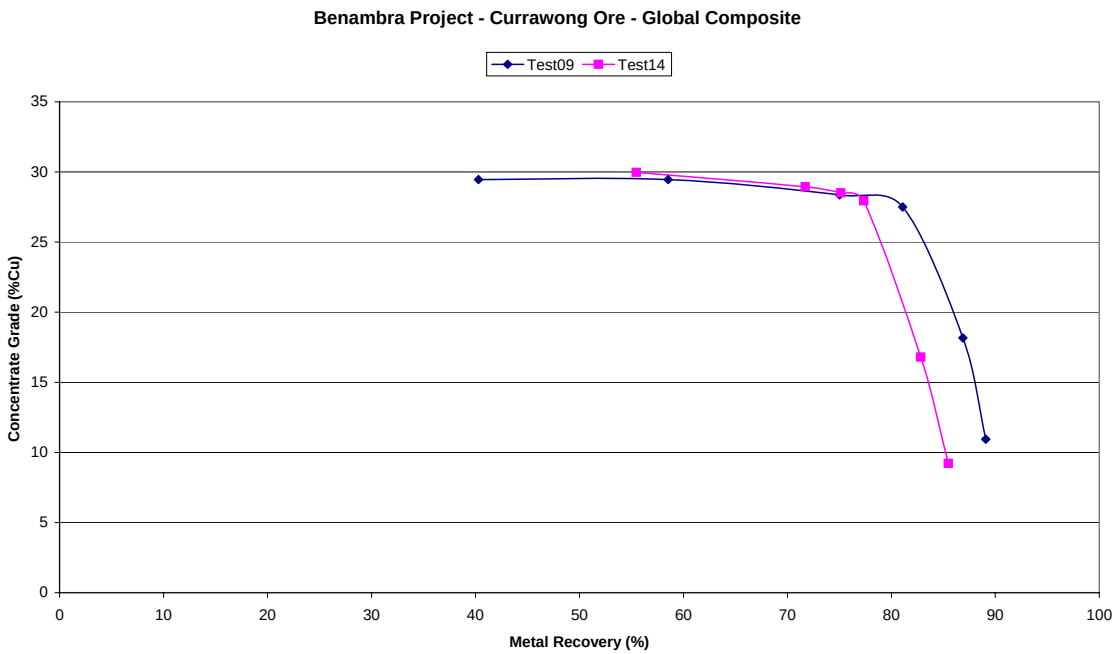


4.3.1.3 Effect of Primary Grind Size

One test was done on the Currawong global composite at a primary grind size of p_{80} 45µm for comparative purposes.

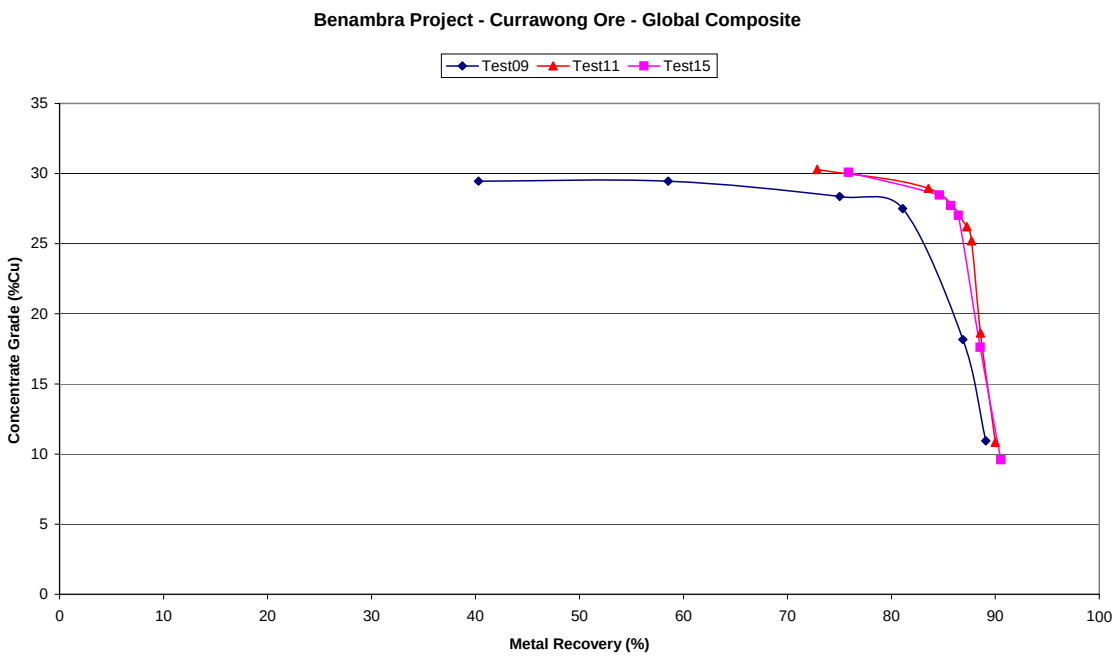
TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn
CGC09	27.5	81.1/2.0
CGC14	28.0	77.4/2.1

As with the Wilga global composite, the coarser primary grind resulted in loss of recovery, in particular. Final concentrate grade can be maintained at or about similar levels at both grind sizes even though starting primary rougher concentrate grade is lower.



4.3.1.4 Effect of Recycled Water

One test was conducted in which the water used throughout the test had been recovered from products generated in a series of 'standard' tests on other composites. From experience in the laboratory, with locked cycle tests using recycled water as well, any effects of water recycle usually show up in the second cycle (first use of recycled water). On this basis, it was hoped to demonstrate whether any short term effects of water recycle could be anticipated.



The results indicate that there is no adverse effect and the result achieved was equivalent to the best result achieved from any test in the programme.

4.3.2 Test Results – Zinc Flotation

As was the case for the Wilga global composite, zinc flotation was conducted when acceptable results had been achieved from copper flotation. Zinc flotation conditions were also based on the Wilga global composite conditions.

4.3.2.1 Reagent Effects

One test was conducted to assess the effect of reducing activator and collector in line with the tests done on the Wilga global composite.

TEST	REAGENTS	ZINC	
	$CuSO_4$/Collector	Concentrate Grade %Zn	Recovery %
	g/t		ToZn/ToCu/Total
CGC07	1000/80	24.8	91.0/5.1/96.1
CGC10	500/50	23.8	94.0/2.6/96.6

This test indicated that there was little effect from the reduced reagent addition and that the main difference was due to the different performance with respect to zinc depression in the copper circuit.

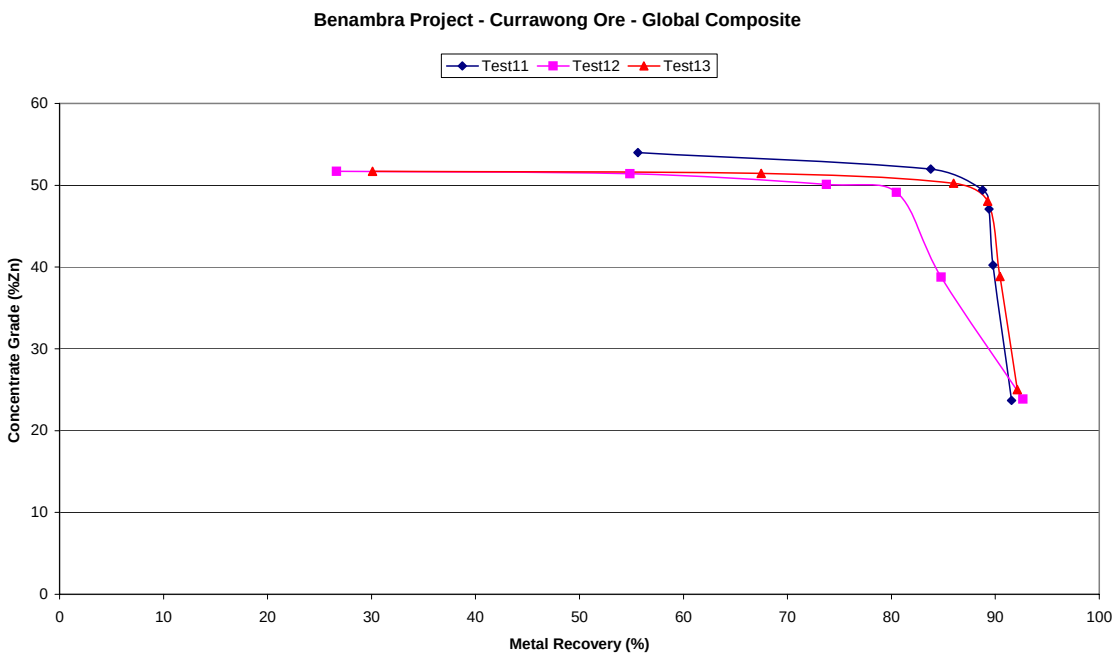
4.3.2.2 Ultra-fine Flotation

The only effect investigated in zinc concentrate upgrading was the effect of primary rougher concentrate grind size.

Two tests were conducted with identical chemical conditions for primary roughing but with primary rougher concentrate reground to p_{80} 15μm and p_{80} 10μm. A third test incorporated a small addition of collector to the ultra-fine rougher to reduce excessive losses through this circuit.

TEST	ZINC	
	Grade %Zn	Recovery %
		Zn/Pb/Cu
CGC11	47.1	89.4/16.5/4.9
CGC12	49.1	80.5/17.0/6.4
CGC13	48.1	89.3/15.7/6.7

The change in flotation performance between the two grind sizes was limited to a 1% increase in concentrate grade although the zinc losses in the copper circuit were higher in the latter two tests. There is, relatively, only a small difference in the grade recovery curve at the two product sizes suggesting that, as with the Wilga global composite, the regrind product size could be relaxed without sacrificing significant metallurgical performance.



4.4 Locked Cycle Testing

When it was considered that reasonably stable test conditions and metallurgical performance had been achieved, locked cycle tests were conducted. Two tests on each of the two global composites were planned to allow for minor adjustments of test conditions if any effects of recycled process streams were evident. It was also planned to accumulate the concentrates produced for conducting thickening and filtering tests on concentrate. A sample of tailings was also to be prepared for thickening testing.

The total test sequence mass balance head assay was calculated, including all process streams. The sample head grade was also calculated from concentrates and final tailing for the last two cycles that were used for the test result determination.

Grade Source	Test	%Cu	%Pb	%Zn
Assay		2.18	0.42	5.95
Total Mass Balance	**LCT2**	2.10	0.40	5.79
	LCT4	2.21	0.42	5.78
Cycle 5/6 Product Balance	**LCT2**	2.11	0.40	5.82
	LCT4	2.23	0.42	5.81

The agreement between the assay head grade and the calculated head grade for both copper and zinc indicates that the assaying has been consistent and that there are no major imbalances due to high circulating loads.

4.4.1 Wilga Global Composite

Two tests were conducted on the Wilga global composite. The results of these tests are summarised in the following sections.

4.4.1.1 Copper Flotation Results

The first locked cycle test conducted on the Wilga global composite was to use the test conditions that were used for test WGC28 except that, following the completion of the first test on the Currawong global composite that had been done earlier, the collector addition to the copper primary rougher was reduced. For the second test in the series, the addition of MBS to the ultra-fine flotation stages was increased and the small addition of collector to the zinc ultra-fine rougher was eliminated.

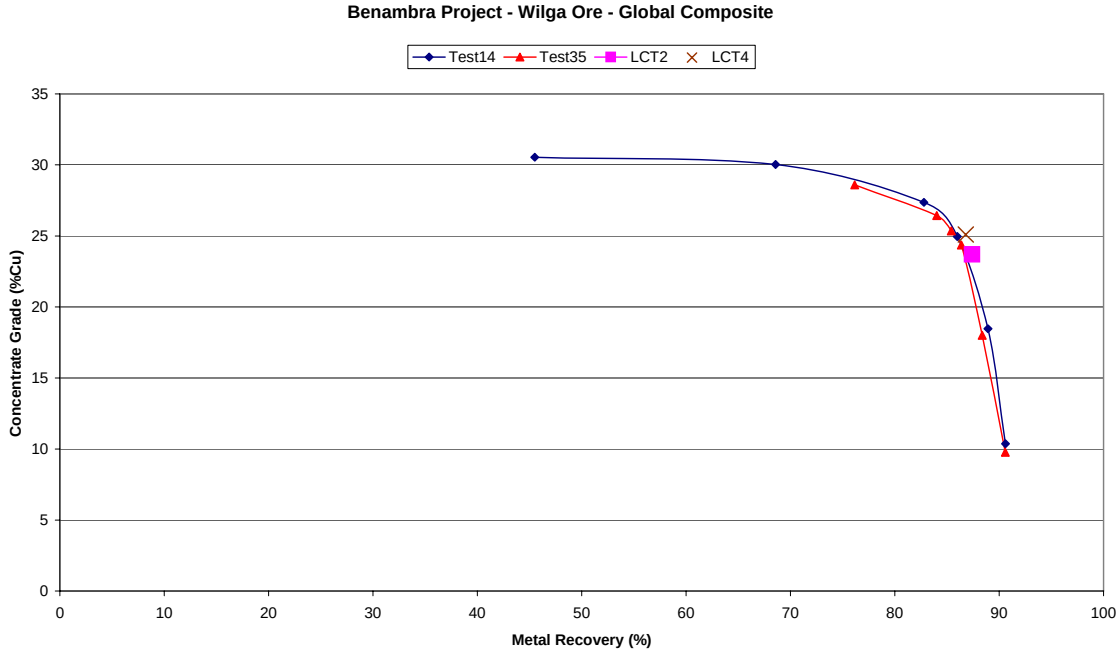
The test conditions stabilised very quickly, probably because of the minimal circulating streams that were involved. There was some instability in the concentrate mass recovery and corresponding variations in the concentrate grade. A variation in one direction was invariably followed by a compensating change in the next cycle. There was no evidence of increases in the circulating load through the test cycles and the final intermediate product assays were little different from those reported in the open circuit test results. This gives confidence that the open circuit test results are a valid representation of flotation performance.

Locked Cycle Test 2 – Wilga Global Composite – Copper Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Cu Concentrate Cycle 1	20.27	1.83	5.31	89.59
Cu Concentrate Cycle 2	22.42	1.95	4.75	81.38
Cu Concentrate Cycle 3	22.23	1.96	4.83	80.71
Cu Concentrate Cycle 4	22.48	1.99	4.63	83.84
Cu Concentrate Cycle 5	26.07	2.17	4.12	71.90
Cu Concentrate Cycle 6	21.37	2.01	4.50	80.56

Locked Cycle Test 4 – Wilga Global Composite – Copper Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Cu Concentrate Cycle 1	25.38	2.25	3.67	74.12
Cu Concentrate Cycle 2	26.88	2.28	3.87	72.93
Cu Concentrate Cycle 3	25.48	2.32	4.16	77.24
Cu Concentrate Cycle 4	27.46	2.48	3.57	68.81
Cu Concentrate Cycle 5	25.16	2.31	4.13	75.29
Cu Concentrate Cycle 6	25.10	2.27	3.83	74.56

The overall result for the two tests are summarised in the following table and chart.

Test Number	Concentrate Grade			Recovery		
	%Cu	%Pb	%Zn	%	%	%
LCT2	23.5	2.1	4.3	87.5	40.4	5.8
LCT4	25.1	2.3	4.0	86.9	42.1	5.3



The locked cycle test results have reproduced very closely, the best of the open cycle test results that were achieved from this composite.

4.4.1.2 Zinc Flotation Results

The test conditions for zinc flotation were set as for the open cycle test WGC28 for the first test on the Wilga global composite, LCT2. For the second test (LCT4), the small addition of collector to the ultra-fine rougher stage was discontinued.

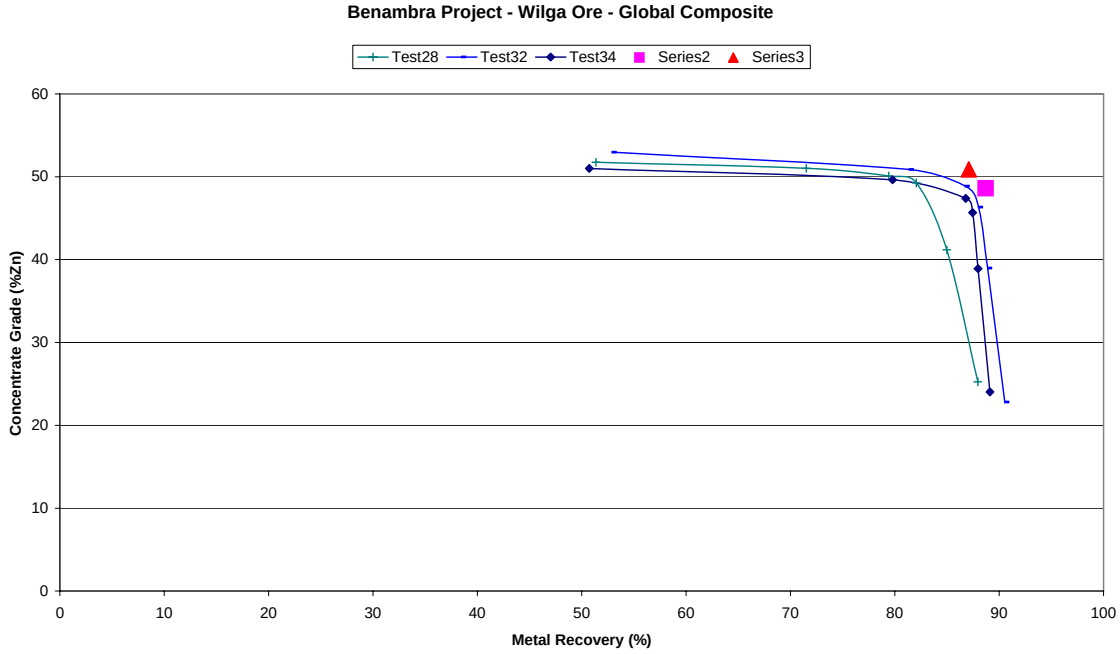
As for the copper flotation stage, the conditions stabilised quickly and showed only minor variations throughout the series.

Locked Cycle Test 2 – Wilga Global Composite – Zinc Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Zn Concentrate Cycle 1	0.77	0.45	48.37	95.51
Zn Concentrate Cycle 2	0.90	0.48	48.08	103.90
Zn Concentrate Cycle 3	0.88	0.41	49.93	99.57
Zn Concentrate Cycle 4	0.91	0.45	48.03	110.41
Zn Concentrate Cycle 5	0.94	0.44	50.39	100.44
Zn Concentrate Cycle 6	0.97	0.47	46.71	108.75

Locked Cycle Test 4 – Wilga Global Composite – Zinc Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Zn Concentrate Cycle 1	0.91	0.45	51.20	91.80
Zn Concentrate Cycle 2	1.06	0.50	52.20	96.92
Zn Concentrate Cycle 3	1.13	0.45	53.00	93.24
Zn Concentrate Cycle 4	1.31	0.45	52.20	101.84
Zn Concentrate Cycle 5	1.20	0.44	51.00	92.88
Zn Concentrate Cycle 6	1.28	0.45	50.70	100.91

As for the copper flotation stage, the overall test performance was assessed for the final two cycle results.

Test Number	Concentrate Grade			Recovery		
	%Cu	%Pb	%Zn	%	%	%
LCT2	1.0	0.5	48.5	4.8	12.0	88.7
LCT4	1.2	0.5	50.9	5.6	10.6	87.2



The locked cycle test results have reproduced very closely, and possibly slightly exceeded the best of the open cycle test results that were achieved from this composite.

4.4.2 Currawong Global Composite

As for the Wilga global composite, two tests were conducted with minor adjustments to conditions as information became available from other tests in the series.

The head grade comparison for these two tests is tabulated below.

Grade Source	Test	%Cu	%Pb	%Zn
Assay		2.11	0.61	6.23
Total Mass Balance	**LCT1**	2.18	0.58	6.14
	LCT3	2.35	0.57	5.83
Cycle 5/6 Product Balance	**LCT1**	2.16	0.58	6.08
	LCT3	2.33	0.59	5.78

The agreement between calculated and assay head is much better for LCT1 than for LCT3 with copper values being higher and zinc being lower in the latter case. The mass recovery for both test series was quite accurate. The total mass balance and the cycle 5/6 values agree suggesting that the difference could be due to reasonably systematic assay variations. However, check assays carried out during comprehensive concentrate analysis (reported later in this report) indicate that the error (if any) was not large. In fact, the largest difference occurs between the two sets of assays for copper concentrate in LCT1.

4.4.2.1 Copper Flotation Results

The first locked cycle test conducted was based on the conditions used for WGC28 that included 10g/t of collector addition to the ultra-fine rougher stage. For the second test in the series, the collector addition to the ultra-fine rougher was eliminated and the addition to the primary rougher was reduced from 40g/t to 30g/t. The addition of MBS to the ultra-fine circuit was increased from 300/150g/t to 400/200g/t.

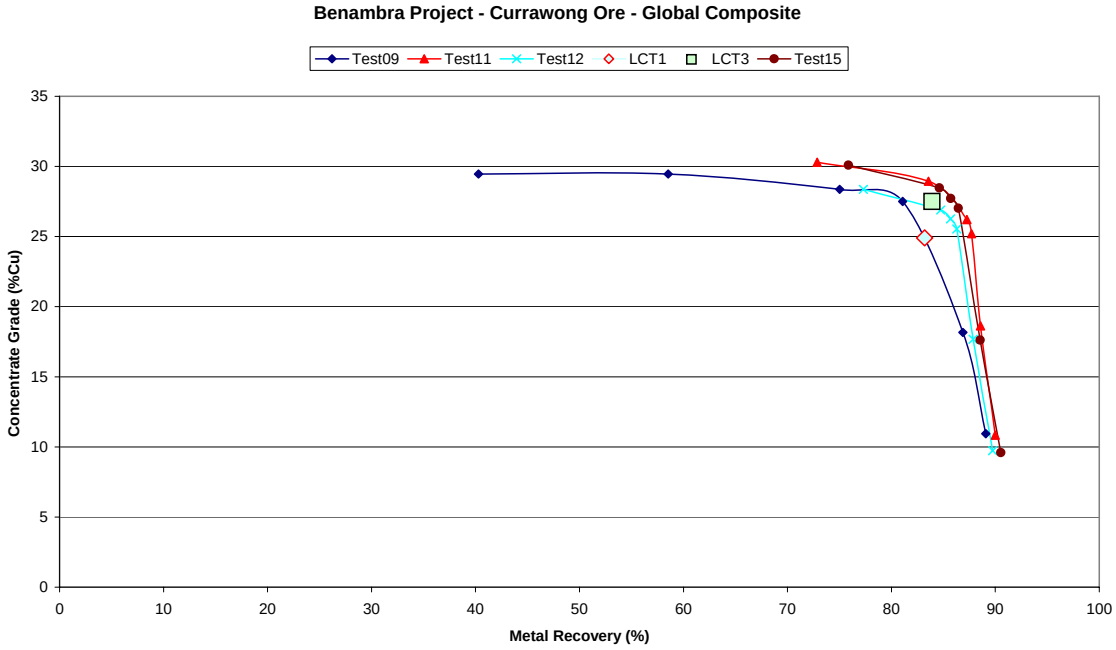
As for the tests on the Wilga global composite, there was no evidence of accumulation of circulating load and only short term instability between cycles.

Locked Cycle Test 1 – Currawong Global Composite – Copper Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Cu Concentrate Cycle 1	26.42	4.15	3.37	70.58
Cu Concentrate Cycle 2	24.75	4.34	3.57	75.95
Cu Concentrate Cycle 3	26.43	4.23	3.01	67.50
Cu Concentrate Cycle 4	21.37	3.87	4.03	82.10
Cu Concentrate Cycle 5	25.40	4.34	3.08	75.03
Cu Concentrate Cycle 6	24.44	4.23	4.51	74.01

Locked Cycle Test 3 – Currawong Global Composite – Copper Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Cu Concentrate Cycle 1	32.85	3.33	2.08	57.74
Cu Concentrate Cycle 2	31.35	3.47	3.01	63.35
Cu Concentrate Cycle 3	31.64	3.90	2.10	69.06
Cu Concentrate Cycle 4	29.05	3.87	2.81	75.73
Cu Concentrate Cycle 5	28.95	4.36	2.43	65.16
Cu Concentrate Cycle 6	25.95	4.15	2.67	67.98

The overall result for the two tests are summarised in the following table and chart.

Test Number	Concentrate Grade			Recovery		
	%Cu	%Pb	%Zn	%	%	%
LCT1	24.9	4.3	3.8	82.9	52.9	4.5
LCT3	27.4	4.3	2.6	85.1	52.4	3.2



The result of LCT3 has reproduced to a high degree the results of open cycle tests that were conducted on this composite.

4.4.2.2 Zinc Flotation Results

The test conditions for zinc flotation were set as for the open cycle test WGC28 for the both tests on the Currawong global composite, LCT1 and LCT3.

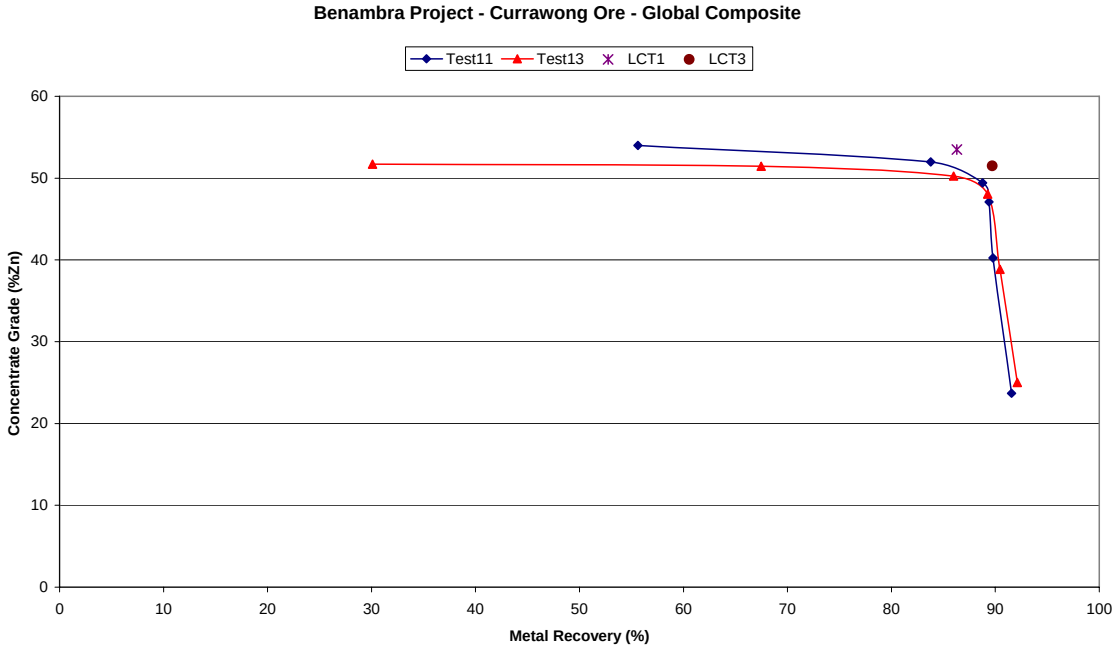
As for the copper flotation stage, the conditions stabilised quickly and showed only minor variations throughout the series.

Locked Cycle Test 1 – Currawong Global Composite – Zinc Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Zn Concentrate Cycle 1	1.08	0.78	53.47	96.03
Zn Concentrate Cycle 2	1.03	0.57	51.28	96.71
Zn Concentrate Cycle 3	1.47	0.87	51.17	100.15
Zn Concentrate Cycle 4	1.24	0.70	50.69	102.11
Zn Concentrate Cycle 5	1.23	0.76	54.11	102.29
Zn Concentrate Cycle 6	1.46	0.71	52.78	100.35

Locked Cycle Test 3 – Currawong Global Composite – Zinc Flotation				
	%Cu	**%Pb**	**%Zn**	**Mass**
Zn Concentrate Cycle 1	1.34	1.62	51.30	98.04
Zn Concentrate Cycle 2	1.23	1.20	51.78	100.64
Zn Concentrate Cycle 3	1.40	0.97	51.54	102.75
Zn Concentrate Cycle 4	1.32	0.85	52.43	101.94
Zn Concentrate Cycle 5	1.46	0.74	51.57	103.04
Zn Concentrate Cycle 6	1.34	0.71	51.34	98.98

As for the copper flotation stage, the overall test performance was assessed for the final two cycle results.

Test Number	Concentrate Grade			Recovery		
	%Cu	%Pb	%Zn	%	%	%
LCT1	1.4	0.7	53.4	6.1	12.3	85.9
LCT3	1.4	0.7	51.5	6.0	12.4	89.5



The results of both of these locked cycle tests have reproduced or even exceeded the best results achieved in open cycle testing on this composite.

4.5 Variability Testing – Wilga Ore

Tests conducted on a range of individual diamond drill hole composites in the Phase I test programme indicated that some variation in flotation performance and, possibly, in test conditions would be experienced as head grade and mineralogy varied. For this reason, a range of these individual drill hole composites was selected for further testing to demonstrate the flotation response to the 'standard' flowsheet and to explore some of the variations in reagents required to achieve the target concentrate grade. The composite selection from the Wilga ore body was done based on head grade since this was indicated to be a likely significant factor in determining the flotation response of the sample.

4.5.1 Composite BERD05

This sample was selected as an example of low copper ore with moderate to high zinc content.

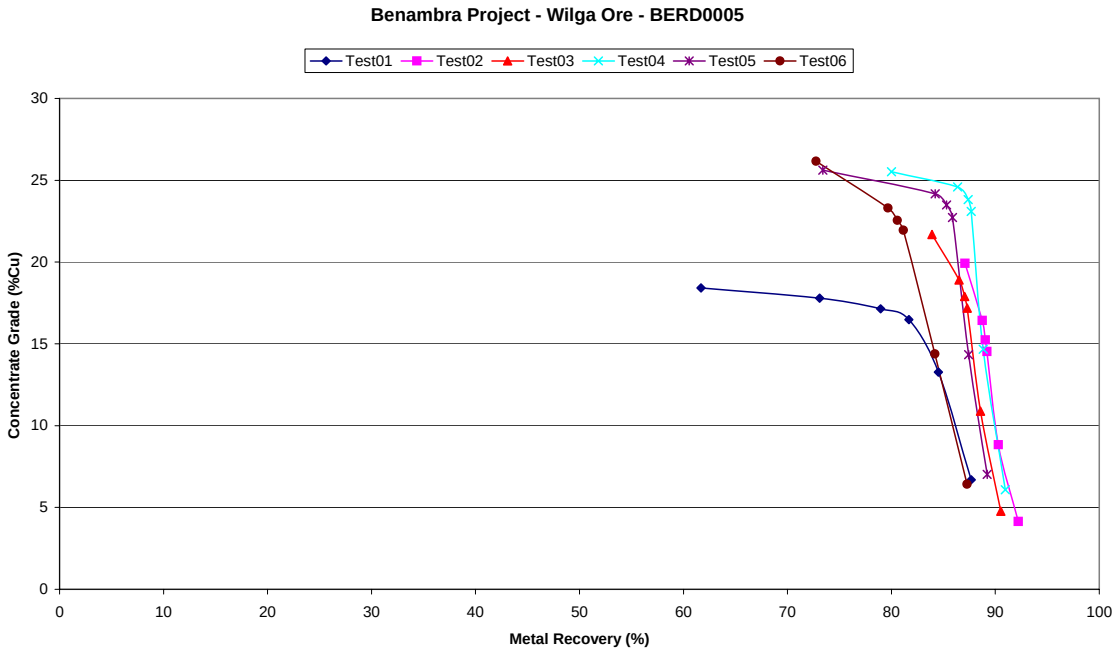
4.5.1.1 Test Results – Copper Flotation

The test result using the 'standard' copper flotation conditions was not satisfactory and a series of tests was necessary to improve the concentrate grade to achieve the target minimum grade of 25%Cu.

4.5.1.1.1 Effect of Collector Addition

The primary emphasis in this phase of testing was to vary the amount and dosing point for collector. The particular aim was to improve the primary flotation concentrate grade to reduce the upgrade ratio that had to be achieved in the ultra-fine flotation stage. Changes were also made to the collector addition to the ultra-fine roughing stage.

TEST	REAGENTS	Copper	
	Collector Addition	Grade %Cu	Recovery %
	g/t		Cu/Zn/Pb
BERD05-02	60/10/30	14.5	89.2/9.3/24.5
BERD05-03	45/10/20	17.2	87.3/7.5/25.0
BERD05-04	25/5/15	23.1	87.7/5.0/22.8
BERD05-05	20/5/10	22.7	85.9/4.8/26.5
BERD05-06	15/0/10	21.9	81.1/4.3/14.6

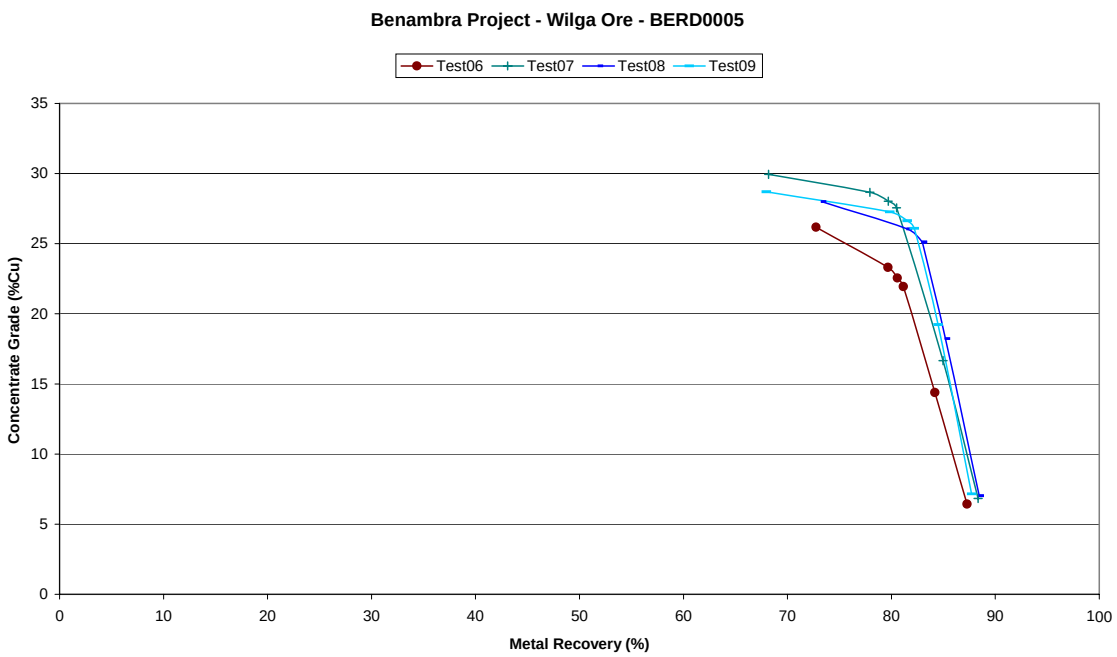


The results of this series show that the primary concentrate grade has been improved by the significant reductions in collector addition to the primary flotation stage. A reduction in recovery has accompanied this, probably in line with the primary grade/recovery curve trend. Although the zinc recovery to copper concentrate has been significantly reduced, the zinc assay of the final concentrate is still high due to the high Zn/Cu ratio in the ore.

4.5.1.1.2 Effect of increased MBS Addition

Because of the high zinc assay of the copper concentrate, the remaining tests in the series were conducted with significantly increased addition of MBS to the primary flotation stage to try to improve even further the selectivity against zinc.

TEST	REAGENTS	Copper	
	MBS Addition	Grade %Cu	Recovery %
	g/t		Cu/Zn/Pb
BERD05-06	2000	21.9	81.1/4.3/14.6
BERD05-07	3000	27.6	80.5/2.6/16.0
BERD05-08	3000	25.1	83.0/3.8/20.3
BERD05-09	3000	26.1	82.2/2.6/22.0

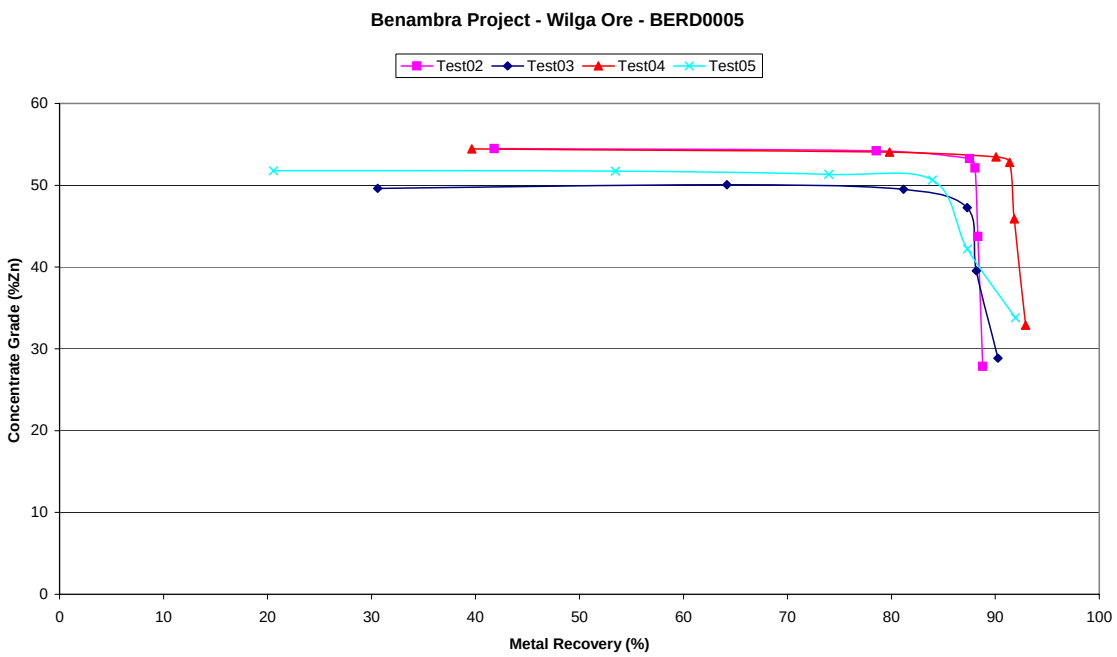


The increase in MBS addition appears to have been effective in improving the copper concentrate grade, although the absolute effect on the zinc assay of the concentrate was relatively small. This suggests that additional depression of pyrite has been achieved although the recovery of lead to concentrate remained at high levels.

4.5.1.2 Test Results – Zinc Flotation

The test conditions for zinc flotation were not changed from the 'standard'. Not all tests were extended to the zinc flotation stage when it became apparent that zinc flotation performance was generally satisfactory.

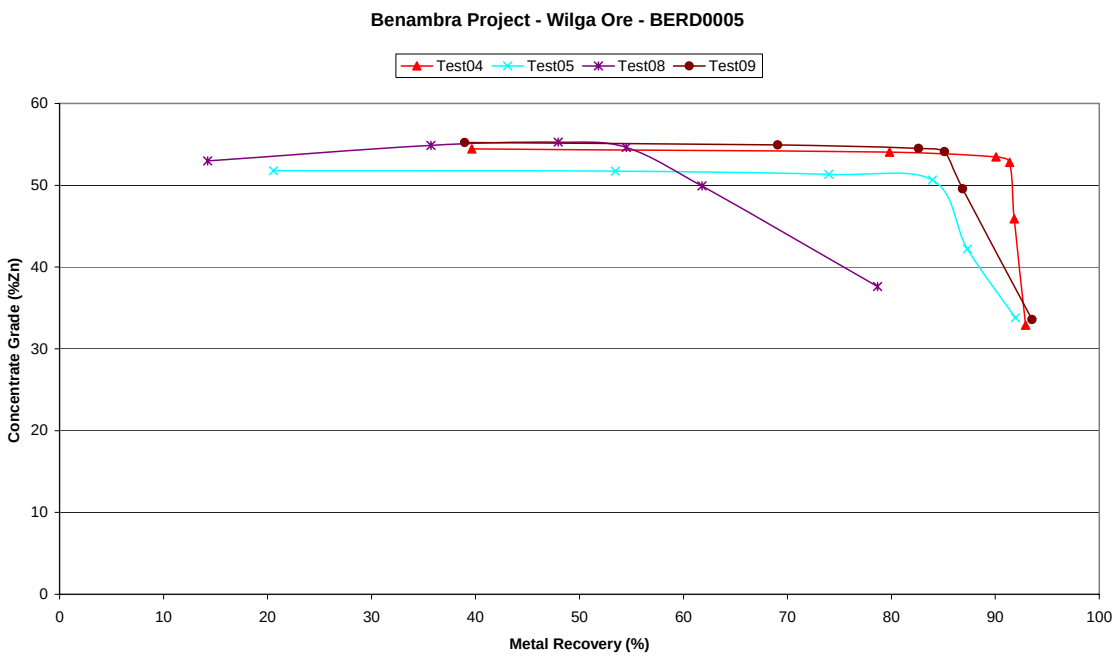
TEST	ZINC	
	Grade %Zn	Recovery %
		Zn/Pb/Cu
BERD05-02	52.1	88.1/25.4/5.9
BERD05-03	47.3	87.3/34.3/7.3
BERD05-04	52.8	91.4/31.2/7.1
BERD05-05	50.7	84.0/34.5/7.5



The zinc flotation performance from this composite is very favourable with high grade primary concentrate being produced at high recovery. This is in line with the higher zinc head grade of the sample. Some of the changes in metal recovery relate to the changes in selectivity in the copper flotation stage. It is likely that the change in ultra-fine flotation recovery is related to the reduction in collector addition that was made in the copper flotation stage across this series of tests. This result confirms the potential for significant dependencies between upstream and downstream flotation requirements.

As a follow up to the effects of changing to a single collector observed on other composites, two tests were conducted to determine the effect on this composite.

TEST	ZINC	
	Grade %Zn	Recovery %
		Zn/Pb/Cu
BERD05-05	50.7	84.0/34.5/7.5
BERD05-08	54.6	54.5/14.6/7.1
BERD05-09	54.1	85.1/20.0/10.0



The changes to the collector regime were not effective in improving flotation performance in this case. In retrospect, this was probably to be expected given the already high concentrate grade and recovery. It does illustrate however that the use of the dual collector system is not universally necessary for equivalent acceptable performance.

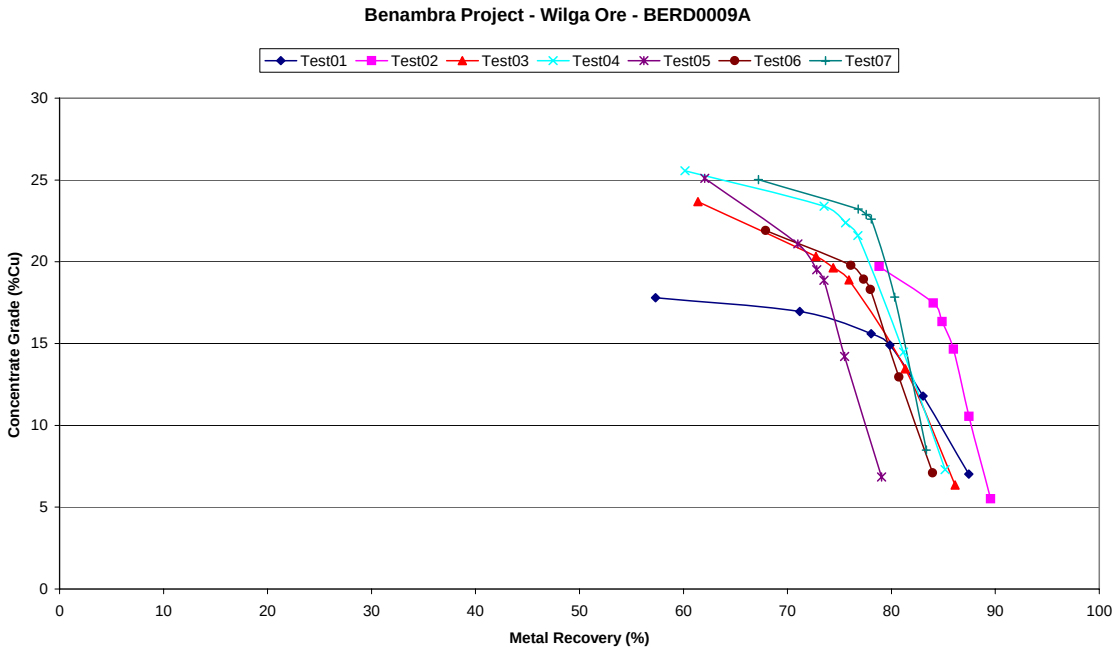
4.5.2 Composite BERD09A

The composite was selected as representing medium grade copper ore with about average zinc head grade.

4.5.2.1 Test Results – Copper Flotation

As for the previous composite, the preliminary 'standard' test was followed by a number of tests intended to improve the copper concentrate grade. The test series followed a similar path of reducing and varying the addition point for the copper collector. The last test in the series also eliminated the addition of CMC to the copper flotation stage.

TEST	REAGENTS	Copper	
	Collector Addition	Grade %Cu	Recovery %
	g/t		Cu/Zn/Pb
BERD09A-02	60/10/30	14.7	86.0/9.2/41.9
BERD09A-03	20/5/10	18.9	75.9/5.1/27.9
BERD09A-04	15/0/10	21.6	76.8/4.2/25.3
BERD09A-05	10/0/15	18.9	73.5/4.5/11.3
BERD09A-06	15/0/15	18.3	78.0/5.2/28.8
BERD09A-07	15/0/15	22.6	78.1/3.8/21.2

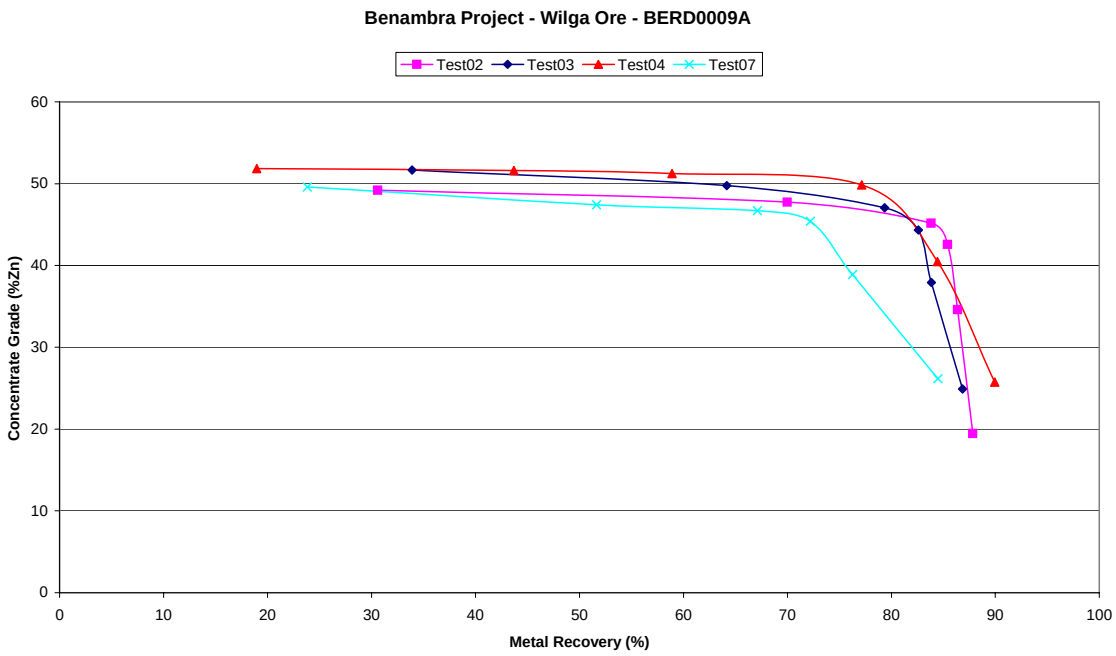


These results illustrate that, at least for this composite, the target concentrate grade for copper concentrate will be only achieved at a lower recovery than from higher head grade samples. This result is in line with the previous observation that the metallurgical performance was likely to be less favourable from lower head grade ore. Also, in line with other observations, it appears that the primary rougher concentrate grade should approach or exceed 10%Cu to allow relatively high recovery to the target concentrate grade.

4.5.2.2 Test Results – Zinc Flotation

The zinc concentrate grade achieved from the 'standard' test conditions was not in line with the target grade and test conditions were varied to try to improve the concentrate grade to the target level. These variations related principally to the addition rate and type of collector.

TEST	REAGENTS	ZINC	
	Collector/ Addition	Grade %Zn	Recovery %
	Type/g/t		Zn/Pb/Cu
BERD09A-02	Mixed/25/25	42.6	85.4/23.8/7.0
BERD09A-03	Mixed/20/20	44.3	82.6/25.2/13.2
BERD09A-04	SIBX/30	49.9	77.1/23.6/11.9
BERD09A-07	Mixed/40/10	45.4	72.2/22.2/10.3



This composite has proved to be relatively difficult to achieve the target final concentrate grade despite being able to achieve relatively high primary rougher concentrate grade. However, the test programme was quite limited and developmental testing was continuing in the copper flotation so that the best possible performance probably has not been achieved.

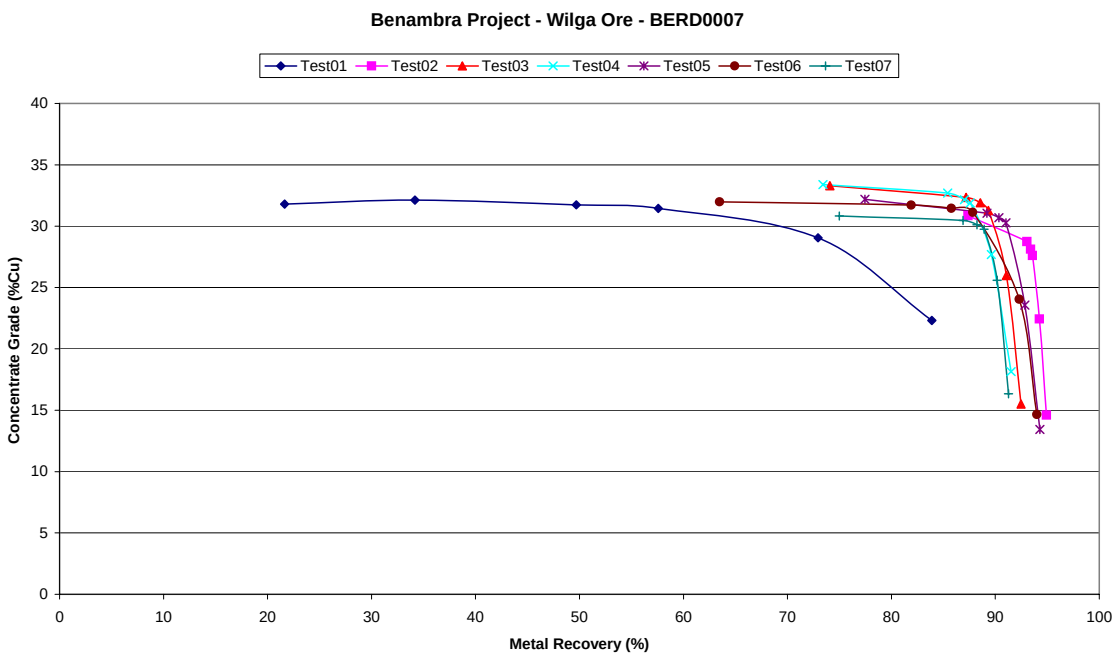
4.5.3 Composite BERD07

This composite was selected to represent high grade copper ore with low grade zinc.

4.5.3.1 Test Results – Copper Flotation

Although the concentrate grade and recovery from the initial 'standard' test conditions were good some minor changes were made to the copper flotation conditions in the form of reduced collector addition. Based on possible flow-on effects to the subsequent zinc flotation stage, the conditions were reverted to the 'standard' after two tests at the reduced conditions.

TEST	REAGENTS	Copper	
	Collector Addition	Grade %Cu	Recovery %
	g/t		Cu/Zn/Pb
BERD07-02	60/10/30	27.6	93.6/14.3/22.0
BERD07-03	50/0/25	31.3	89.3/7.5/21.3
BERD07-04	50/0/25	31.9	87.6/6.2/24.2
BERD07-05	60/10/30	30.3	91.0/7.8/25.5
BERD07-06	60/10/30	31.1	87.8/4.4/17.1
BERD07-07	60/10/30	29.8	88.9/7.0/22.5

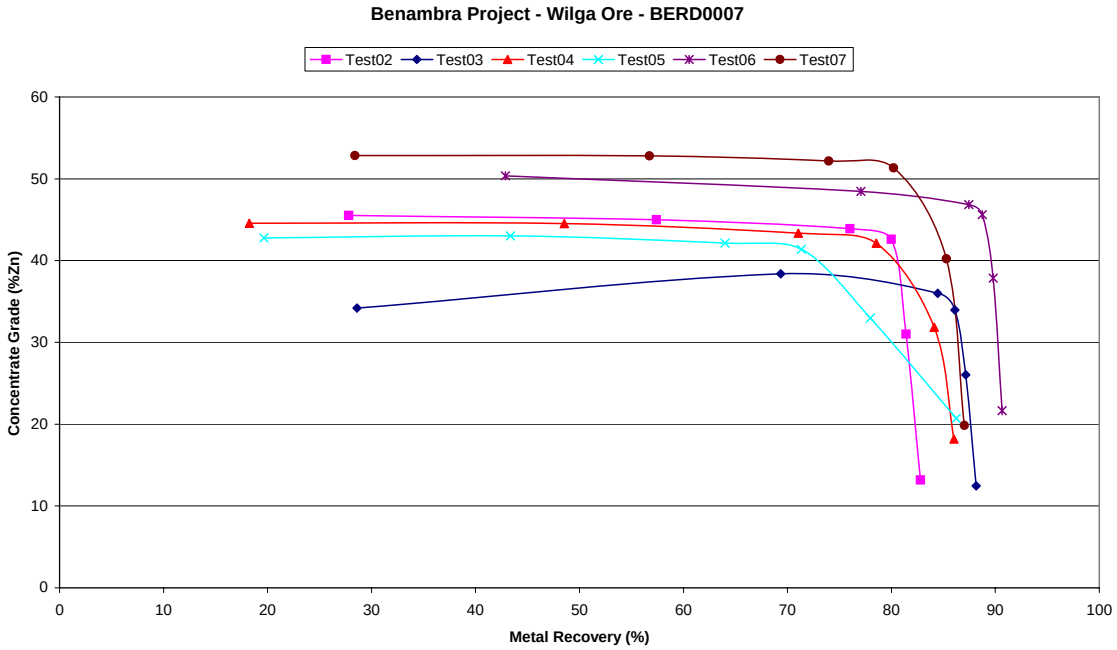


It is clear that the test results reproduce the grade recovery curve quite well although the actual test endpoint is a little variable. The final test in the series was done with no addition of CMC to the copper section. This might explain the reduction in concentrate that was produced. It is not clear why the zinc recovery reduced significantly in test BERD07-06.

4.5.3.2 Test Results – Zinc Flotation

The test programme for zinc flotation, on this composite, involved variations to the addition rate of copper sulphate and to the addition rate and type of collector.

TEST	REAGENTS	ZINC	
	$CuSO_4$/Collector/ Addition	Grade %Zn	Recovery %
	g/t/Type/g/t		Zn/Pb/Cu
BERD07-02	500/mixed/25/25	42.6	80.0/5.2/2.6
BERD07-03	400/mixed/20/20	34.0	86.1/9.7/6.0
BERD07-04	200/A4037/20	42.1	78.6/8.4/6.3
BERD07-05	200/A4037/20	41.4	71.4/7.8/4.2
BERD07-06	500/SIBX/20	45.6	88.7/7.1/6.4
BERD07-07	250/SIBX/15	51.3	80.2/4.7/4.4



These results indicate that the correct balance of reagents will be important for achieving the required target concentrate grade. In particular, it will be important to be able to produce a primary rougher concentrate in excess of 20%Zn. The balance between collector addition and copper sulphate addition is also important, as is the level of collector addition and the balance between the two collector types that have been used. For this particular composite, SIBX alone appears to be superior to the mixed collector types.

4.6 Variability Testing – Currawong Ore

The range of variability composites tested from the Currawong ore body was more restricted. Rather than covering a range of head grades, the composite used for the bulk of the work (BERD10) was selected initially as a closer representation of the resource head grade than the Currawong global composite. The higher talc content of this composite resulted in the further testing of composite CURB135 representing a low talc sample at about resource grade. A sample interval from diamond drill hole BERD12 (BERD12B) was tested because of the high arsenic content occurring in this section of the ore body.

4.6.1 Composite BERD10

The composite BERD10 was selected for particular attention because it was close to the reserve grade for the Currawong ore body and because it also contained a significant proportion of the 'talc' that was present in the Currawong global composite.

4.6.1.1 Test Results – Copper Flotation

The number of tests that examined variations to conditions applied for copper flotation was limited and the majority of variation was applied to the conditions for zinc flotation.

4.6.1.1.1 Effect of CMC Addition

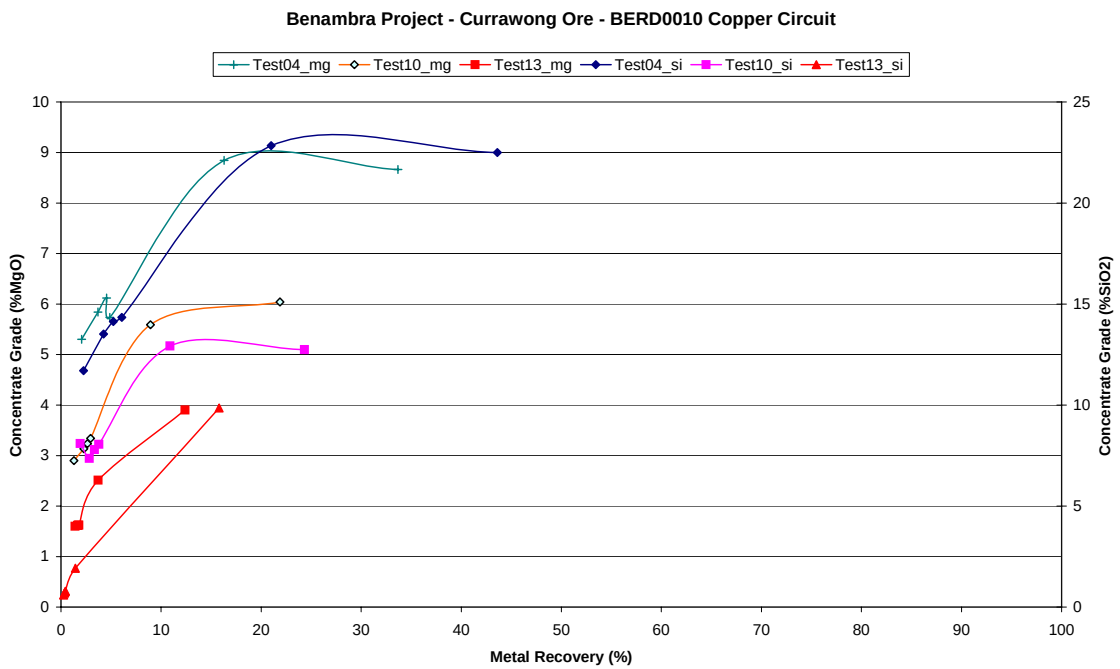
One test was conducted with the 'standard' addition of CMC to copper primary rougher flotation (150g/t). Most subsequent tests reverted to the original higher addition of CMC (250g/t) to primary rougher and (50g/t) to scavenger. Towards the end of the test programme, additional tests were conducted with increased additions of CMC to gauge the potential for, and possible benefits of, improved 'talc' depression.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn/Pb
BERD10-04	23.5	60.8/1.6/30.9
BERD10-05	25.9	82.2/3.1/44.7
BERD10-13	29.4	81.1/3.0/40.6

The results show a significant improvement in both copper concentrate grade and recovery with the initial 'intermediate' addition. Zinc and lead recovery

are also increased though in higher proportion than for the copper. Further improvement in concentrate grade resulted with the increased addition of CMC with little change in the recovery of any of the base metals. It would appear that the 'intermediate' addition of CMC might have been less than optimum although no rigourous optimisation was conducted.

Products from three tests involving use of varying amounts of CMC addition (150/0/50/25, 250/50/50/25, 350/50/75/50) were analysed for magnesia (MgO) and silica (SiO_2). The grade/recovery curves for these species in this group of tests are plotted in the following chart.

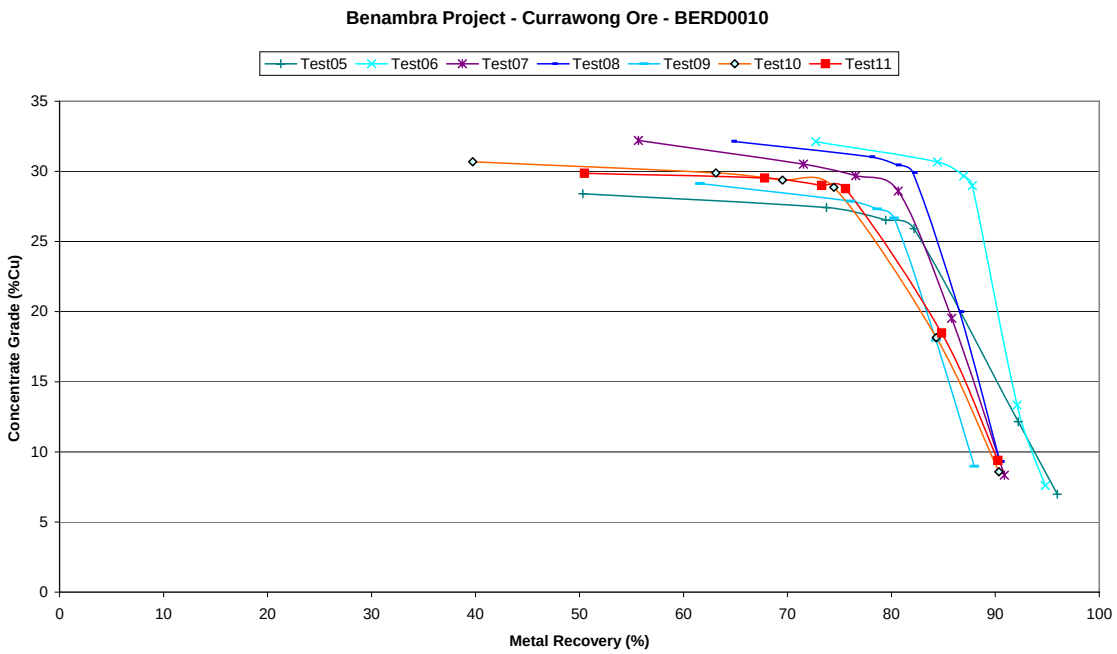


This shows the significant beneficial effect of the increased CMC additions and amply demonstrates that the talc content of the primary rougher concentrate and, ultimately, the final concentrate can be controlled by use of this reagent in the appropriate quantity. The close correspondence between the curves for each test indicates that the two species are intimately related although there is an excess of silica compared to the stoichiometric formula for talc.

4.6.1.1.2 'Standard' Test Reproducibility

The conditions of test BERD10-05 were accepted as the new 'standard' for copper flotation from this composite and all subsequent tests were conducted using this regime.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn/Pb
BERD10-05	25.9	82.2/3.1/44.7
BERD10-06	29.0	87.8/3.3/41.2
BERD10-07	28.6	80.7/3.2/47.6
BERD10-08	29.9	82.1/2.5/38.1
BERD10-09	26.7	80.3/2.3/38.1
BERD10-10	28.9	74.5/1.6/37.0
BERD10-11	28.8	75.6/1.6/31.6



These results show some significant variability, particularly in the performance of the ultra-fine flotation stages. In general, the primary rougher flotation performance appears to be quite reproducible. Possible explanations for variation in ultra-fine flotation performance include variability introduced by the application of ultra-fine grinding and possible effects of small but varying amounts of 'talc' in the concentrate. The laboratory reported significant variations in the grind time required to achieve the target 10μm grind.

4.6.1.1.3 Effect of pre-Flotation

A single test was conducted in which a pre-flotation concentrate was made by floating without collector after conditioning with MBS. This was done to remove the bulk of the free-floating 'talc' to a concentrate that could be examined for mineral identification. The pre-float concentrate removed 14.8% of the sample mass, 8.9% of the copper, 7.8% of the zinc and 10.2%of the lead. After the pre-float stage the 'standard' test conditions were applied

TEST	COPPER	
	Grade %Cu	**Recovery %**
		Cu/Zn/Pb
BERD10-05	25.9	82.2/3.1/44.7
BERD10-12	26.7	74.6/3.9/48.2

The significantly lower copper recovery achieved compared to a 'standard' test is a reflection of the loss of copper that went with the 'talc' pre-float concentrate.

4.6.1.2 Test Results – Zinc Flotation

The first two tests in this series involved using the 'standard' zinc flotation conditions. The results achieved were relatively poor with concentrate grade, in particular, falling short of expectations.

TEST	ZINC	
	Grade %Zn	**Recovery %**
		Zn/Pb/Cu
BERD10-04	37.6	90.8/27.6/29.4
BERD10-05	44.7	88.2/15.2/9.6

These results reflect the effect of the changed CMC additions to the copper circuit both in the effects of changes in metal recovery and probably on-going depression of 'talc' as well.

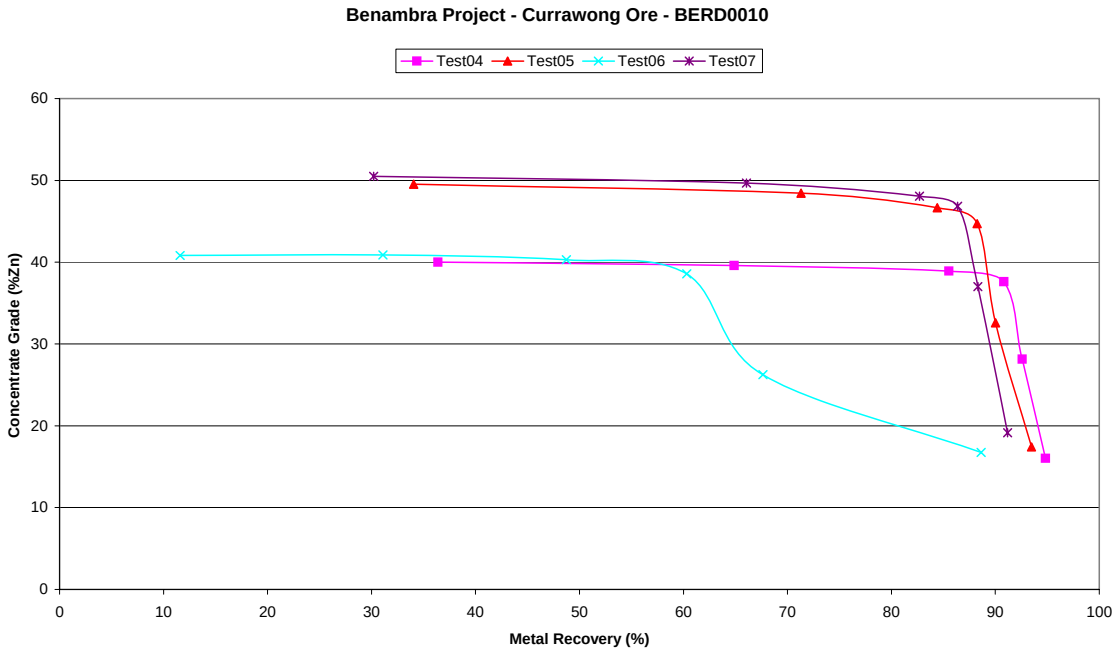
The further tests in this series were directed to improving the flotation performance in the zinc section following 'standard' flotation through the copper section. As noted previously, there were variations in copper section

performance that would have had flow-on effects into the zinc section, causing some variations unrelated to the reagent variations that were made.

4.6.1.2.1 Effect of Activator/Collector Variation

The initial tests investigating the zinc flotation addressed the effects of changes in the addition rate of copper sulphate and the collector mix on flotation performance.

TEST	REAGENTS	ZINC	
	$CuSO_4$/Collector	Grade %Zn	Recovery %
	g/t		Zn/Pb/Cu
BERD10-05	500/50	44.7	88.2/15.2/9.6
BERD10-06	400/40	38.6	60.3/10.4/4.7
BERD10-07	500/60	46.9	86.4/9.5/7.9



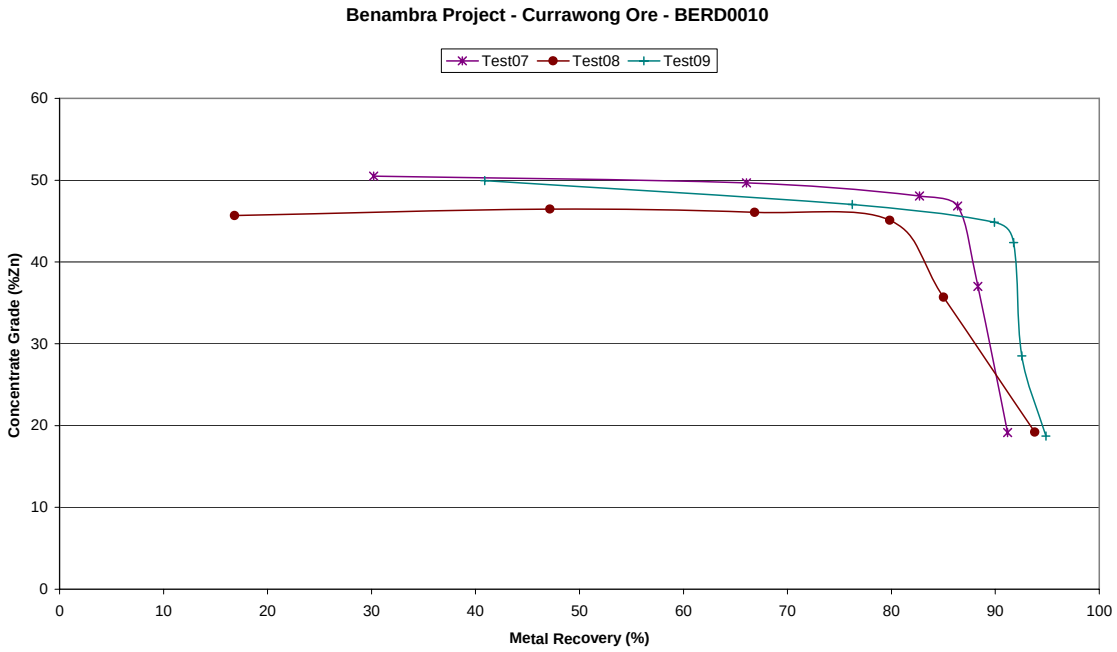
The results are considered to be ambiguous on the basis that the result from BERD10-06 was adversely affected by an unusually high loss with little upgrading in the ultra-fine rougher stage. There was evidence of two quite distinct concentrate grade outcomes but there was no ready explanation for this difference other than due to concentrate dilution.

On the basis of the higher grade concentrate produced from test BERD10-07, the conditions for this test were used as a basis for subsequent testing.

4.6.1.2.2 Effect of CMC Addition

On the basis that 'talc' flotation could be adversely affecting the grade of concentrate being achieved in zinc flotation, tests were conducted using CMC in zinc flotation as well as in copper flotation. The first test included CMC in both primary and ultra-fine flotation. In the second test, the addition to primary flotation was reduced and the addition to ultra-fine flotation was eliminated.

TEST	REAGENTS	ZINC	
	CMC Addition	Grade %Zn	Recovery %
	g/t		Zn/Pb/Cu
BERD10-07	0/0/0	46.9	86.4/9.5/7.9
BERD10-08	150/50/25	45.1	79.8/14.6/8.2
BERD10-09	100/0/0	42.4	91.8/21.4/9.4



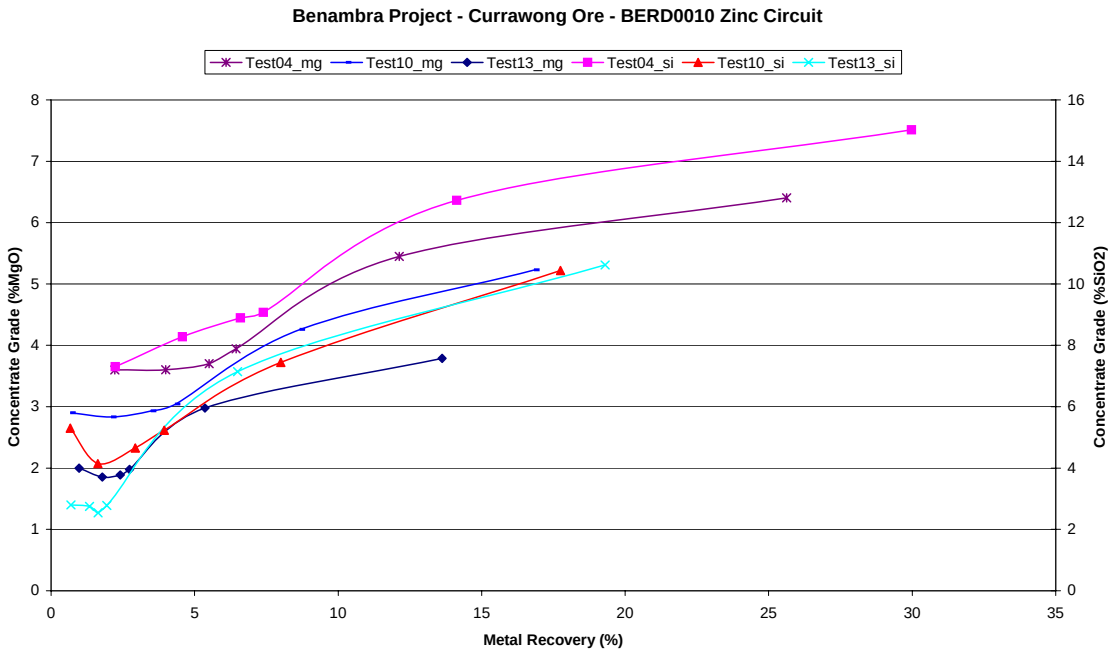
These results suggest that the addition of CMC to the ultra-fine flotation was deleterious in affecting the recovery through the two stages. The addition to the primary rougher appeared to be neutral in its effect. Removal of the addition to the ultra-fine flotation did not entirely restore the overall flotation response.

In the 'talc' pre-float test, the zinc primary float was also dosed with 100g/t of CMC.

TEST	REAGENTS	ZINC	
	CMC Addition	Grade %Zn	Recovery %
	g/t		Zn/Pb/Cu
BERD10-09	100/0/0	42.4	91.8/21.4/9.4
BERD10-12	100/0/0	48.3	84.1/9.4/9.1

The final concentrate grade was one of the best achieved from this composite although the recovery was lower. However, the recovery offset from the earlier tests corresponded to the zinc losses that occurred in the pre-flotation stage.

The magnesia and silica grade/recovery curves for the zinc flotation in the series of tests that were assayed in full for these two species are plotted in the following chart.

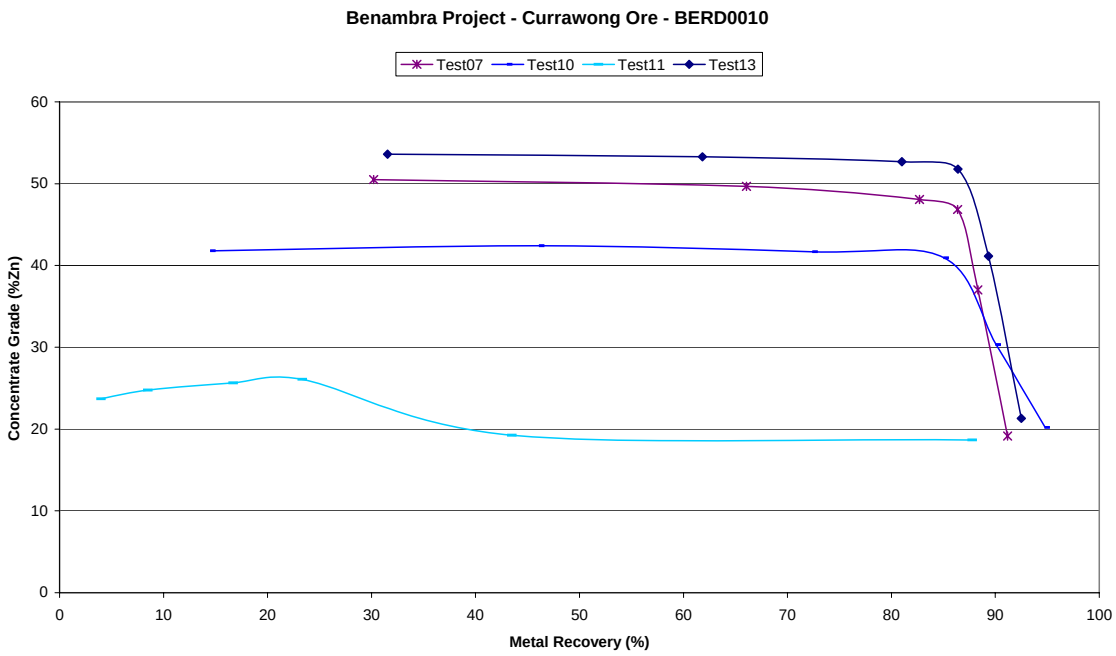


Although the trends are less well defined, these results show that there is a significant downstream effect of the addition of CMC to the copper flotation stage.

4.6.1.2.3 Effect of Collector Addition and Type

Because the primary concentrate grade being produced was relatively low (<20%Zn), some tests were conducted to try to increase the grade to improve the chances of being able to produce concentrate of the required grade after ultra-fine flotation.

TEST	REAGENTS	ZINC	
	Collector/ Addition	Grade %Zn	Recovery %
	Type/g/t		Zn/Pb/Cu
BERD10-07	Mixed/60	46.9	86.4/9.5/7.9
BERD10-10	Mixed/40/10	40.9	85.0/20.4/16.3
BERD10-11	A4037/30	26.1	23.3/11.1/10.5
BERD10-13	SIBX/40	51.8	86.4/17.3/7.3



Changing the addition of the mixed collector appeared to have had a deleterious effect on zinc flotation performance. It is not clear why such a dramatic effect was produced.

The use of A4037 alone was ineffective, particularly during the ultra-fine flotation stages. This result suggests that this reagent is not 'relocated' to fresh surfaces during/after regrinding. On the other hand, the use of SIBX

alone was very effective, producing concentrate grade above target. This result seems to confirm that SIBX is relocated to fresh surfaces during/after regrinding. It should be noted that this test result was achieved following the higher CMC addition to copper flotation and improved talc depression would have helped the result.

4.6.2 Composite BERD12B

This composite was prepared to assess the likely disposition of arsenic bearing minerals in intersections that contained higher than normal grades of arsenic. The arsenic grade of the interval sample was 0.9%As. All other ore intersections considered in the metallurgical test programme had indicated arsenic assays below 0.2%As.

4.6.2.1 Test Results – Copper Flotation

Only a single test was conducted with the 'standard' test conditions.

TEST	COPPER	
	Grade %Cu/As	Recovery %
		Cu/Zn/Pb/As
BERD12B-01	24.4/0.33	90.0/4.0/13.6/2.5

This test indicates that the arsenic mineral (arsenopyrite) has been well depressed although the level of arsenic in the concentrate is relatively high. It is considered that, with normal mining practice and blending, the presence of arsenic will not prove to be a problem although the distribution and proportion of high arsenic ore should be monitored.

4.6.2.2 Test Results – Zinc Flotation

Zinc flotation conditions were also applied without modification to the 'standard'.

TEST	ZINC	
	Grade %Zn/As	Recovery %
		Zn/Pb/Cu/As
BERD12B-01	51.2/0.43	75.8/64.7/5.8/8.8

A significantly higher proportion of arsenic was recovered to the zinc concentrate and the concentrate assays would be sufficient to cause concern

for the marketability of the concentrate. Once again, the dilution possible through mining and blending should be sufficient to ameliorate any difficulties but the distribution and proportion of high arsenic ores should be monitored.

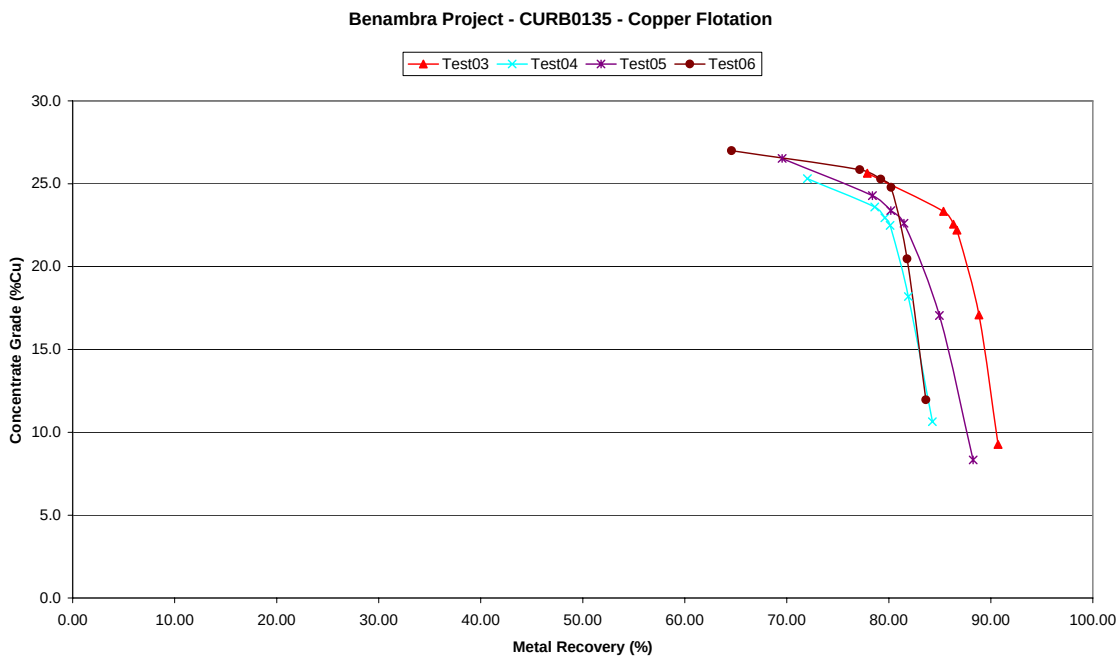
4.6.3 Composite CURB135

This composite was prepared from the core of diamond drill hole B135 that was retrieved from the core shed after storage under ambient conditions on site for several years. Additional tests were conducted on this sample as representing ore at about 'reserve' grade but without the complication of high 'talc' that existed in composite BERD10.

4.6.3.1 Test Results – Copper Flotation

The copper flotation test programme reflected relatively minor changes to the amount and distribution of collector between the primary rougher and the ultra-fine flotation circuit. The last test in the series involved a significant increase in the addition of MBS to the primary rougher together with the elimination of the CMC addition. The latter test represented an attempt to achieve an improvement in concentrate grade without affecting, or even improving, copper recovery.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn/Pb
CURB135-03	22.2	86.7/4.9/47.5
CURB135-04	22.5	80.1/4.3/40.7
CURB135-05	22.6	81.5/4.3/36.9
CURB135-06	24.8	80.2/3.3/36.1

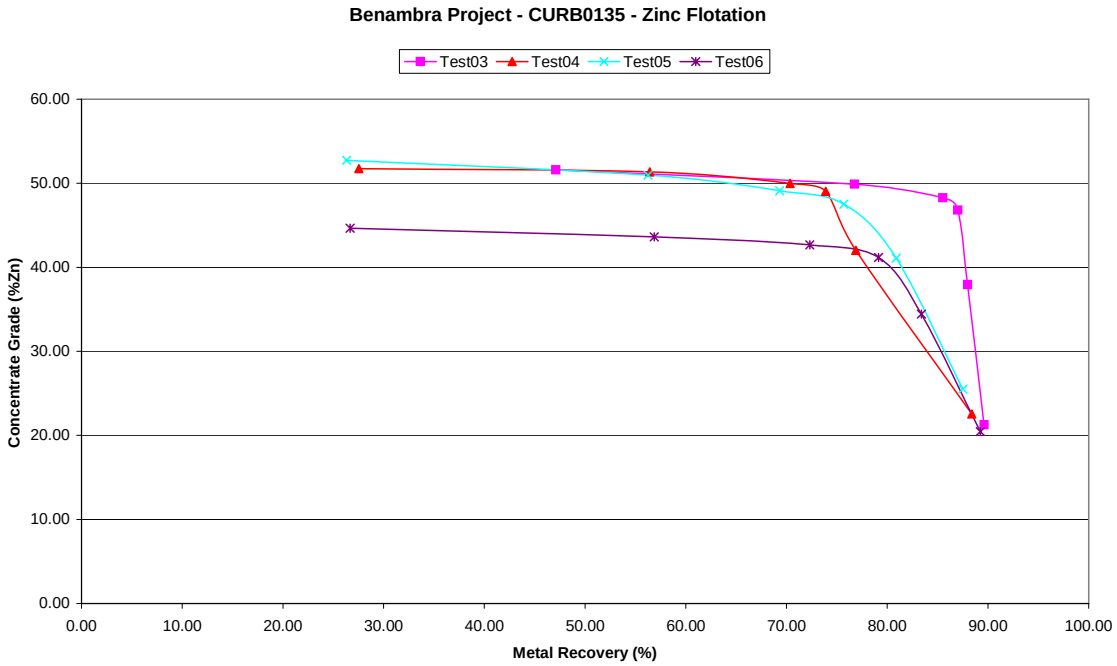


These test results have demonstrated that it is possible to produce the required minimum concentrate grade of 25%Cu from this composite. However, it is likely that the test conditions have not been optimised and further improvements in performance might be possible. It should also be kept in mind that the storage conditions for the drill core that made up this sample are considered less than ideal.

4.6.3.2 Test Results – Zinc Flotation

The test programme for zinc flotation concentrated on variations in reagent additions to try to demonstrate the ability to produce the minimum zinc concentrate grade of 50%Zn. Addition of copper sulphate and the quantity and type of collector were the variables tested.

TEST	REAGENTS	ZINC	
	$CuSO_4$/Collector/ Addition	Grade %Zn	Recovery %
	g/t/Type/g/t		Zn/Pb/Cu
CURB135-03	500/mixed/25/25	46.8	87.0/14.9/6.8
CURB135-04	500/SIBX/30	49.0	73.9/13.5/6.2
CURB135-05	500/SIBX/40	47.5	75.7/13.5/6.3
CURB135-06	300/mixed/40/10	41.1	79.1/15.0/9.6



The changes to the reagent regime did not prove to be particularly effective in improving the primary concentrate or the final concentrate grade. In test CURB135-05, an improvement in primary concentrate grade did not translate to an improvement in final concentrate grade. Partial reinstatement of the secondary collector (A4037) did not restore ultra-fine flotation performance. The result appears to have been positively deleterious with respect to final concentrate grade.

It is possible that more testing could have produced further improvements in recovery at the target concentrate grade, however it was decided that, given the provenance of the sample, continuing with the programme at this stage was not warranted.

4.7 Mineralogy

During the Phase I test programme, a sample of head size fractions from sample BERD08 was submitted to Julius Kruttschnitt Mineral Research Centre (JKMRC) for analysis using their scanning electron microscope based mineral liberation analysis (MLA). Similar samples produced from the Currawong global composite sample were prepared and submitted during the current programme. In order to assess the potential upgrading characteristics of the primary rougher concentrates that were produced, a concentrate was prepared from the Wilga global composite and corresponding size fractions were also analysed using this technology.

The feed sample fractions from the composite BERD10 were re-examined by Gary McArthur using optical mineralogy. A sample of the pre-float (talc) concentrate was also examined in an attempt to identify the naturally floatable

mineral. The refloat concentrate was also submitted for semi-quantitative XRD analysis.

Because of the apparent difficultly in recognising the talc mineralisation in the drill core, a range of samples was submitted for determination of infra-red spectra using a field instrument (Portable Infra-red Mineral Analyser (PIMA)) developed by Integrated Spectronics Pty Ltd.

4.7.1 MLA Mineralogy

The JKMRC operates a mineralogical service based upon automated scanning of mounted particles using a scanning electron microscope, which allows large numbers of particles to be counted and accurate determination of particle composition. The data collected is analysed to present mineral liberation and mineral associations as well as other information relating to particle specific gravity, mineral analysis and particle size distribution among other parameters. The particle and mineral analyses can be combined to calculate limiting grade/recovery curves for the size fractions analysed. It is possible, with some assumptions to estimate the limiting flotation performance for the original sample.

4.7.1.1 Mineral Liberation

The liberation of a mineral from this type of analysis is conventionally estimated by reference to a standard Cumulative Liberation Yield (CLY). This parameter is defined as the proportion of the total number of mineral grains that contain the mineral of interest, which contain at least the nominated percentage of that mineral. A convention that is generally accepted is that the liberation is given by CLY_{90}.

4.7.1.1.1 Composite BERD08

As foreshadowed in the previous report, the data reported for the fine fraction of the sample from BERD08 (Wilga ore) was in error and the new results are reported below (previous result in brackets).

Size Fraction	**Mineral Liberation (CLY_{90})**		
μm	**Chalcopyrite**	**Sphalerite**	**Pyrite**
-45+29	42.6	49.0	88.0
-29+15	56.3	62.9	89.0
-15+8	81.0 (58.7)	82.3 (58.9)	87.2 (79.1)

The new data fits closely with the expectations from the optical mineralogy data. It also confirms the conclusion that a regrind product size finer than 15μm is required for a high degree of liberation of chalcopyrite and sphalerite.

4.7.1.1.2 Currawong Global Composite

Size fractions from a head sample from the Currawong global composite were also submitted to JKMRC for MLA analysis. The liberation results are tabulated below.

Size Fraction	**Mineral Liberation (CLY_{90})**		
μm	**Chalcopyrite**	**Sphalerite**	**Pyrite**
+30	52.7	54.9	87.8
+16	76.4	76.1	91.6
+8	84.5	87.2	92.6

These results follow the trend established for the earlier sample from the Wilga ore body. Based on these results, both the chalcopyrite and the sphalerite are slightly better liberated than for the sample from BERD08. This difference could result in improved metallurgical performance and there is some evidence from the metallurgical testing results that the Currawong composite gave slightly superior results to those achieved from the Wilga ore.

4.7.1.1.3 Wilga Primary Rougher Concentrate

A sample of primary rougher concentrate from the Wilga global composite was prepared and the size fractions were subjected to MLA analysis.

Size Fraction	**Mineral Liberation (CLY_{90})**		
μm	**Chalcopyrite**	**Sphalerite**	**Pyrite**
+30	54.1	34.0	67.3
+16	73.1	59.5	79.0
+8	87.3	80.7	86.5

The trends evident from the feed samples with respect to liberation and particle size are confirmed in this data. Although the samples are not strictly comparable with the other material analysed, there is some evidence of the

selection process into the primary rougher concentrate with slightly higher liberation of chalcopyrite and lower liberation of sphalerite and pyrite.

4.7.2 Talc Identification

A number of methodologies were applied to try to confirm the presence of talc as a component of the ore and to identify a possible method for field identification and analysis.

4.7.2.1 Semi-quantitative XRD

A sample of the talc pre-concentrate, produced from the composite BERD10, was submitted to Amdel for semi-quantitative XRD analysis to identify the main minerals present. The method gave a positive identification of talc. The other minerals identified are listed below together with the relative abundance reported.

Mineral	**Abundance**
Pyrite	Dominant
Talc	Sub-dominant
Chalcopyrite	Trace/accessory
Chlorite	Trace/accessory
Magnetite	Trace
Stilpnomelane	Trace
Quartz	Trace
Galena	Trace
?Ankerite	Trace

NOTE: Ankerite was identified in the MLA suite of minerals

4.7.2.2 Optical Mineralogy

The composite head size fractions from composite BERD10 and a sample of the talc pre-float concentrate were examined by Gary McArthur of McArthur Ore Deposit Assessments Pty Ltd (MODA) with specific emphasis on identifying the talc amongst the non-sulphide gangue minerals. The major

non-sulphide minerals identified in the head sample and their relative proportions are listed below.

Non-sulphide Mineral	Approximate Proportion
Carbonate	40-45%
Chlorite	35-40%
Talc??	10-15%
Sericite	<5%
Quartz	<5%

Talc and sericite are reported to be difficult to identify and distinguish using optical mineralogy. This caveat also applied to the examination of the pre-float concentrate that showed abundant soft pliable phyllo-silicate that could be either mineral. As both minerals are known to be present, it has not been possible to make a reasonable quantitative determination.

4.7.2.3 PIMA Spectral Response

A portable infra-red spectral analyser (PIMA) is used for identifying and quantifying the content of alteration minerals associated with hydrothermal and porphyry mineral deposits. Samples were provided to the equipment manufacturer and to a service provider to assess the possible application of this technology for detecting and/or quantifying the presence of talc in Benambra ore.

The PIMA equipment manufacturer (Integrated Spectronics Pty Ltd) was provided with crushed reject pulps from the core intersection that represented the composite BERD10. They were also provided with a sample of the talc pre-float concentrate. The service provider (Ausspec Pty Ltd) was provided with samples of flotation tailings from BERD10 ('high' talc) and Wilga global composite ('low' talc/talc free) as well as a sample of the pre-float talc concentrate.

The spectra determined were reported to be 'noisy', probably due to the dark colour of the ore. It was possible to detect the presence of talc in the BERD10 tailings and the pre-float concentrate but it was not possible to provide even a semi-quantitative estimate. Talc could not be detected in any of the crushed reject samples (representing <= 1m intervals). Chlorite was detected in a group of samples close to the footwall of the intersection.

It was concluded that the PIMA technology does not offer a reliable method of detecting/measuring the presence of talc in the massive sulphide Benambra ore.

4.8 Tailings Re-treatment*

*** Note: In order to provide a comprehensive overview on tailings retreatment, the report on phase 1 testwork has been combined with the phase II report in this summary document.**

During the operation of the Benambra concentrator some 700 000 tonnes of tailings were placed in the tailings storage area. It is estimated that the average grade of this material is about 1%Cu and about 4%Zn. Some sections of the dam are expected to contain values significantly higher than this average although whether such high grade 'lodes' could be reclaimed separately is probably doubtful.

4.8.1 Samples Collected

In order to assess the potential for recovering saleable concentrates from this material, three shallow samples were obtained using a hand operated spear from a dinghy. Because of the crude method available for collecting a sample, the sample locations were selected in the shallower areas of the covering water. Because of this, the samples obtained were likely to be coarser than normal due to being located near to tailings pour points and they would most likely represent material that had been deposited during the closing stages of the operation. In addition, there was a possibility that the samples would be 'contaminated' by lime that was added to the tailings pond after the operation ceased in order to raise the pH of the covering water pond.

The samples were thoroughly mixed and sub-sampled into 1kg lots in preparation for exploratory metallurgical testing. Samples were submitted for analysis. Size distributions were determined and composite fractions were prepared for mineralogical analysis.

SAMPLE	**HEAD ASSAY**		
	%Cu	**%Zn**	**%Fe**
BET1	0.75	4.69	25.10
BET2	6.77	21.13	18.46
BET3	1.16	7.59	28.03

The composite BET2 was obviously anomalous and no further work was commissioned other than a check assay that confirmed the original determination. It is not clear how such contamination could have occurred, unless some concentrate clean up was deposited after the plant had ceased operation.

The size distribution of the sample BET1 was relatively normal while the size distribution of the composite BET3 was significantly coarse with 40% of the mass reporting to the +28μm fraction. This must represent quite severe segregation, as there was no evidence of particle agglomeration reported by either the metallurgical or the mineralogy laboratory.

4.8.2 Preliminary Testing

As a first approach to test the flotation response of the two samples, the general reagent scheme developed for the fresh ore samples was to be applied but with a pre-float stage with only frother added. This was intended to remove any organic material that might have accumulated in the tailings disposal area since mine closure. This would also serve to remove any talc that might be present in the samples.

TEST	COPPER		ZINC	
	Grade %Cu	Recovery %	Grade %Zn	Recovery %
		Cu/Zn		
BET1-01	8.9	20.7/1.6	31.8	79.8
BET3-01	10.4	9.1/0.6	42.7	65.1

The results for copper flotation were quite disappointing particularly with respect to copper recovery. In test BET1-01, there were high losses through both the cleaning and re-cleaning stages (Rougher recovery 51.1%). For sample BET3 the rougher recovery was low and the re-cleaner response was quite abnormal with concentrate grade improving with flotation time. This response might be related to the abnormal size distribution of the sample or possibly due to the presence of a fast floating low grade material such as talc. The zinc rougher recovery was quite satisfactory and the upgrading achieved was reasonably satisfactory. The losses in the re-cleaner stage for BET3 were quite high and adversely affected the result. There is no clear explanation for these losses.

4.8.3 Surface Cleaning

The next stage of testing involved applying a short grinding time to the sample prior to flotation. The pre-float stage was eliminated on the basis that there was no visual evidence of organics or talc floating in the preliminary tests.

TEST	COPPER		ZINC	
	Grade %Cu	Recovery %	Grade %Zn	Recovery %
		Cu/Zn		
BET1-02	7.8	13.6/1.0	37.6	83.0
BET3-02	9.5	41.7/2.6	44.4	82.2

The result for copper flotation from BET1 was inferior, despite the rougher recovery being higher (58.7%). There is no obvious reason for the high losses in cleaning and re-cleaning but it is thought that regrinding and additional collector might produce a significant improvement in performance. The zinc flotation performance was apparently improved by the pre-treatment.

The copper flotation from sample BET3 was significantly improved compared to the untreated sample. It is not clear why such an improvement was forthcoming although the natural pH of this sample is significantly lower than that of BET1. It is possible that sample BET1 includes residual lime from the additions that were made to the tailings dam after operations ceased. The cleaning response of this sample was also more 'normal' with relatively low losses although the concentrate grade produced was still very low.

4.8.4 Rougher Concentrate Re-grinding

One test was carried out on sample BET1 to test the effect of regrinding on the copper cleaner performance of this sample. The test did not proceed to zinc flotation. An accurate regrind time determination was not made and a thief sample was taken to assess the progress of the re-grinding. The notional product size was p_{80} 15μm.

TEST	COPPER	
	Grade %Cu	Recovery %
		Cu/Zn
BET1-03	9.1	18.5/1.2

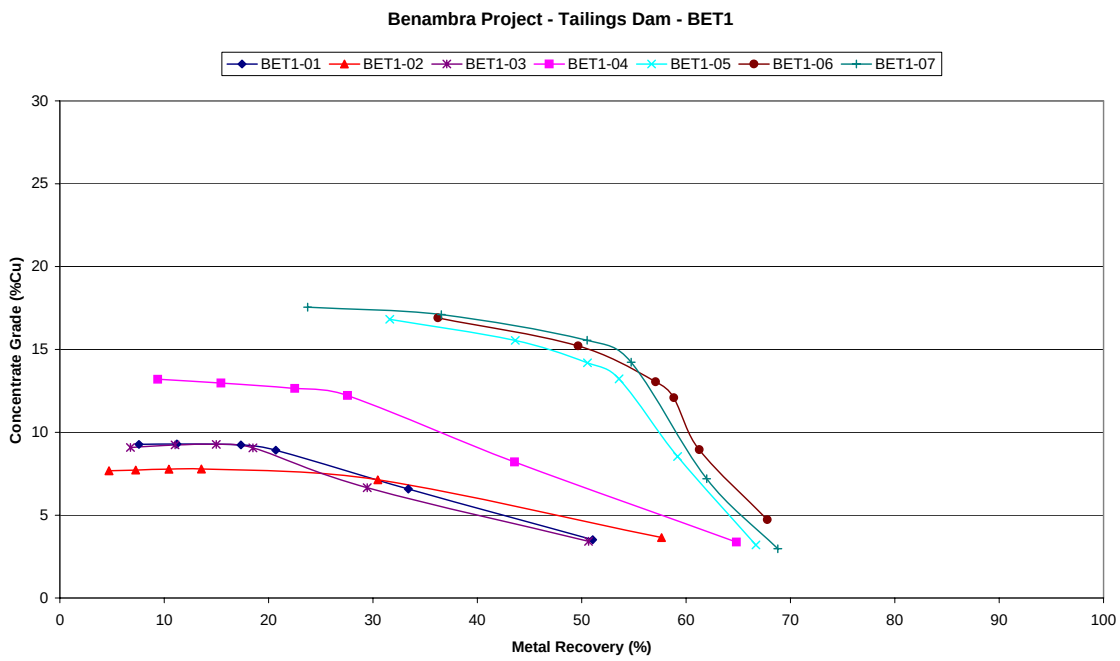
The concentrate grade achieved and the losses in the cleaning stages were not affected by the re-grinding stage.

4.8.5 Phase II Testing

The test programme that was started during Phase I was extended by conducting a limited number of tests on the sample BET1. The zinc flotation performance from the limited tests done previously was relatively promising and the copper flotation performance was poor. Because the economic viability of tailings treatment was heavily dependent on revenue from copper recovery, the tests conducted in this programme were done for copper flotation only. There was an expectation that, if reasonable copper performance could be achieved, adequate zinc results would follow with relative ease.

A group of three tests was done to investigate the effect on copper flotation of significant variations to the conditions that were tested in the Phase I programme. The pre-grind time was extended, the pre-float was eliminated and the MBS addition to the primary rougher stage was reduced to 'normal' levels (2000g/t). The primary rougher concentrate grind size was reduced to 10μm and the addition of MBS and collector to the ultra-fine flotation stage was increased (Test BET1-04). These changes were extended by changing the collector used to RTD12 (Test BET1-05) and further increasing the MBS addition to the ultra-fine flotation stages (Test BET1-06).

TEST	**COPPER**	
	Grade %Cu	**Recovery %**
		Cu/Zn
BET1-03	9.1	18.5/1.2
BET1-04	12.2	27.6/1.5
BET1-05	12.0	53.6/9.9
BET1-06	12.1	58.8/9.7



The initial changes made were effective in improving the grade of concentrate produced with a moderate increase in the copper recovery. The use of the stronger collector was effective in improving the recovery of copper but at the expense of increased recovery of zinc. The net effect was that the concentrate grade was unchanged.

The final test in this series involved grinding the primary rougher concentrate to 6µm and increasing the rate of MBS and collector addition to the ultra-fine flotation stages (Test BET1-07).

TEST	**COPPER**	
	Grade %Cu	**Recovery %**
		Cu/Zn
BET1-07	14.2	54.7/4.1

This result was an improvement in terms of the concentrate grade achieved. However, the product was still far from marketable quality and the recovery of copper was still low. The possibility of achieving significant improvements in either concentrate grade or recovery, to levels sufficient to meet the requirements for economic success of tailings retreatment, was assessed. The major improvements required and the apparent limits encountered resulted in a decision to terminate the investigation at this point in favour of concentrating on process development for ore treatment.

5 ENGINEERING DATA COLLECTION

In conjunction with the development and optimisation of the test conditions for copper and zinc flotation, a number of test programmes were commissioned on products from flotation testing to provide some data suitable for equipment selection and process design.

From the locked cycle tests on the Wilga and Currawong global composite samples, samples of copper and zinc final concentrate and final tailings were prepared for thickening tests. The concentrate samples were then used for conducting filtering tests. The remnants of this material were then used as a basis for the preparation of sufficient concentrate to allow testing of transportable moisture limit. Samples of final concentrates were also submitted for comprehensive elemental analysis.

A number of copper and zinc primary rougher concentrate samples were produced for submission to grinding mill manufacturers for ultra-fine grinding tests.

A bulk sample of ore that had been stored on site since the mining operation ceased in 1996 was used to confirm basic crushing and grinding properties of the whole ore.

5.1 Thickening Test Results

The test programme for thickening performance was conducted by Outokumpu Technology who were the suppliers of the existing thickeners on site and who were in possession of operating and laboratory test data generated during the previous mine operation. The tests programme included simple screening of flocculant types on the tailings sample and spot checks with the selected grade on the concentrate samples. The tailings sample was also used for thickener feed density determination. Conventional cylinder settling tests were done on the two concentrates while continuous high rate thickener tests were conducted on the tailings sample. Spot checks of the high rate thickening performance of the concentrates were also conducted.

The following two sections are a copy of the Discussion and Conclusions from the Outokumpu Supaflo® Thickener test report S0108 (refer to this report for full details and results).

5.1.1 <u>Concentrates</u>

<quote>

The cylinder settling tests indicated that low dosages of Magnafloc 1011 (or equivalent) could be used to achieve satisfactory settling rates and clarities.

A minimum dosage of approximately 8 g/tonne of flocculant was required to achieve good clarity for both copper and zinc concentrates.

Increased floc addition enabled higher solids flux rates to be used to theoretically achieve the same (55%w/w) underflow density. In practise, this would mean smaller thickeners at increased floc dosages.

Continuous bulk testing confirmed that at commonly accepted (and average) industry solids flux rates (0.23 t/m2/hr) underflow densities of over 62% could be achieved. This is higher than the cylinder settling tests due probably to both higher bed compression and lower loading rates.

The samples used did show a propensity to froth, and this behaviour has traditionally limited loading rates to thickeners in order to avoid adverse clarity and loss of product.

The copper concentrate formed small tight bubbles and whilst higher solids flux and smaller thickeners are theoretically possible for the copper concentrate application, we would recommend a limitation on solids flux to 0.25t/m2/hr.

Similarly for zinc, during some bucket transfer of samples the froth formed had the nature of predominantly larger bubbles (10-15mm size) which although were more easily broken did persist during sample mixing. Again, we would suggest a more conservative loading rate of a maximum of 0.15 t/m2/hr for the zinc.

Recommended Design Criteria – Copper Concentrate

Feed density:– In the range 8 to 15% w/w solids

Solids Flux : Maximum 0.25 t/m2/hr

Floc Dosage : 8-10 g/tonne dry solids (powder floc)

Underflow Density: 61% w/w

Recommended Design Criteria – Zinc Concentrate

Feed density:– In the range 8 to 15% w/w solids

Solids Flux : Maximum 0.15 t/m2/hr

Floc Dosage : 8-10 g/tonne dry solids (powder floc)

Underflow Density: 62% w/w

<unquote>

5.1.2 Tailings

<quote>

The tailings material settled very readily to densities between 60 and 65 % w/w solids with low to moderate floc dosages at 10-15 g/tonne.

If the aim of the tailings thickener is to maximise water recovery (underflow density of 64 %w/w) then we suggest a solids flux rate of 0.7 t/m2/hr be adopted for design. If the project can tolerate a slightly lower underflow density of around 60-61 % w/w solids then a smaller thickener could be used based on a flux rate of 0.9 t/m2/hr.

Recommended Design Criteria – Tailings

Feed density:– In the range 10 to 12% w/w solids

Solids Flux : Nominal 0.7 t/m2/hr, (Maximum 0.9 t/m2/hr)

Floc Dosage : 10-15 g/tonne dry solids (powder floc)

Underflow Density: 64% w/w

<unquote>

5.2 Filtering Test Results

After the thickening tests were completed, the concentrate samples were forwarded to Larox Pty Ltd as the suppliers of the existing filter on site. When the tests at Larox were completed, the samples were then forwarded to Jord Australia Pty Ltd for testing from an alternate filter supplier.

The following sections summarise the results of these tests and the original reports should be referred to for details.

5.2.1 Larox Filter Tests

<quote>

A Larox pressure Filter with 60mm chamber would be ideal for both concentrate applications. It would also be possible to alternate between the two duties in one filter. In both cases, cake thickness of 45mm in the 60mm chamber would be achievable, and the filter should be sized on this basis. The 60mm chamber enables a much higher throughput per unit area. In a full-scale filter, automatic cake thickness control would ensure that the filter automatically adjusted the pumping time to obtain this cake thickness.

The cake moisture target of less than 10% was not achieved on either duty. However, it is likely that the requirement to repulp the cake and filtrate after each test had a negative impact. It would be expected that the cake moisture obtained on a full-scale filter with fresh thickener underflow would be lower. In the tests performed, the lowest cake moistures obtained were 11.3% and 11.0% for the zinc and copper respectively.

For design purposes, a filtration rate of 380 or 510kg/m2h for Larox filters with 45 or 60mm chamber respectively, should be used for either duty.

<unquote>

5.2.2 Jord Filter Tests

The results of the testing conducted for Jord Engineers Pty Ltd was reported to Metplant Engineering Services Pty Ltd in the report Pressure Filtration Testwork on Benambra Copper and Zinc Concentrate Slurry.

The testwork found that the copper concentrate could be filtered to 12.2% moisture with a specific filtering rate of approximately 1170kg/m^2/h. The corresponding data for the zinc concentrate sample were 11.2%moisture at approximately 1450kg/m^2/h.

The moisture content achieved was higher than that reported by Larox in both cases. However, this result is probably within experimental error and, given the history of the samples, can be considered an equivalent result.

The specific filtering rates reported by Jord are significantly higher than those reported by Larox. There are a number of possibilities that could explain such a difference ranging from the history of the sample to grade of filter cloth to experimental technique. Jord report that the parameter is not used directly in filter selection and that filter area is generally well in excess of what would be indicated by direct calculation from this unit. Larox also state that the value reported is adjusted to reflect their filter operation and so the two values cannot be compared directly.

5.3 Regrinding Test Results

Samples of copper primary concentrate and zinc primary concentrate were prepared for testing for regrind mill specification. A range of samples was prepared over a period to resolve some unexpected results and to provide comparative data for different mill vendors.

5.3.1 Svedala Laboratory Tests

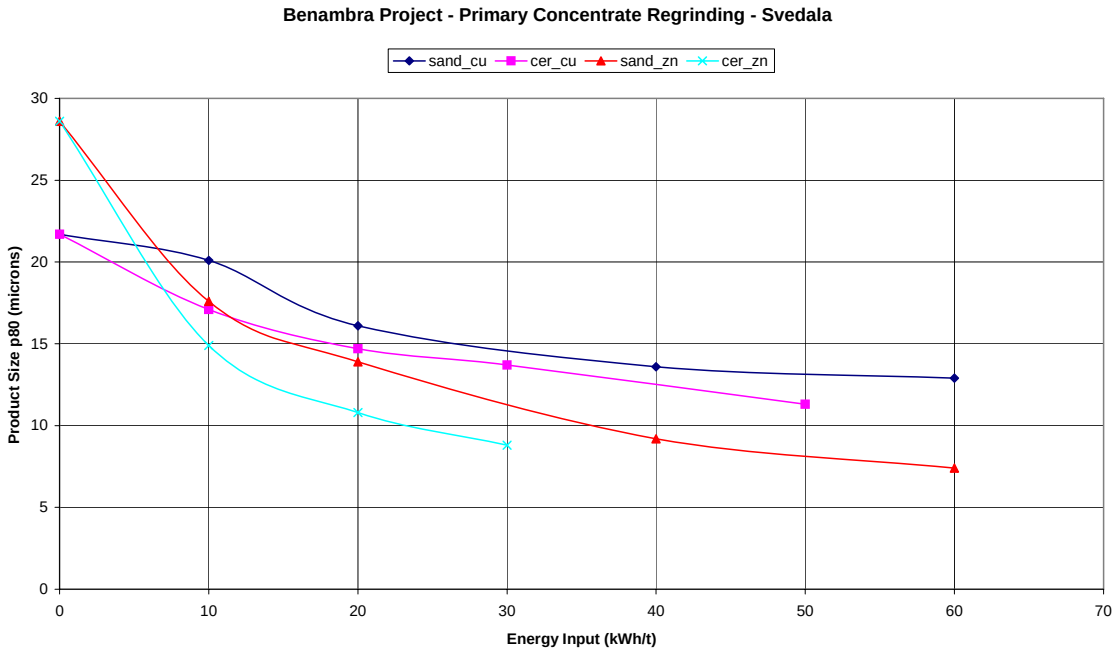
The initial samples were prepared from the Wilga and Currawong global composites and the concentrates from each composite were combined. The samples were filtered and the wet filter cake was despatched to the Svedala laboratory in England for testing. This was done at the request of Svedala on the basis that a greater range of grinding media types would be available for testing than was available in Australia.

After some difficulties, relating to possible contamination of the samples by fibres from the filter paper, results were obtained using both Colorado Coarse sand (10/20) and Carbolite (12/18) ceramic media. Both charge types were seasoned. The test results indicated a significant difference in the grinding response of copper concentrate compared to that of zinc concentrate. A significant energy reduction (of the order of 10kWh/t over a wide range of product sizes) could be attributed to the use of the more robust and spherical ceramic media.

The grind curves determined in the Svedala laboratory are plotted in the graph below. This shows the difference in the grinding characteristics of the

two concentrate samples and the improvement due to the use of ceramic media. A preliminary assessment of the economics of ultra-fine grinding, using ceramic media compared to Colorado sand, indicates that the savings in energy cost and lower consumption of media more than offsets the higher unit cost of the material. There is also a potential benefit in reduced capital cost if smaller machines can be used for a given duty. This benefit was not taken into account in the analysis.

It is possible that some of the 'abnormalities' (non-regular curve) in the grind curves determined might have been caused by the presence of residual filter paper fibres. It is also possible that sampling or other sources of error could be responsible.



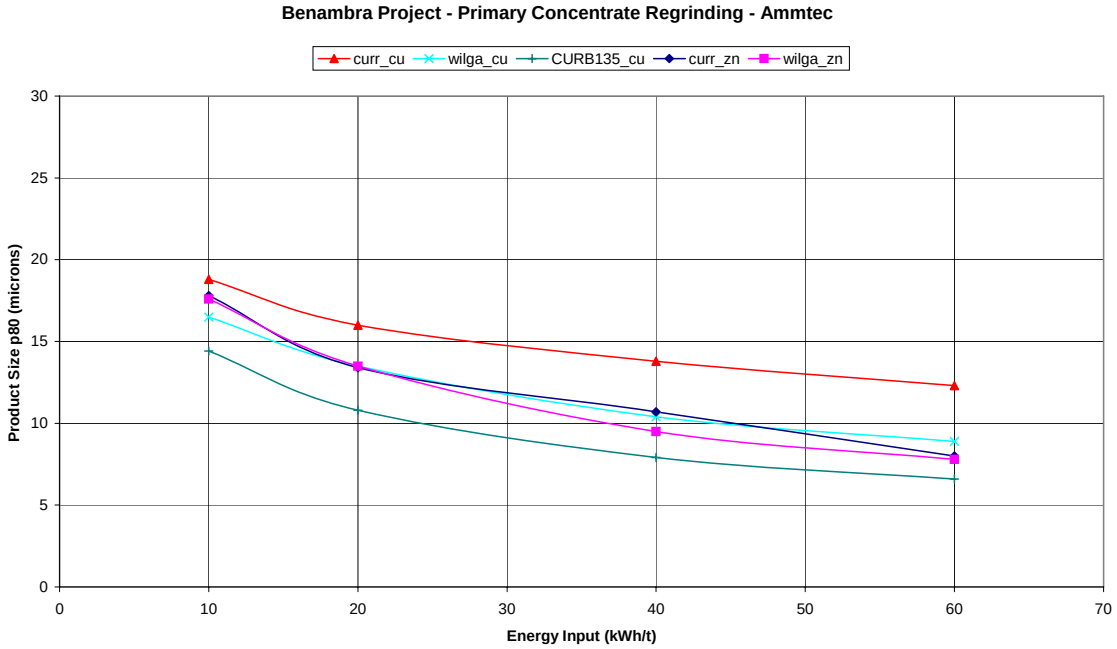
The results of these tests together with those performed at AMMTEC were fully reported in the Svedala Test Report No 01/100 – Laboratory Stirred Media Detritor Testwork.

5.3.2 AMMTEC Laboratory Tests

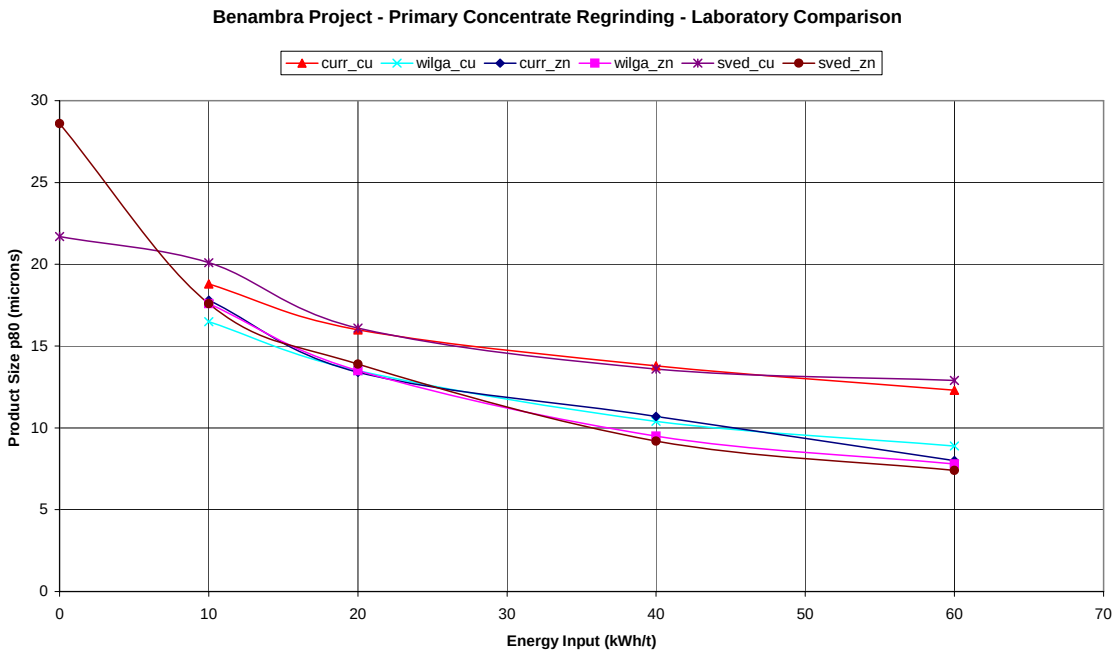
Because of the differences in grinding performance between copper and zinc concentrate indicated in the first test series, new samples of primary concentrate were produced from Wilga and Currawong global composites separately. These new samples were tested in the AMMTEC laboratories in Perth, WA. In addition, a sample of copper concentrate was produced from the composite CURB135 and tested. These tests could only be conducted using Colorado sand media.

These tests showed that the grind curve for Currawong copper concentrate was significantly more energy intensive than for the Wilga copper concentrate. They also showed that the grind curve for Wilga copper concentrate was almost identical to those for zinc concentrate from either composite. The

curve for the CURB135 concentrate was significantly more favourable than any determined from the global composite samples.



The tests also indicated that the grind curve could be reproduced between laboratories and that the high energy demand of the combined copper concentrate could be attributed to the presence of some component of the Currawong concentrate.



There was no comparable effect on the response of the zinc concentrate.

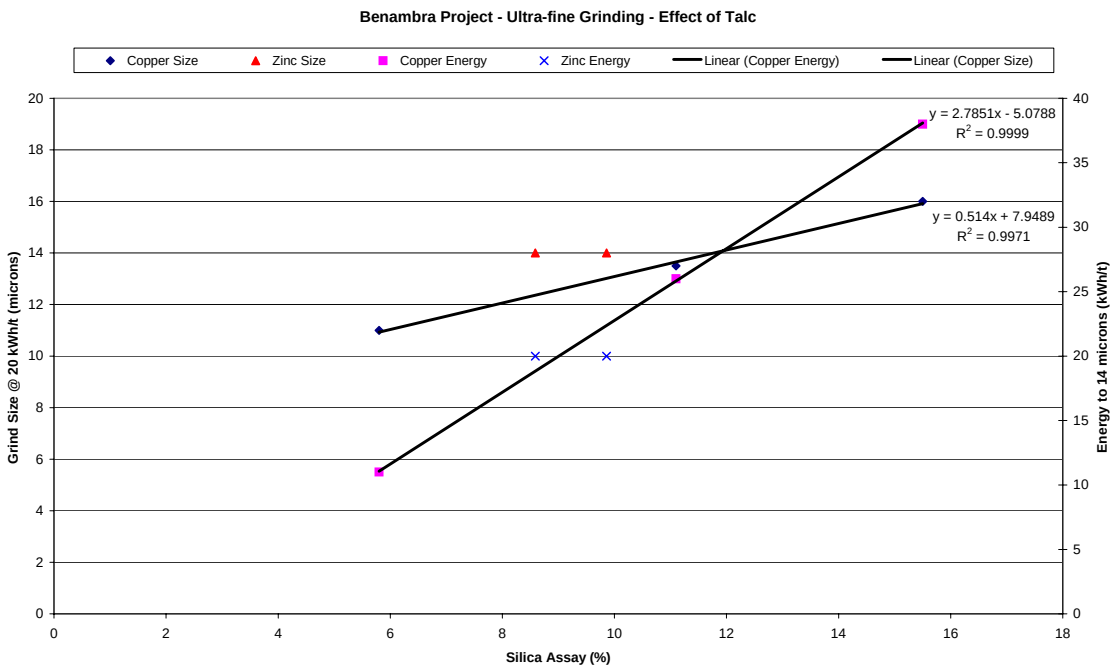
The only known significant difference between the Wilga global composite and the Currawong global composite was the presence of a higher content of talc in the prepared composite. Further samples of concentrate were produced from a Currawong sample that was assessed as being free from significant talc. The copper concentrate produced was tested at AMMTEC and this sample proved to be significantly freer milling than even the concentrate from the Wilga global composite.

Because of the possible effect of the presence of talc on the grinding performance of the concentrate produced, the samples despatched to AMMTEC were assayed for the following elements – copper, lead, zinc, iron, sulphur, silica and magnesia. The results of this analysis are tabulated below.

Sample	Cu	Pb	Zn	Fe	S	SiO2	MgO
CGC Copper	9.71	1.30	4.42	34.0	30.3	15.5	4.48
WGC Copper	8.41	0.96	5.96	37.1	36.1	11.1	1.56
CURB135 Copper	9.82	1.50	4.53	37.1	41.3	5.80	0.65
CGC Zinc	0.48	0.71	18.7	31.3	35.6	8.59	1.58
WGC Zinc	0.47	0.40	17.9	31.8	37.1	9.86	0.83

In order to assess the likely effect of the talc content of the concentrate on grinding performance, two parameters were extracted from the grind curves determined at Ammtec. These parameters were the grind size achieved from an energy input of 20kWh/t and the energy required to grind to 14μm (kWh/t). These parameters for copper concentrate were plotted against the silica assay of the sample. Although there were only three points, the correlation was highly linear. The corresponding data points determined for the zinc concentrate gave reasonable agreement with the relationship for copper concentrate. This result is highly suggestive that the ultra-fine grinding performance is affected by the non-sulphide mineral content of the concentrate.

Similar trends occur for the magnesia content of the samples although the relationship appears to be non-linear.



5.3.3 MIM Technology Laboratory Tests

In order to be able to obtain competitive bids for the supply of fine grinding mills, additional sample was prepared from the 'talc free' Currawong sample (CURB135) and these were despatched to MIM Process Technologies for testing. These tests were conducted using a sample of Carbolite 12/18 which is the ceramic medium that was used in the Svedala tests in England. The media charge was not seasoned.

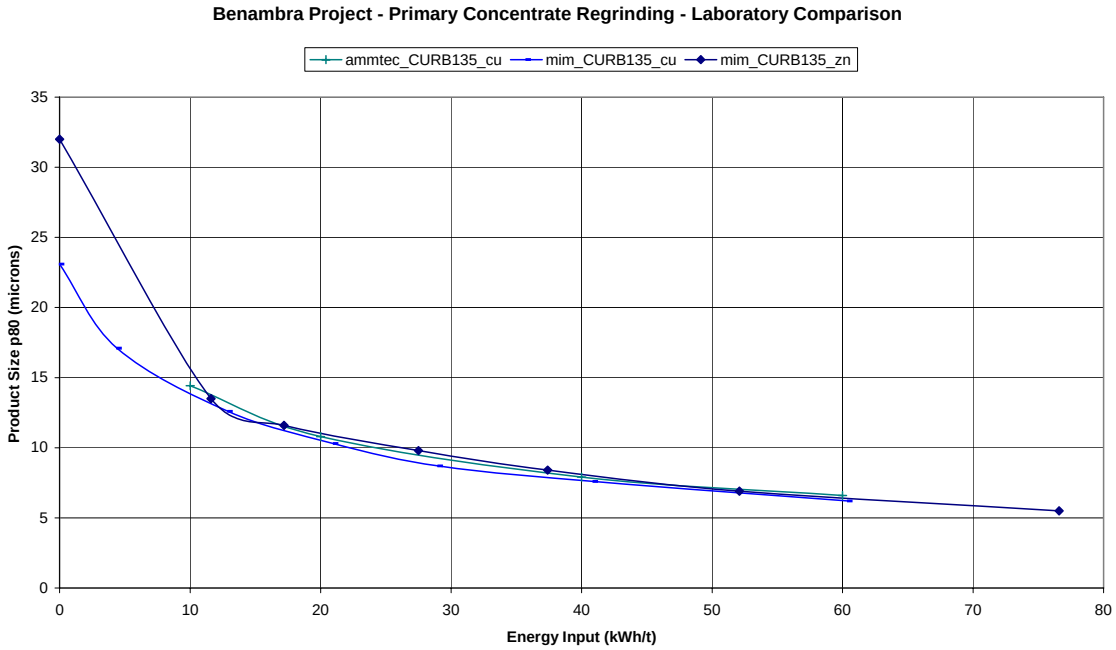
The results of this testing are reported in the MIM Process Technologies Filenote – IsaMill Tests with Benambra Copper and Zinc Rougher Concentrates. The MIM results confirmed the response determined by Ammtec on the copper concentrate and indicated that there was little difference in the grind curve between the copper and zinc concentrate after the initial effect of different feed size distributions.

The other interesting feature is the apparent lack of benefit due to the use of ceramic grinding media in the MIM tests. The AMMTEC testing was done with Colorado sand and the energy input was measured as energy input to the mill (using a torque measurement and rotor speed). The MIM Process Technologies testing was done with ceramic media and the energy input was measured as energy consumed (energy input to drive less no-load).

It is not clear that the difference in energy measurement technique is sufficient to offset the significant effect of using ceramic media that was evident in the Svedala laboratory tests. Svedala suggest that an allowance of 5% should be added to their laboratory readings to allow for production drive efficiencies. Svedala claim that their scale-up is exact. MIM suggest that Svedala readings can be up to 25% or more different to theirs for comparable duties.

They also claim that the MIM methodology of measuring motor power draw and correcting for no-load gives excellent scale up to commercial units.

If the Svedala energy measurements are understated, then the energy efficiency attributable to use of ceramic media could be offset and result in the apparent congruence of results. The only way to confirm the effect reported by the Svedala laboratory would be to conduct comparative tests using the different media to grind the same product.



5.4 Crushing and Grinding Test Results

A sample of ore from the Wilga ore body that was collected immediately prior to the closure of the mine in 1996 had been stored in a 200l drum in the core storage shed at Benambra. This sample was sent to Amdel where a standard suite of crushing and grinding characterisation tests was conducted. It was decided that full characterisation using the JKMRC drop tests and tumbling tests would not be done at this stage because the use of autogenous or semi-autogenous milling had been eliminated from consideration as a process option.

The parameters determined are tabulated below, and, for comparison, values of parameters that have been determined on other samples of ore using the standard methods are also included. From this table it can be seen that the major grinding indices are consistent over a number of samples. This indicates that the ore grinding performance should be well predicted from historical operating performance. Full details of the tests and results are included in the Amdel Report No N020CO01 – Comminution Tests – Benambra Project.

Index	Wilga ROM 2001		Wilga MS 1988		Wilga Adit MS 1988	Currawong 1989
		range		range		
Bond Crushing	10.4	3.1/17.9		2.4/23.4		
Bond Rod Mill	18.2		21.5		17.0	19.0
Bond Ball Mill	12.4/11.9		13.3		12.5	12.6
Bond Abrasion	0.22				0.16	
Unconfined Compressive Strength (UCS)						
Average	149	MPa				
Maximum	194	MPa				

5.5 Concentrate Comprehensive Analysis

In order to provide information to potential purchasers of concentrate, a number of samples of concentrate (including material produced from the locked cycle tests) were submitted for comprehensive elemental analysis. The data generated is summarised in the following table together with typical analyses extracted from documentation available from the Denehurst files.

	Currawong						Wilga					
Concentrate	**Copper**			**Zinc**			**Copper**			**Zinc**		
Test No /Source	**CGC07**	**LCT1**	**LCT3**	**CGC07**	**LCT1**	**LCT3**	**WGC07**	**LCT4**	**Dene-hurst**	**WGC07**	**LCT4**	**Dene-hurst**
Element/ Species												
Cu %	23.77	23.3	26.9	0.92	1.49	1.13	22.17	23.3	19.8	0.64	1.28	2.5
Pb %	4.26	4.31	3.42	0.71	0.75	0.6	2.07	2.31	3.4	0.38	0.45	1.3
Zn %	4.30	4.42	2.16	54.9	52.3	51.9	4.0	4.05	6.2	51.7	50.7	46.9
Fe %	29.9	25.9	26.3	8.1	8.43	8.7	23.9	30.1	28.6	10.1	10.1	12.1
Ag g/t	167	165	169	86	72	72	129	140	135	93	83	70
As ppm	<100	350	1100	900	900	500	<100	250	200	<100	100	200
SiO_2 %	0.60	5.73	1.81	0.6	1.49	1.1	0.5	1.42	4.8	0.64	1.33	1.4
MgO %	0.23	2.17	0.96	0.18	0.45	0.5	0.13	0.35	1.7	0.12	0.23	0.45
Cd ppm	150	130		1490	1650		120	125	200	1410	1350	1300
Co ppm	130	60		40	40		150	80	60	60	80	46
Bi ppm	400	250	340	100	100	150	320	320	300	60	97	100
Hg ppm	16	0.4	4.7	10	4.4	4.1	19	0.8	<1	14	5	3
F ppm	<200	500	<100	<200	600	<100	<200	300	200	<200	200	<100
Cl ppm	<200	<100		<200	200		<200	<100	300	<200	<100	300
Au g/t	1.00	1.29		0.71	0.70		1.4	1.13	<1.0	0.41	0.51	<1.0
Tot S %	38.45	33.00	33.0	36.53	33.40	33.7	41.41	37.7	34.1	37.57	34.9	34.3
Ni ppm		50			100			<50			100	
Sb ppm		<50			<50			50			50	
Se ppm		230			150			155			70	
Te ppm		5			3.3			0.3			0.2	
Mo ppm		<50			<50			<50			<50	
MnO ppm		100			200			<0.01			0.02	
Sn ppm		50			100			50			100	
V ppm		<20			30			<20			<20	
CaO %		0.21			0.48			0.09			0.44	
Na_2O %		0.12			0.10			0.12			0.13	
Al_2O_3 %		0.13			0.21			0.02			0.11	
TOTAL	101.51	99.29	94.55	101.94	99.10	97.63	94.18	99.46	98.60	101.15	99.67	98.95

These results indicate that the minor element content of the concentrate produced is generally in line with the values reported by Denehurst from actual production. The main areas for concern from a marketing point of view are the content of lead and zinc in copper concentrate. A secondary concern relates to the potential for higher levels of magnesia and silica. Otherwise, the minor element analyses fall within acceptable ranges with arsenic showing the most potential to attract penalties particularly from ore with high arsenic (arsenopyrite) content.

Also of concern are the differences in assay of what should be similar products, for instance the silica and magnesia assays of the Currawong concentrate where different laboratories gave significantly different results. Check assays were conducted at the same laboratory, where possible, and the reported results were confirmed.

Of further concern were some differences in the original test assay and the value reported through comprehensive analysis. The routine assays were done using AAS, which does have some shortcomings for the analysis of materials in particular grade ranges. The comprehensive analyses used the more reliable volumetric techniques where appropriate.

Test	Product	Original			Comprehensive		
		%Cu	%Pb	%Zn	%Cu	%Pb	%Zn
CGC07	Copper	24.1	4.2	4.3	23.8	4.3	4.3
	Zinc	0.9	0.7	53.3	0.9	0.7	54.9
LCT1	Copper	24.4	4.2	4.5	23.3	4.3	4.4
	Zinc	1.5	0.7	52.8	1.5	0.8	52.3
LCT3	Copper	27.3	4.3	2.5	26.9	3.4	2.2
	Zinc	1.4	0.7	51.5	1.1	0.6	51.9
WGC07	Copper	22.5	1.7	4.0	22.2	2.1	4.0
	Zinc	0.6	0.3	51.0	0.6	0.4	51.7
LCT4	Copper	25.1	2.3	3.8	23.3	2.3	4.1
	Zinc	1.3	0.5	50.7	1.3	0.5	50.7

This table, showing original and comprehensive assays for the main base metals shows that at least two of the copper concentrate samples returned assays over 1% below that originally reported and one zinc concentrate over 1% above that originally reported. There is no evidence to suggest that there were any systematic errors involved or that any of the reported test results are compromised in any way.

5.6 Concentrate Transportable Moisture Limit

The transportable moisture limit (TML) of a concentrate is a critical parameter, particularly if it is to be exported to off shore smelters. It was hoped that there would be sufficient concentrate from the thickening and filtering tests on locked cycle test products to allow meaningful results to be obtained. This was not the case and, even with a modified test technique, additional material was required for testing.

Following discussions with the test laboratory, it was determined that the sample mass could be increased by incorporation of product from open cycle tests that had been dried and stored at the flotation test laboratory. This had the effect of consuming the bulk of the concentrates that had been produced during the test programme. Even with this expedient, there was only just enough material available to conduct the tests.

The results determined for the two concentrates, on the sample provided are listed in the following table.

Concentrate Type	**Measured TML (%moisture)**
Copper Concentrate	13.37
Zinc Concentrate	14.21

These results are above the reported filter cake moisture content of 11-12% by a comfortable margin. This is in line with the observation from the filter tests that “the filter cakes were dry and evenly formed, and would discharge very well from the filter chamber” and suggests that there should be no problem with transportation of concentrate produced direct from the pressure filter.

ADDENDUM

Flotation Tailings Cyanidation Leach Testing

Three tests were carried out on flotation tailings samples from the following composites:

- Wilga global composite,
- Currawong global composite and
- Currawong composite BERD12B.

For the global composite samples, a number of test products were combined to produce 'bulk' quantities of sample against the requirement to carry out additional testing. For BERD12B, only one flotation test had been done so that this test was 'one off' and at half volume compared to the other two tests.

The conditions selected for all tests were as follows:

- Sample mass 1000g (500g for BERD12B)
- Pulp density 45%solids
- Ambient temperature
- Cyanide concentration 0.05%
- Slurry pH 10.5 adjusted with lime
- Oxygen injection throughout test
- Sampling intervals 2,4,8,24,48,72 hours

The results of the tests are reported in the attached Optimet test reports. The gold leach profiles are plotted in the associated graphs.

The results show that, at most, 20-30% of the gold can be extracted by cyanide leaching after 72h. They also show that a large proportion (30-50%) of the gold extracted goes into solution within the first two hours of the test period. They also show that there is a close parallel between the extraction of gold and copper. This suggests that the gold that is being extracted is that which is associated with chalcopyrite. The cyanide consumption, while not extraordinary, is quite high, probably as a reflection of the solubilisation of copper.

Analysis and testing carried out by Denehurst and Macquarie Resources showed the following for samples that reported significantly higher gold grades:

- A minor proportion of the gold is soluble in mercury (free gold)
- The majority of the gold is associated with pyrite (generally >90%)
- Extraction of gold in cyanide leaching was low (approximately 10%)

- Most extraction occurred within 2h
- Cyanide consumption (2.9kg/t) was similar to the current test series
- Lime consumption (1.9kg/t) was similar to the current test series
- Roasting of the sample increased the gold extraction but at the expense of greatly increased cyanide and lime consumption

The test results show that flotation tailings from the general run of ore from Benambra is unlikely to be a candidate for profitable recovery of gold by cyanidation, either directly or following some form of pre-treatment such as roasting or bacterial oxidation. Ore that might contain sufficient value to warrant further consideration for gold recovery is likely to be small in tonnage and difficult to isolate. In addition, some evidence suggests that ore containing significant gold is likely to be high in arsenic and/or bismuth which, in themselves, are likely to pose additional problems from an operational, marketing and environmental point of view.

Unless significant new information becomes available, it is not considered that it is worth pursuing cyanidation extraction of gold from the Benambra ore further.

PROJECT 00040

Agitation Cyanide Leach Testing

1. Sample Tested (4/9/2001)

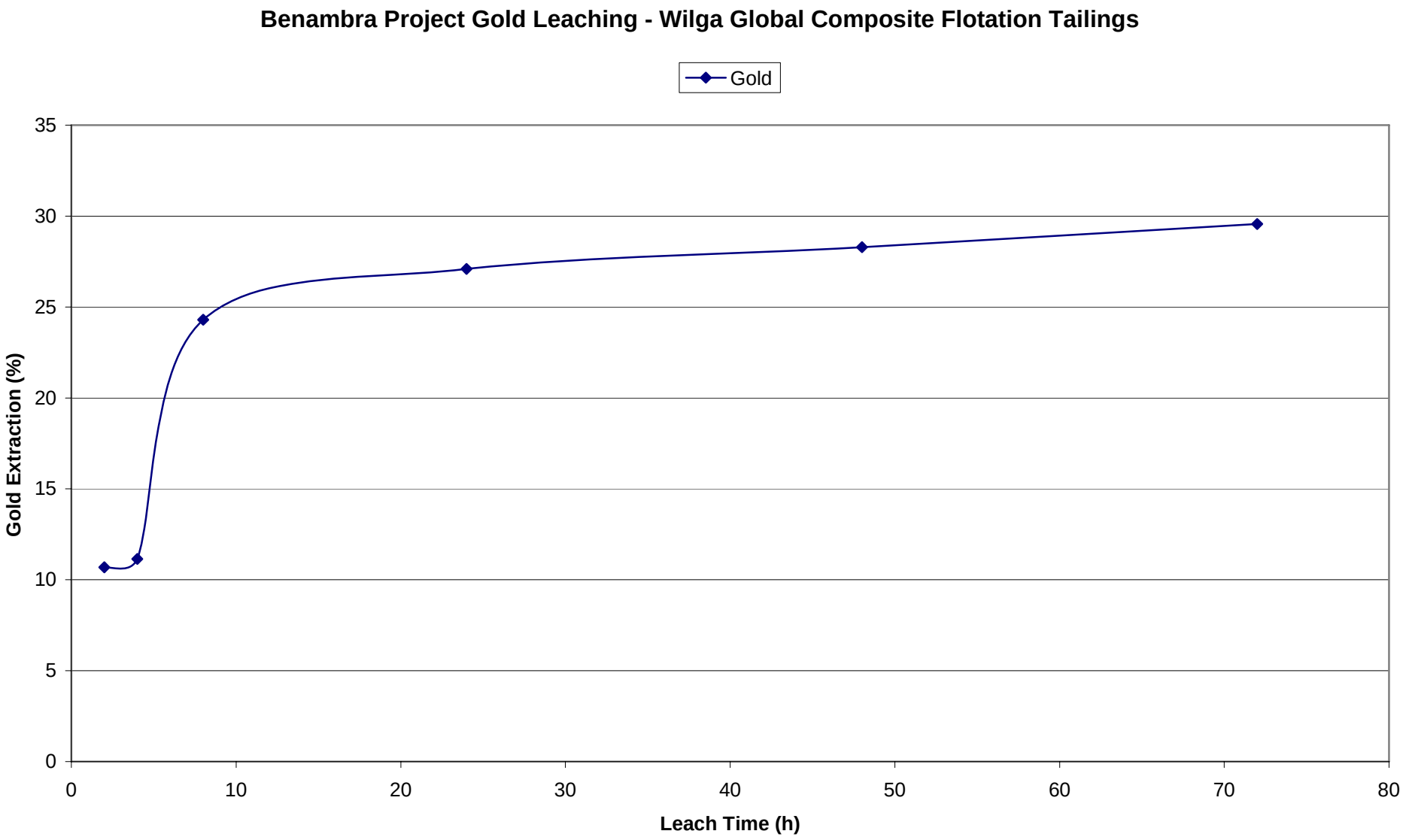
Leach Test 1 - Combined Zn Rougher Tail (Wilga Global Composite) 0.57 g/t Au, 14 g/t Ag, 0.19% Cu

2. Conditions Used

Test charge:	1000 g
Pulp density:	45% solids
Temperature:	Ambient ~19°C
Cyanide:	0.05% maintained at all sampling intervals.
pH:	10.5 regulated with lime and maintained at all sampling intervals.
Sampling:	2, 4, 8, 24, 48 and 72 h

3. Results Obtained

Pregnant Liquor								Residue			Extraction			Calculated			Reagent	Reagent	
Time	Vol.	pH	[NaCN]	DO	Assay			Assay			%			Head			Added	Consumption	
h	ml		%	mg/l	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	NaCN	kg/t	
					mg/l	mg/l	mg/l	g/t	g/t	%				g/t	g/t	%	g	NaCN	Lime
2	1232	10.20	0.008	8.2	0.05	0.66	169				10.68	0.00	11.25				0.61	0.51	1.42
4	1237	10.80	0.012	8.4	0.05	1.71	210				11.15	0.00	14.49				1.12	0.97	1.42
8	1230	10.10	0.011	9.2	0.11	2.85	260				24.30	0.00	18.26				1.58	1.44	1.42
24	1219	8.84	0.008	8.9	0.12	3.54	322				27.10	0.01	22.85				2.06	1.96	1.60
48	1228	8.58	0.010	9.4	0.12	4.00	376				28.29	0.01	27.43				2.57	2.45	1.90
72	1242	8.28	0.011	8.8	0.12	4.1	402	0.41	9	0.1	29.56	0.01	30.41	0.6	9	0.19	3.06	2.92	1.9


Benambra Project Gold Leaching - Wilga Global Composite Flotation Tailings
Gold
Gold Extraction (%)
35
30
25
20
15
10
5
0
0
10
20
30
40
50
60
70
80
Leach Time (h)

5.6.1.1.1 PROJECT 00040

Agitation Cyanide Leach Testing

1. Sample Tested (4/9/2001)

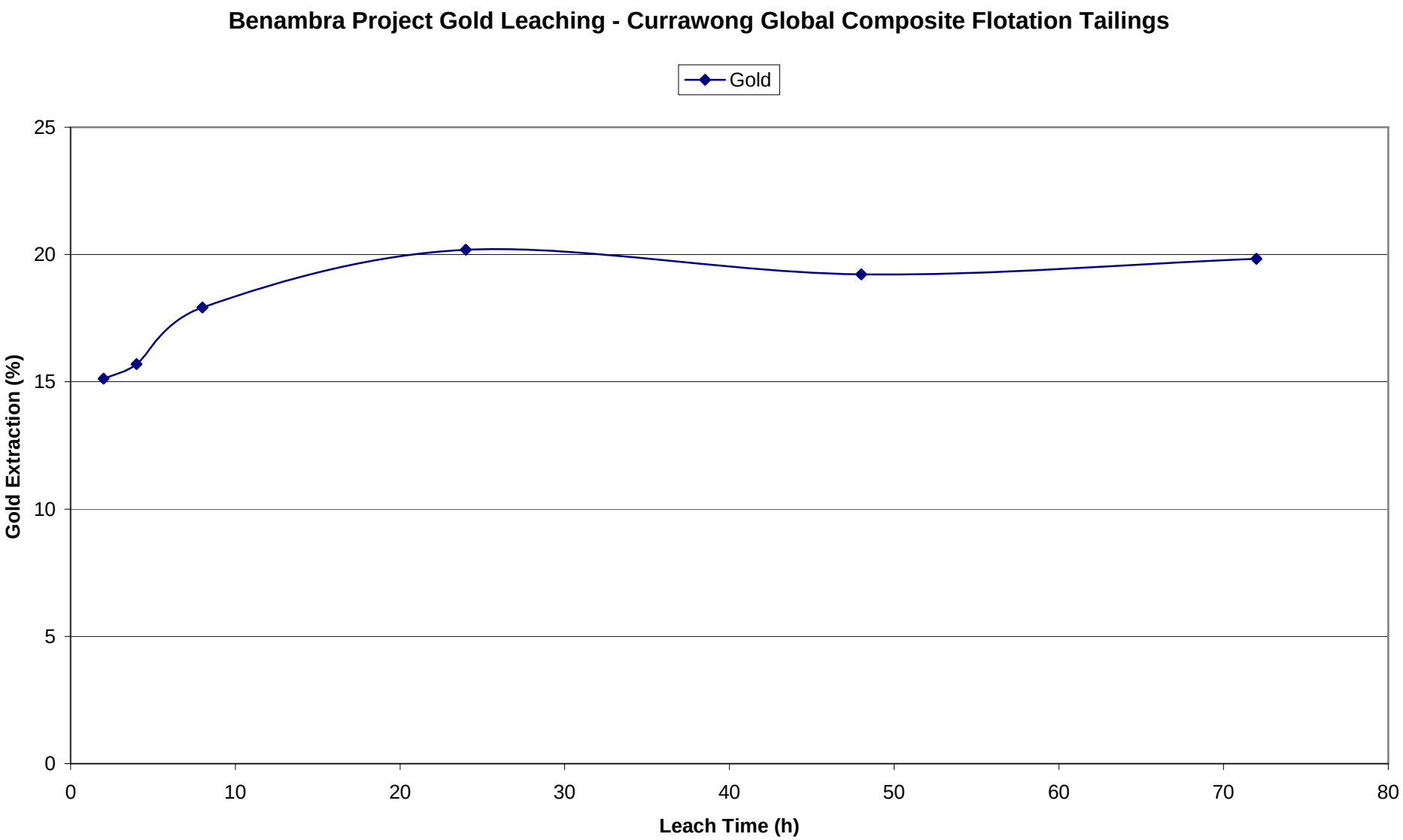
Leach Test 2 - Combined Zn Rougher Tail (Currawong Global Composite) 0.64 g/t Au, 16 g/t Ag, 0.31% Cu

2. Conditions Used

Test charge: 1000 g
Pulp density: 45% solids
Temperature: Ambient ~19°C
Cyanide: 0.05% maintained at all sampling intervals.
pH: 10.5 regulated with lime and maintained at all sampling intervals.
Sampling: 2, 4, 8, 24, 48 and 72 h

3. Results Obtained

	Pregnant Liquor							Residue			Extraction			Calculated			Reagent	Reagent	
Time	Vol.	pH	[NaCN]	DO	Assay			Assay			%			Head			Added	Consumption	
h	ml		%	mg/l	Au mg/l	Ag mg/l	Cu mg/l	Au g/t	Ag g/t	Cu %	Au	Ag	Cu	Au g/t	Ag g/t	Cu %	NaCN g	kg/t NaCN	kg/t Lime
2	1225	10.26	0.005	8.2	0.09	0.22	185				15.13	0.0	7.53				0.61	0.55	1.56
4	1221	10.85	0.009	8.1	0.09	1.23	269				15.70	0.0	11.22				1.16	1.05	1.56
8	1219	10.15	0.009	9.2	0.10	2.02	334				17.92	0.0	14.26				1.66	1.55	1.56
24	1214	9.07	0.007	8.8	0.11	2.80	410				20.19	0.0	17.81				2.16	2.08	1.56
48	1212	8.46	0.010	9.2	0.10	3.47	500				19.22	0.0	22.06				2.68	2.56	1.71
72	1208	8.17	0.01	8.6	0.1	3.71	538	0.59	11	0.23	19.83	0	24.32	0.74	11	0.304	3.17	3.05	2.03


Benambra Project Gold Leaching - Currawong Global Composite Flotation Tailings
Gold
Gold Extraction (%)
25
20
15
10
5
0
0
10
20
30
40
50
60
70
80
Leach Time (h)

5.6.1.1.2 PROJECT 00040

Agitation Cyanide Leach Testing

1. Sample Tested (4/9/2001)

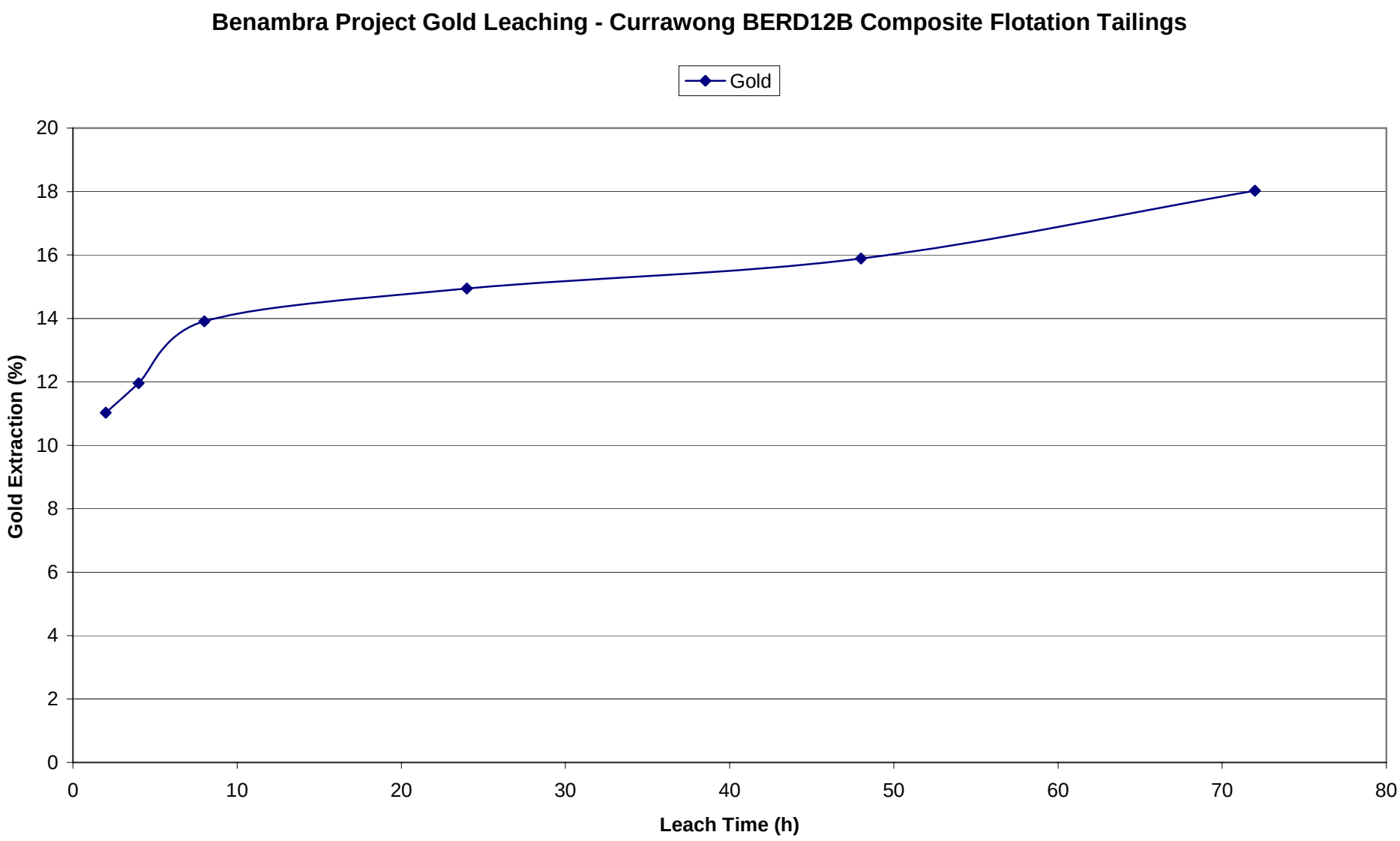
Leach Test 3 - Test 154 Zn Rougher Tail (BERD012B Drill Hole Composite) 1.00 g/t Au, 11 g/t Ag, 0.07% Cu

2. Conditions Used

Test charge: 500 g
Pulp density: 45% solids
Temperature: Ambient ~19°C
Cyanide: 0.05% maintained at all sampling intervals.
pH: 10.5 regulated with lime and maintained at all sampling intervals.
Sampling: 2, 4, 8, 24, 48 and 72 h

3. Results Obtained

			Pregnant Liquor					Residue			Extraction			Calculated			Reagent	Reagent	
Time	Vol.	pH	[NaCN]	DO	Assay			Assay			%			Head			Added	Consumption	
h	ml		%	mg/l	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	Au	Ag	Cu	NaCN	kg/t	
					mg/l	mg/l	mg/l	g/t	g/t	%				g/t	g/t	%	g	NaCN	Lime
2	622	10.35	0.022	11.6	0.10	1.66	80				11.03	0.0	14.90				0.31	0.35	1.60
4	625	10.54	0.040	11.9	0.10	1.73	83				11.96	0.0	16.72				0.48	0.46	1.60
8	623	10.32	0.039	12.3	0.11	1.79	86				13.91	0.0	18.47				0.54	0.59	1.60
24	627	10.18	0.021	10.6	0.11	1.82	96				14.94	0.0	21.71				0.61	0.96	1.60
48	628	10.10	0.026	11.1	0.11	1.77	96				15.89	0.0	23.10				0.79	1.25	1.66
72	632	10.04	0.028	11.4	0.12	1.89	101	0.93	8	0.05	18.03	0	25.56	1.13	8	0.067	0.94	1.53	1.74


Benambra Project Gold Leaching - Currawong BERD12B Composite Flotation Tailings
Gold
Gold Extraction (%)
Leach Time (h)
0
2
4
6
8
10
12
14
16
18
20
0
10
20
30
40
50
60
70
80

F A C S I M I L E T R A N S M I S S I O N

To:	Mr Andrew McDougal	**Date**:	23 Noovember 2001
Company:	Austminex NL	**From:**	Kwan Wong
City:	Melbourne, Vic	**Project No.:**	00040
Fax No.:	(03) 9670 8111	**No. of Pages:**	1

M E S S A G E

Andrew,

Wilga/Currawong Ore Testwork

Indium assay results obtained for concentrates are as follows:

Wilga Global Composite - Cu concentrate 11.5 ppm In, Zn concentrate 20.0 ppm In.

Currawong Global Composite - Cu concentrate 22.5 ppm In, Zn concentrate 40.0 ppm In.

The analytical method used for In assay was ICP Mass Spec with a detection limit of 0.05 ppm.

Regards,

K.Y. Wong
Managing Director/Consulting Metallurgist

Foot Note:
The testing of concentrate samples for Indium came about as a result of a request made by Alan Martin (Austminex Director) that an investigation be conducted to see if there was any possibility of Indium content providing extra revenue and project enhancement. Alan was aware that some base metals mines benefited from Indium credits. These mines had extractable levels and were associated with tied smelters.

- The above results were obtained from Wilga and Currawong concentrate samples.
- Indium is generally extracted in the zinc smelting process.
- The levels in Benambra concentrates are not high.
- Russell Colwell of Colken advises that there are no smelters in Australia set up and capable of extracting Indium. Indium extraction can only occur at a limited number of overseas smelters.
- Non tied smelters have a practice of not paying mines for any benefits the smelters receive from the extraction of Indium.

The conclusion is that Indium and any other minor elements (from previous analysis) will not enhance the Benambra Project economics.

A McDougall 29/11/01

Optimet LABORATORIES PTY LIMITED
A.C.N. 008 263 401 ABN: 67 075 206 340
Mineral Processing Consultants
Flotation Technology Specialists

25a Tenth Street
Bowden 5007
South Australia
Tel: 61-8-8340 3466
Fax: 61-8-8340 3404

FACSIMILE TRANSMISSION

To:	Mr Andrew McDougal	**Date:**	10 October 2001
Company:	Austminex NL	**From:**	Kwan Wong
City:	Melbourne, Vic	**Project No.:**	00040
Fax No.:	(03) 9670 8111	**No. of Pages:**	3

MESSAGE

Andrew,

Wilga/Currawong Ore Inventory

Attached are inventories of the composite ore samples prepared for testwork and also the remaining and unused drill core intercepts received for investigation.

All samples are currently kept in the cold store and we can continue to maintain the same storage arrangement or otherwise according to your further instructions as you wish.

Regards.

Kwan

K.Y. Wong
Managing Director/Consulting Metallurgist

PROJECT 00840: INVENTORY OF COMPOSITE ORE SAMPLES
9/10/01

Drill Hole Composite	Sample No.	Remaining Crushed Ore Pepared, kg	
		As -12.7 mm	As -2 mm
BERD002	173733 - 173768	~11	-
BERD003	173671 - 173681	~9	-
BERD004	173502 - 173514	-	17
BERD005	173564 - 173574	-	8
BERD006	173518 - 173562	~175	34
BERD007	173602 - 173612	~1	5
BERD008	173614 - 173625	-	-
BERD009A	173577 - 173586	-	10
BERD010	173640 - 173667	-	3
BERD011	173791 - 173807	~20	-
BERD012A	173699 - 173708	~17	-
BERD012B	173717 - 173726	-	6
CURB0135	B135/2 - B135/48	~80	9
Wilga Global	(BERD002+003+007+008)	-	-
Currawong Global	(BERD010+011+012A+012B)	-	-
Wilga 755 m	(Old Drill Core Intercept)	14	6

BE2 Tailings	(11 x 1 kg)
BE3 Tailings	(9 x 1 kg)

PROJECT 00040

INTERCEPTS - COLD STORAGE DETAILS
9 October 2001

Intercepts Totally Used			Composite
P173502	-	P173514	BERD004
P173518	-	P173562	BERD006
P173602	-	P173612	BERD007
P173614	-	P173625	BERD008
P173640	-	P173667	BERD010
P173671	-	P173681	BERD003
P173699	-	P173708	BERD012A
P173717	-	P173726	BERD012B
B135/2	-	B135/48	CURB0135

Intercepts ~Half Remaining			Composite
P173564	-	P173574	BERD005
P173577	-	P173586	BERD009A
P173733	-	P173768	BERD002

Intercepts ~Quarter Remaining			Composite
P173791	-	P173807	BERD011

Intercepts ~Unused			Composite
P173501			
P173515	-	P173517	
P173563			
P173575	-	P173576	
P173587	-	P173588	
P173589	-	P173601	
P173626	-	P173639	
P173668	-	P173669	

AUSTMINEX NL

BENAMBRA PROJECT

PROCESS PLANT ENGINEERING STUDY

SUMMARY REPORT

Prepared by:

Metplant Engineering Services Pty Ltd

A.C.N. 009 256 508

Date: 10 August 2001

Ref: 1344/AXD365RevB/046

CONTENTS*

*** NOTE: The summary document provides an overview of the process plant engineering. For details on Process Design, General Layouts, Equipment List, Flowsheets, Metallurgical and Mass Balances, Process Engineering Flowsheets, Electrical Single Line Drawing, Works Implementation Schedule, Detailed Breakdown of Direct Capital Cost, Detailed Breakdown of Operating Costs and Contract Crushing Proposals refer to Attachments A1 to A11 in the full Metplant Engineering Report.**

1. EXECUTIVE SUMMARY

1.1 General

Austminex NL (Austminex) has an option to acquire the Benambra Mining Leases and Operating Facilities in Northern Victoria.

Metplant Engineering Services Pty Ltd (Metplant) was engaged by Austminex to undertake a detailed engineering feasibility study to identify the level of capital expenditure required to upgrade and recommission the existing copper-zinc concentrator and to prepare operating cost estimates for a processing rate of 600,000 tonnes per year.

The processing route for the previous operations, which was based on sulphides flotation processing, has been retained for the upgraded concentrator except that concentrates regrinding capacity will be included to enable improved metal recoveries into higher grade concentrates.

Where practicable, existing concentrator equipment has been re-used, in conjunction with a value added engineering approach to ensure that the plant will be basic, yet robust and technically suited to the required purpose. Further, the scope has been developed to include works to ensure that the plant complies with all relevant governing regulations and environmental requirements.

1.2 Study Findings

The study has identified the following works as being required.

- The existing crushing plant is in poor condition and will be demolished and replaced with a new plant with an increased capacity.
- The existing ball mills will be retained as secondary mills with a new ball mill to be installed for primary grinding.
- New facilities will be installed for copper and zinc rougher flotation.
- New stirred media mills will be installed for the ultrafine grinding of the copper and zinc rougher flotation concentrates.
- The existing copper and zinc flotation facilities will be re-used and converted for the flotation of the ultrafine copper and zinc rougher concentrates.
- The existing zinc concentrate thickener will be re-used and converted for the partial dewatering of the final copper flotation concentrate.
- A new zinc concentrate thickener will be installed.
- The existing pressure filter will be refurbished for dewatering the final zinc flotation concentrate.
- A new pressure filter will be installed to dewater the final copper flotation concentrate.
- Flotation reagent mixing, storage and distribution facilities will be upgraded to meet the process requirements of the upgraded plant.

- Tailings disposal facilities will be upgraded to include a new tailings thickener, upgraded tailings pumps and a new tailings discharge line.
- A new, contractor supplied and operated power station will be installed to meet the electrical power demands of the upgraded plant.
- A centrally located electrical substation, switchroom and motor control centre will be installed.
- A new process control system will be installed.
- The existing power distribution system will be upgraded.
- The existing water and air services will be upgraded.

1.3 Works Capital Cost

The capital cost to implement the above works is estimated at AUD20,629,127. This cost has been prepared to an accuracy of plus or minus ten (10) percent.

1.4 Process Operating Cost

The estimated operating costs for the Benambra concentrator is AUD20.48 per tonne of ore processed. In addition to this cost is a provision for overhaul of major equipment and ball mill relines which is equal to AUD0.57 per tonne of ore over the projected ten year life of mine.

A one off cost of AUD0.17 per tonne should also be allocated to the first year of operation as a commissioning expense.

1.5 Works Implementation Time

The time required to implement the above works is estimated at fifty eight (58) calendar weeks inclusive of project preliminaries and plant commissioning.

1.6 Further Work

It is recommended that further work be undertaken as follows:

- Additional work to reach a decision on the viability of a contractor designed and installed crushing plant.
- Additional work to reach a decision on possible equipment rationalisation in the areas of ultrafine grinding, thickening and filtration.
- A comprehensive search of the secondhand equipment market when project financing has been approved.

2. INTRODUCTION

2.1 Objective

The overall objective of the study was to define the required works and to prepare capital and operating costs associated with the proposed upgrade and recommissioning of the existing copper-zinc concentrator at Benambra. The projected processing rate is 600,000 tonnes per year

2.2 Background

The Benambra concentrator was built and commissioned in 1992 by UC Engineers Pty Ltd for the Benambra Mine Joint Venture, a 50:50 partnership between Macquarie Resources Pty Ltd and Denehurst Limited.

The original plant was designed to process a high grade copper rich chloritic ore at a rate of 200,000 tonnes per year to produce a copper concentrate.

The plant was operated for approximately four years. During that period, progressive plant upgrades and modification works were completed, which allowed for an increase in capacity to 300,000 tonnes per year and the processing of copper-zinc ore to produce separate copper and zinc concentrates.

There were several operating problems during the previous period of operation. One significant problem was reported to be an insufficient grinding capacity available for acceptable mineral liberation in order to achieve satisfactory copper and zinc concentrate grades and recoveries. Other problems reported were the poor operability of the crushing plant and the inadequacy of the flotation reagent systems.

The Benambra Operations were shut down in 1996 due to low copper and zinc prices at that time. Austminex has an exclusive option to purchase the Benambra Copper-Zinc Project.

Prior to this Feasibility Study, Metplant has provided technical support to Austminex as follows:

- Plant Report and Valuation. A value of the plant and equipment at Benambra was established, and a cost estimate was prepared to restore the plant to an operating condition. (Refer to Metplant Report 1273/AXR002, March 2000.)
- Preliminary Study. Comparative capital and operating cost estimates were prepared for a range of crushing and grinding options for the Benambra plant, at throughput rates of 300,000 and 600,000 tonnes per year. (Refer to Metplant correspondence 1333/AXL025/01, March 2001.)
- Preliminary Project Costs. Preliminary project cost estimates were prepared for a 600,000 tonnes per year operation at Benambra. (Refer to Metplant Reports 1337/AXD072/046, May 2001 and 1337/AXR105RevA/046.)

A metallurgical testwork program has been completed by Austminex NL. The results of the testwork have been used to develop a process flowsheet that is appropriate for the optimum recovery of copper and zinc into separate concentrates.

The processing route includes the following stages:

- Crushing
- Grinding
- Copper flotation, including ultrafine grinding of the copper rougher concentrate.
- Zinc flotation, including ultrafine grinding of the zinc rougher concentrate.
- Thickening and filtration of the separate flotation concentrates.
- Thickening and discharge of tailings to a management facility.

2.3 Study Scope

2.3.1 Work Included

The extent of the works covered incorporated the following:

- Development of process flowsheets and mass balances to reflect the upgraded concentrator.
- Development of process design criteria defining the upgraded concentrator, based on data from Austminex, equipment vendors and Metplant's database.
- Development of a process description to reflect the upgraded concentrator.
- Development of preliminary equipment lists for the upgraded concentrator, showing new equipment and existing equipment to be re-used.
- Preparation of site layout drawings for the upgraded concentrator.
- Review of electrical loadings and the development of a power generation and distribution system to suit the upgraded plant.
- Development of the preliminary process control philosophy for the upgraded concentrator, describing how the new areas will be integrated into the existing plant.
- Development of the upgraded concentrator capital cost estimate to a plus or minus ten percent level of accuracy.
- A review of further opportunities to reduce the project capital cost through rationalisation and the use of second hand equipment.
- Development of a process operating cost estimate for the upgraded concentrator.
- Development of the preliminary works implementation schedule describing the duration of engineering design, procurement, construction and commissioning.

2.3.2 Work Excluded

The following work was excluded from the scope:

- All work associated with liaising with the relevant governing authorities.
- All work associated with the supply of raw water to the project site.
- All work associated with the supply of potable water to the project site.
- All work associated with re-establishing the tailings management facilities.
- All work associated with the site infrastructure required for the project inclusive of workshop, warehouse, administration office, personnel accommodation, communication, ablutions, sewage treatment and change-rooms.
- All work associated with laboratory and facilities required for the project.
- All work associated with site access roads required for the project.
- All work associated with mobile equipment required for the project inclusive of cranes, bobcat, forklift, front end loader(s) and emergency vehicles.
- All work associated with all emergency services required for the project, with the exception of fire water systems for the processing facilities.
- All work associated with defining to bankable status, the geological mining, metallurgical, environmental, regulatory, marketing and financial aspects of the project.
- All work associated with concentrates transport and processing.

2.3.3 Terminal Limits

The study terminal limits were as follows:

Process
- ROM ore feed. Delivery of ore into ROM bin by Austminex. ROM bin and associated facilities by Metplant.
- Tailings. Slurry discharge into existing tailings management facility by Metplant.
- Concentrate Storage - Discharge of concentrate into existing storage facilitates by Metplant.
- Concentrate Load-out. Loading of concentrate into single trailer type trucks by Austminex.
- Concentrates Weighing. Weighing facility recommissioning by Metplant.
- Reagents. Mixing, storage and distribution systems by Metplant.
- Bulk reagents and consumable storage on site by Metplant.

Water	-	Raw water. Outlet of discharge pipe into site storage facility.
	-	Potable water. Outlet of discharge pipe into site storage facility.
	-	Tailings return water. Inlet of return water pipeline at the tailings management facility.
	-	Environmental water run-off collection by Metplant.
	-	Fire water services by Metplant.
Power	-	Power supply and distribution by Metplant.
Air	-	Inlet into air generation facilities.

2.4 Authors

The study has been prepared by the following staff of Metplant.

- Marco Battaglia — Engineering Manager (Director)
- Ken Graham — Metallurgy Manager
- Udo Grahl — Chemical Engineer
- John Callaghan — Electrical Engineer

3. PROCESS AND ENGINEERING DESIGN

3.1 Introduction

The technical requirements to implement the upgrade works have been derived from the preliminary process and engineering design generally as described below.

This design of the process plant and infrastructure formed the basis from which the capital cost estimates were developed.

3.2 Design Basis Discussion

3.2.1 General

Preliminary engineering was undertaken to sufficiently define the extent of process, plant and infrastructure to support the new works.

The proposed plant design is not the optimum layout. It is a compromise based on limited available space and the emphasis on maximising equipment re-use and minimising equipment relocation.

Additional piping required for this concept will only add minor capital cost and does not compromise on process functionality.

Overall plant design is based on a minimum level of automation.

3.2.2 Process

The basis under which the preliminary process design was completed is described as follows:

- Process Route – the process route is based on:
  - simple, well-proven, robust technology that is appropriate to the treatment of Benambra ores,
  - the results of metallurgical testwork that was completed by Austminex.
- Process Design Criteria – equipment selection was made in accordance with the process design criteria as summarised in Attachment One. Where practicable, existing equipment has been re-used either for the same duty or an alternative duty as appropriate.
- Crushing and Grinding Circuits Selection – the previous concentrator operations at Benambra included conventional three stage crushing followed by two stages of ball milling.

  An analysis of alternative crushing and grinding options was completed by Metplant which included:
  - two stage crushing to High Pressure Grinding Rolls, followed by two ball mill grinding stages and
  - primary crushing to Semi-Autogenous Mill and Ball Mill.

The lowest capital cost option is three stage crushing followed by two stages of ball milling, given that the existing ball mills can be re-used in a secondary grinding duty in conjunction with a new primary ball mill.

- Crushing Circuit Capacity – the size and condition of the existing equipment in the crushing circuit is unsuitable for the higher duty of 600,000 tonnes per year. The size of the three new crushers and new product screen have been determined by equipment vendors using simulation modelling techniques.

- Grinding Circuit Capacity – the size of the new primary ball mill has been estimated by equipment vendors using simulation modelling techniques. The simulation modelling was based on a two-stage grinding circuit, with each grinding stage operating in closed circuit with cyclones. As mentioned above, the existing ball mills would be re-used in a secondary grinding duty. Selection and sizing of cyclones for the grinding circuit has been determined by equipment vendors.

- Flotation Route Selection – a sequential flotation route has been selected in which a copper rich flotation concentrate is collected, followed by collection of a zinc rich flotation concentrate. This flotation route is similar in principle to that used in the previous operation at Benambra.

  A significant difference in the flotation route for the proposed plant upgrade is the incorporation of ultrafine grinding to regrind the copper and zinc rougher flotation concentrates, followed by additional stages of flotation. The ultrafine grinding stages provide for separation of the sulphide minerals in the fine grained Benambra ores, providing improved copper and zinc concentrate grades and recoveries as compared to those achieved during the previous operation.

  The number of flotation stages, flotation time requirements and fineness of rougher flotation concentrate re-grind have been determined by metallurgical testwork completed by Austminex. Flotation cell capacity requirements for the concentrates was subsequently calculated by applying scale-up factors. Sizing of the ultrafine mills was determined from testwork conducted by the equipment vendors.

- Flotation Equipment Capacity – where possible, the existing banks of flotation cells have been re-used. In most cases the duty will be changed but without compromise to process efficiency. For the zinc ultrafine flotation stages some of the cells will be taken off line to ensure that acceptable flotation retention times are maintained.

- Flotation Concentrate Dewatering – the previous concentrate dewatering stages included thickening and pressure filtration to produce separate copper and zinc concentrate filter cakes in a form suitable for storage and transport by road and sea. The two concentrates were batched through a single filter.

  Testwork conducted by Austminex has determined the thickener and pressure filtration equipment sizes required for the concentrator upgrade, and takes into account the finer particle size and increased production rates of the two concentrates. Two filters have been nominated, with a filter dedicated to each concentrate.

- Flotation Reagent Selection – flotation reagent selection, dosage rates and dosing points have been determined from metallurgical testwork completed by Austminex. Flotation reagent mixing, storage and distribution systems have been developed to meet the required dosage rates. Storage tanks have been sized to provide where appropriate, a minimum of 24 hour capacity.

- Tailings Disposal – the residue from the concentrates will be dewatered in a thickener, to recover water for re-use in the process. The thickened residues will be pumped to a tailings management facility. The size of the thickener required for the duty has been determined from testwork conducted by Austminex.

  Residues in the previous concentrator operations were not dewatered before disposal. The reasons for including a tailings thickener in the concentrator upgrade are:

  – improved water management and reduced make-up water requirements which becomes more significant at the increased ore processing rate,

  – lower concentrator operating costs arising from reduced volumes of slurry requiring pumping to residue disposal and reduced reliance on the return water system for operational continuity.

3.2.3 Engineering

3.2.3.1 General Layout

Refer to Attachment Two for general layout drawings of the Benambra plant site and concentrator area.

The basis under which the preliminary design was completed is described as follows:

- Sufficient road access has been provided between the primary and secondary milling plants.

- Allowance has been made for mobile crane equipment to access the crushing area.

- Conveyor layout has incorporated the requirement for front-end loader access to emergency fine ore stockpiling and reclaim.

- MCC has been positioned in a central location, in an area of low dust exposure.

- Cranes and hoists will be installed to maintain major equipment in the crushing building, screen house and the new rougher flotation building. A central gantry type hoist will be installed in the reagent mixing area to allow bag handling for all powder type reagents.

- Overhead walkway access has been provided between the primary and secondary milling plant.

3.2.3.2 Crushing Circuit

The basis under which the preliminary design was completed is described as follows:

- The existing crushing circuit is poorly designed and of inadequate capacity. It will be replaced with a new 3-stage crushing plant, located in the same position.
- The new crushing circuit will comprise a jaw crusher (primary stage), two cone crushers (secondary and tertiary stage), a triple deck product screen, building structure and associated ancillary equipment.
- The product screen is located above the fine ore bin. This proven concept reduces the footprint area of the plant and eliminates the need for additional FOB and crusher feed conveyors. The screen has been sized generously to maintain good screening efficiency under high circulating load conditions.
- ROM bin and fine ore bin will also be replaced with larger ones to accommodate the increased plant capacity.
- A dust cover will be provided for the new ROM bin. Insertable type dust collectors will be installed in areas of potential dust generation such as crusher discharge conveyor, conveyor transfer station and fine ore bin.
- The existing fine ore bin will be used as an emergency fine ore feed bin for reclaimed stockpile material.
- A short sacrificial crusher discharge conveyor is provided to protect the main transfer conveyor.
- The primary crusher will be located downstream of the secondary and tertiary crushers. This will allow the secondary and tertiary crusher product to provide a bed of material for the primary crusher product, minimising wear on the conveyor belt and making more available any entrained tramp metal to the removal facilities.
- The handled ore is expected to contain high levels of tramp metal. Therefore, a tramp metal magnet and detector will be installed to protect the secondary and tertiary crushing units.
- Reclaim from the fine ore bin will be a passive slot feeder.

3.2.3.3 Grinding Circuit

The basis under which the preliminary design was completed is described as follows:

- A new primary ball mill will be added to the existing grinding circuit, including primary cyclones, discharge hopper and discharge pumps. The new mill will be located west of the existing mills. There is the possibility that No. 2 Secondary Mill could become redundant. This needs to be further investigated in the detail design stage.
- There is no confirmation at this stage that a previously identified second hand mill (the “Red, White and Blue” mill) will become available.

- Both existing ball mills will be re-arranged to form the secondary stage of the grinding circuit with both mills operating in parallel.

- Both grinding stages will be operated in closed circuit. The existing primary and secondary cyclones will be replaced with new clusters sized for the revised duty.

- The secondary stage will have a common cyclone bank with the underflow split between the two secondary mills. Discharge product from Secondary Mill No. 2 will be transferred to Secondary Mill No. 1 discharge hopper for cyclone feed. This proposed arrangement is not ideal but workable and has been adopted to minimise capital cost in this area.

- The overflow from the Secondary Cyclone will feed a new copper / zinc rougher flotation circuit.

3.2.3.4 Copper and Zinc Rougher Circuit

The basis under which the preliminary design was completed is described as follows:

- A new copper and zinc rougher circuit will be installed to produce a suitable feed for the ultrafine flotation circuit.

- The new circuit will be similar for the two products copper and zinc, utilising the area west of the grinding mills.

- The new rougher process will be accommodated in an enclosed building comprising conditioning tanks, ISA boxes, banks of flotation cells, classification cyclones, concentrate storage tanks and regrind milling stages.

- The products from the regrind mills will be pumped to the existing copper and zinc conditioning tanks which feed the ultrafine rougher and cleaner flotation cells.

- New tailings hoppers and pumps will be provided to transfer copper rougher tailings to the zinc rougher flotation and zinc rougher tailings to the tailings thickener respectively.

- Additional area sump pumps will be provided to recover spillage and return it to the relevant conditioning and concentrate storage tanks.

3.2.3.5 Ultrafine Copper Circuit

The basis under which the preliminary design was completed is described as follows:

- The ultrafine copper circuit will utilise all of the existing copper flotation equipment, including conditioning tank, flotation cells, blowers, pumps and hoppers.

- Current design is based on the installation of a new copper concentrate filter to allow independent filtration for each concentrate.

- The existing zinc concentrate tank and concentrate thickener will be converted for the copper duty.

- All other existing equipment of the copper flotation and will be re-furbished and re-utilised for the revised process configuration.

3.2.3.6 Ultrafine Zinc Circuit

The basis under which the preliminary design was completed is described as follows:

- The ultrafine zinc circuit will utilise all of the existing zinc flotation equipment, including conditioning tank, flotation cells, pumps, hoppers and pressure filter.
- The existing copper concentrate tank will be converted for the zinc duty.
- A new larger zinc concentrate thickener will be required, installed in the location of the current copper concentrate thickener. The existing concrete bund in this area needs to be extended to reflect the new thickener dimensions.
- A zinc cleaner tailings hopper will be added to the circuit to return cleaner tailings back to the rougher stage.

3.2.3.7 Tailings Circuit

The basis under which the preliminary design was completed is described as follows:

- A new tailings thickener and tailings tank will be added to the circuit to maximise process water recovery. The thickener will be located west of the current tailings hopper.
- Thickener underflow will gravity feed into the agitated tailings tank.
- The current tailings pumping station will be re-configured into two trains of two stage pumping systems (duty / standby mode). Two of the existing slurry pumps can be refurbished and re-utilised for the first stage of the new pumping systems. The second stage will require new high pressure rated slurry pumps, due to the higher pumping head for the high density slurry.
- The existing tailings to dam transfer line will need to be partially replaced by PE 100 pipe with higher pressure rating (PN25 and PN20). The existing PN 16 pipe section can be relocated and re-used at the tail end.
- Decant water can be returned per the existing gravity line, due to the RL difference between tailings dam and process plant (approximately 70 m). A priming pump will be required for initial line fill. The existing redundant diesel pump, currently located near the tailings dam could be utilised for that purpose.

3.2.3.8 Reagents Area

Available space for reagent mixing and storage facilities is restricted. The proposed layout provides a workable compromise in terms of accessibility and proximity to the actual points where the respective reagents are added to the process streams.

The basis under which the preliminary design was completed is described as follows:

- The existing lime mixing and storage facility will be re-utilised. The distribution ring main will be expanded to include additional take offs in the new rougher flotation area. The lime silo will be modified to improve lime reclaim.

- The existing flocculant dosing facility will be re-utilised. An additional flocculant dosing pump will be added to supply flocculant to the new tailings thickener.

- A new reagent mixing area will be installed, located north of the secondary grinding mill area. This area will cover the mixing and preparation of the reagents CMC, MBS, copper sulphate and Xanthate. Each reagent mixing facility will comprise a mixing tank with an insertable type dust collector and transfer pumps. Provision will be made for screening of Xanthate solution prior to transfer to header tanks. Positioning the new reagent mixing facility on the existing redundant power station slab provides a central location with easy access for reagent delivery. Each reagent area will have a separate spillage collection sump. A portable sump pump will be used to remove any spillage.

- The new reagent mixing area will be provided with a clad roof. A gantry crane covering the entire storage area will be installed for bag handling.

- There will be a new reagent storage area for MBS, CMC and copper sulphate, located south of the existing zinc flotation building. Each reagent storage facility will comprise tank and transfer/distribution pump(s) contained in a bunded concrete slab.

- Each reagent area will have a separate spillage collection sump. A portable sump pump will be used to remove any spillage.

- Header tanks for Xanthate, MBS and CMC will be located inside the flotation buildings, adjacent to the relevant bank of flotation cells.

- It appears to be more economical to install two separate reagent storage and transfer facilities for reagents not requiring mixing, (i.e. frother, zinc collector and copper collector) to service the rougher and ultrafine flotation areas. The facility servicing the new rougher flotation area will be located at the Western side of the new building. Ultrafine copper and zinc flotation areas will be supplied from a common facility, located at the northern side of the existing zinc flotation building. Both facilities will be installed within new bunded slabs including spillage collection sumps.

3.2.3.9 Services - Air

The basis under which the preliminary design was completed is described as follows:

- An additional high pressure air compressor with refrigerated dryer will be installed, located near the existing unit. The existing compressors will be used for standby/emergency supply.

- There will be no additional low pressure air blowers required. All new flotation cells are self-aspirating.

3.2.3.10 Services - Water

The basis under which the preliminary design was completed is described as follows:

- An additional process water pump will be installed, sized for the new duty. Existing process water pumps will be used as standby/emergency supply.
- The existing gland water supply pumps which supply seal water to the tailings pumps are not suitable to service the stage 2 tailings pumps. Two new high pressure pumps will be installed capable to supply the required water pressure and flows to the stage 2 tailings pumps. The existing pumps could be re-utilised to supply gland water to the stage 1 tailings pumps.
- The current fire water supply system needs to be upgraded to comply with current regulations and standards. A diesel fire pump set including electric jacking pump will be installed in the process plant area to supply fire water to hydrants and hose reels at the required flow rates and pressures.

3.2.4 Electrical

The basis under which the electrical design was completed is described as follows:

- Power demand – the power ratings from the mechanical equipment list, in conjunction with the expected surge and average maximum demand loading values were used to identify the capacity requirement of the new power source.
- Power source – the previous power station has been removed from site.

  A traditional diesel power station has been adopted for the power needs. The quantity of generating sets will be the typical “n+2” configuration which allows for sufficient on line generation capacity to meet the average maximum power demand with one unit down for maintenance and with another becoming faulted.

  To use the existing grid power line servicing the region would have needed a major line upgrade at excessive cost and the line would also need to be extended to site. Although the energy cost is attractive this option was excluded on the basis of capital cost.

  A third option of a steam turbine with biomass fuelled boiler and backup diesel system was competitive but was considered to represent a higher operational risk in comparison to proven diesel power for this type of application and was subsequently not adopted.
- Power station location – to ensure there is adequate room for the expanded process plant the power station will be located approximately 250m from the main plant area.
- Power distribution – two high voltage feeders will connect from the power station to the two separate high voltage switchboards in the plant electrical distribution centres. One feeder will go via the switchboard and distribution transformers to service the small power loads represented by the majority of 415 volt motors. The other feeder will service the larger power loads represented by the high voltage mill motors and the distribution transformer for the large 415 volt regrind mill motors.

- Mill starting – to control voltage and frequency excursions the new primary mill will use a wound rotor high voltage motor with a liquid resistance starter.

- Existing switchroom – the existing switchroom is congested and cannot accommodate any additional electrical panels. The electrical panels within the existing switchroom will be reinstated for all of the existing circuits that will be retained unmodified. The switchroom itself will be renovated to improve sealing, lighting and air conditioning and any surplus equipment will be removed to reduce the congestion.

- Existing circuits – a physical inspection revealed a need to replace a quantity of field based control and instrumentation equipment. Provision has been made to upgrade these devices to meet the new requirements.

- New switchroom - all existing circuits that need to be modified or relocated, and all new circuits, will originate from new electrical panels located in a new switchroom.

- Control system – the existing PLC in the existing switchroom is no longer supported so it will be replaced. This will be achieved by installing a new PLC in the new switchroom and using remote racks off this new PLC to replace the existing PLC racks.

- Instream Analysis System (ISA) – the existing 5 station ISA system will be refurbished and new computer equipment and new software will be supplied

- Plant control system – a new "Citect" style plant control and reporting system will be installed. This will interface to the plant via the new PLC system.

- Control room – the existing control room will be renovated and will accommodate the new plant operations computers for the "In Stream Analysis" (ISA) system and the "Citect" plant control and reporting system.

3.3 Process Description

3.3.1 Introduction

The process has been developed based on metallurgical testwork on the ore types to be processed and the application of standard, simple and robust equipment applied in the Australian base metal industry. Process flowsheets 1344-F-001 to F-010 have been developed and are contained in Attachment Four. The metallurgical balances and mass balances accompanying these flowsheets are contained in Attachment Five.

3.3.2 Ore Stockpiling and Reclamation

The ore transport contractor will deliver coarse ore produced by the mine to the run-of-mine (ROM) stockpile area on Waxlip Spur. The stockpile operation will be controlled to minimise the time that ore remains on the stockpile by utilising a first-in/first-out mode of operation.

The stockpile will also be used to achieve additional blending of ore of different quality to stabilise the tenor of the ore that is fed to the process. This will assist in stabilising the

operation of the flotation process and will result in more efficient separation of the valuable components of the ore.

Ore will be reclaimed from the stockpiles by a front-end loader (FEL) that will deliver the material to a run-of-mine (ROM) bin. Beneath this bin will be a variable speed feeder.

Water will be reticulated to the ROM stockpile and ROM bin to provide for dust suppression when required.

3.3.3 Crushing Circuit

The purpose of the crushing circuit is to size reduce the mined ore in preparation for the next stage of the process, grinding. A crushed ore storage facility (fine ore bin) is provided to allow for the crushing circuit to be stopped for maintenance without interrupting the operations of the concentration process which relies on steady, continuous operation to maintain high efficiency. There will also be provision for the generation of an emergency stockpile of crushed ore which would be available for delivery to the grinding circuit by FEL in the event of an extended or unplanned shutdown of the crushing circuit.

The crushing circuit will be designed to produce crushed ore in which 80 per cent of the particles are smaller than 8 mm. Three stages of crushing are proposed.

Ore delivered to the ROM bin will generally be smaller than 650 mm. The occasional rock larger that this size will be size reduced using a rock breaker mounted above the primary crusher.

There will be tramp metal in the ore from the underground mines. This will be removed using an electromagnet and metal detector combination, in order to prevent damage to the secondary and tertiary crushers.

3.3.4 Grinding Circuit

The fine crushed ore will be delivered at a controlled rate by a mill feed conveyor. Process water will be added to the primary mill feed chute and at other locations within the grinding circuit to produce the correct slurry density required for efficient grinding mill operation.

The grinding circuit will consist of two stages of ball mill grinding. Each stage of grinding will be in closed circuit with cyclone classifiers. The finished ground product will be produced from the second stage of cyclones. It will have a particle size in which 80 per cent of the material will have a particle size smaller than 28 microns.

3.3.5 Flotation

3.3.5.1 General

The method for recovering the valuable components from the ore will be the froth flotation process. This process involves the addition of selected chemicals to a slurry of the finely ground ore to render the surface of certain materials to a hydrophobic (water repellent) state. By agitating the slurry and introducing fine air bubbles in a flotation cell, the affected minerals can be floated to the surface attached to the air bubbles.

3.3.5.2 Rougher Flotation Circuit

The copper mineral (chalcopyrite) is collected and recovered in the first stage of rougher flotation. The rougher concentrate will be subjected to further grinding in a stirred media mill to improve the liberation of the valuable mineral particles from each other and from waste minerals. The finely ground concentrates will pass through additional stages of flotation as described in Section 3.3.5.3.

The residue from the first stage of rougher flotation will contain the majority of the zinc mineral (sphalerite) that was originally in the ore. A new addition of flotation chemicals will be made and the zinc minerals will be received into a rougher concentrate. This rougher concentrate will similarly be ground in a stirred media mill, with the finely ground material passing through to additional stages of flotation as described in Section 3.3.5.4.

3.3.5.3 Ultrafine Copper Circuit

The finely ground copper concentrate will pass through two further stages of flotation to reject unwanted material, resulting in an upgrade of the product to final concentrate grade.

The final concentrate will be prepared for dispatch to smelters by removing as much water as practicable. This will be achieved by passing the slurry into a continuous settling tank (thickener). A settling agent (flocculant) will be added to the slurry to accelerate the settling of the solids. A concentrated slurry will be drawn from the bottom of the thickener while clean water will overflow the top. The thickened slurry will be transferred to an agitated storage tank and the clean water will be re-used for ore processing.

The concentrate solids will be further dewatered by filtering. In this process, liquid slurry is forced under pressure into filter chambers that have a porous membrane. Water is expelled through the cloth while the solid particles are retained within the chamber. When the chambers are filled with solids, compressed air is used to remove further amounts of water. The water expelled from the filter will be returned to the thickener for recovery of any contained solids.

When the compressed air blow is complete, the filter cake is discharged through chutes to a concrete bunded area beneath the filter, then stockpiled in a covered storage area by front-end loader.

The filtered concentrate will be reclaimed from the stockpile by front-end loader and will be loaded into trucks for dispatch. A weighbridge is located in the concentrate loading area for weighing trucks during loading operations.

3.3.5.4 Ultrafine Zinc Circuit

The finely ground zinc concentrate will undergo similar processes of flotation, thickening, filtration and storage as described for copper concentrate in Section 3.3.5.3.

3.3.6 Tailings Disposal

The residue from the zinc flotation process will consist of minerals essentially barren of copper and zinc. The tailings stream will pass through a thickening tank where a significant proportion of the contained water will be separated for re-use in the process plant. The thickened tailings slurry will be pumped to the tailings management facility. The tailings management facility is not within the scope of this study and therefore has not been addressed.

The existing decant water return line at the tailings management facility will be re-used to recover water for return to the concentrator.

3.3.7 Reagents

The process design is based on the following reagents (or their equivalent) being used in the process. Refer to Table 1. A summary table showing the dangerous goods classification of these reagents as well as handling and storage measures has been included in Attachment One.

Table 1 – Reagents used in the Process

Reagent	Duty	Copper Flotation	Zinc Flotation	Thickening
Lime	pH adjustment		√	
MIBC	frother	√	√	
Xanthate	sulphide mineral collector		√	
Copper Sulphate	activator		√	
Proprietary Flocculant	settling agent			√
CMC (sodium carboxy methyl cellulose)	depressant for talc	√		
MBS (sodium metabisulphite)	depressant for pyrite and sphalerite	√		
Proprietary Copper Collector	sulphide mineral collector	√		
Proprietary Zinc Collector	sulphide mineral collector		√	

3.4 Plant Description

3.4.1 Introduction

Refer to Attachment Three for an equipment list of the upgraded plant.

The new plant proposed to be installed as a part of the works is separated into nine major areas. These are:

- Crushing Circuit – comprising jaw crusher, cone crushers, vibrating screen, apron and vibrating feeders, conveyors and associated ancillary equipment located in the new Crushing Area.
- Grinding Circuit – comprising primary and secondary ball mills, hydrocyclone clusters, slurry pumps and associated piping and ancillary equipment located within the Grinding Area.

- Copper and Zinc Rougher Circuits – comprising conditioning tanks, flotation cells, slurry pumps, hydrocyclones, regrind mills for both copper and zinc circuits including associated ancillary equipment located within the Roughing Area.

- Ultrafine Copper Circuit – comprising conditioning tank, flotation cells, slurry pumps, in stream analysis (ISA) station, pressure filter and associated ancillary equipment located within the Ultrafine Copper Area.

- Ultrafine Zinc Circuit – comprising conditioning tank, flotation cells, slurry pumps, ISA station, thickener, pressure filter and associated ancillary equipment located within the Ultrafine Zinc Area.

- Tailings Circuit – comprising thickener, storage tank with agitator, slurry pumping system, ISA station and associated ancillary equipment installed within the Tailings Disposal Area.

- Reagents – comprising mixing, storage, distribution and dosing of Lime, Frother, Xanthate, Copper Sulphate, Flocculant, CMC, MBS, Copper Collector and Zinc Collector. Includes all associated ancillary equipment installed within the Reagents Area.

- Services – Water – comprising tankage, piping and pumping of Potable Water, Process Water, Raw Water, Fire Water, Gland Water and Site and Plant Run-off Water. Includes all associated ancillary equipment located within the Water Services Area.

- Services – Air – comprising air compressors, dryers and receivers and all associated ancillary equipment located within the Air Services Area.

3.4.2 Crushing Circuit

The new crushing circuit comprises the following:

- Demolition of existing equipment and structures.

- Supply and installation of one (1) new apron type Primary Crusher Feeder, complete with manganese pans and hydraulic drive.

- Fabrication and installation of one (1) new ROM bin to suit the Primary Crusher Feeder.

- Supply and installation of one (1) new single toggle jaw type Primary Crusher complete with drive coupling, drive-guard, drive motor and drive motor baseplate.

- Fabrication and installation of one (1) new Primary Crusher Feed Chute complete with liner plates.

- Fabrication and installation of one (1) new Primary Crusher Discharge Chute complete with liner plates and primary crusher support steelwork.

- Supply and installation of one (1) new hydraulic pedestal mounted rockbreaker, complete with drive system.

- Supply and installation of one (1) new Crusher Discharge Conveyor complete with drive.

- Fabrication and installation of one (1) new Crusher Conveyor Head Chute complete with liner plates.
- Supply and installation of one (1) new belt type electromagnet Tramp Magnet complete with mounting frame and manual trolley hoist to suit monorail.
- Fabrication and installation of one (1) new Transfer Conveyor complete with drive.
- Fabrication and installation of one (1) new Transfer Conveyor Head Chute complete with liner plates.
- Supply and installation of one (1) new Tramp Metal Detector.
- Supply and installation of one (1) new Product Screen Feed Conveyor complete with drive.
- Fabrication and installation of one (1) new Product Screen Feed Head Chute complete with liner plates.
- Supply and installation of one (1) inclined circular motion triple-deck vibrating screen as Product Screen, complete with dust cover and drive arrangement.
- Fabrication and installation of one (1) new Product Screen oversize chute complete with inspection door and liner plates.
- Fabrication and installation of one (1) new Secondary Crusher Feed Bin complete with plug welded liner plates.
- Supply and installation of one (1) new vibrating pan type Secondary Crusher Feeder.
- Fabrication and installation of one (1) new Secondary Crusher Feed Chute complete with wear liner plates.
- Supply and installation of one (1) new cone type Secondary Crusher complete with feed canister, lubrication system, drive motor, drive coupling, drive guard and motor baseplate.
- Fabrication and installation of one (1) new Secondary Crusher Discharge Chute complete with wear liner plates.
- Fabrication and installation of one (1) new Tertiary Crusher Feed Bin complete with plug welded liner plates.
- Supply and installation of one (1) new vibrating pan type Tertiary Crusher Feeder.
- Fabrication and installation of one (1) new Tertiary Crusher Feed Chute complete with wear liner plates.
- Supply and installation of one (1) new cone type Tertiary Crusher complete with feed canister, lubrication system, drive motor, drive coupling, drive guard and motor baseplate.

- Fabrication and installation of one (1) new Tertiary Crusher Discharge Chute complete with wear liner plates.
- Fabrication and installation of one (1) new self-supporting cylindrical funnel Fine Ore Bin complete with overflow facility, product screen underflow chute and reclaim funnel.
- Fabrication and installation of three (3) new mass flow type passive Fine Ore Bin Reclaim Feeders complete with adjustable outlet spouts and internal bisalloy liner plates.
- Supply and installation of one (1) new monorail wire rope type Crushing Area Hoist complete with motorised lift, motorised travel and roving pendant.
- Supply and installation of one (1) new davit type Product Screen Area Hoist complete with hoist, runway trolley, catenary system, pendant and pedestal structure.
- Supply and installation of one (1) new insertable type Crushing Area Dust Collector complete with reverse pulse cleaning system and exhaust fan.
- Supply and installation of one (1) new insertable type Transfer Station Dust Collector complete with reverse pulse cleaning system and exhaust fan.
- Supply and installation of one (1) new insertable type Fine Ore Bin Dust Collector complete with reverse pulse cleaning system and exhaust fan.
- Supply and installation of one (1) new Crushing Area Safety Shower.
- Installation of new concrete bunded areas for new facilities to contain and isolate spillage.
- Installation of concrete foundations for new structures and equipment.
- Fabrication and installation of new beams, grating, handrailing, stairways, purlins, sheeting and all other structural members as required for reliable equipment performance, personnel access and site safety regulations.
- Installation of new piping systems for operation of new plant.

3.4.3 Grinding

The new grinding circuit comprises the following:

- Demolition of redundant cyclones. Extension of existing structure to accommodate the secondary cyclone clusters.
- Supply and installation of one (1) new Primary Mill Feed Conveyor complete with drive, pulleys, idlers, belt carcass and screw take-up.
- Removal and re-installation of one (1) Primary Mill Weigh Scale.
- Fabrication and installation of one (1) Primary Mill Feed Conveyor Head Chute complete with inspection door and liner plates.

- Fabrication and installation of one (1) new Primary Mill Charge Kibble Hopper.
- Fabrication and installation of one (1) new Primary Mill Feed Chute complete with inspection door and liner plates.
- Supply and installation of one (1) new overflow ball type Primary Mill complete with feed spout, liners, trunnion bearings, discharge trommel, open gear drive, main drive gearbox, couplings and main drive motor. Includes Primary Mill Inching Drive consisting of reducer and drive motor. Includes automatic grease spray type Primary Mill Open Gear Lubrication System. Includes automatic grease Primary Mill Trunnion Bearing Lubrication System.
- Fabrication and installation of one (1) new Primary Mill Discharge Trommel Underflow Chute complete with drainage panel and liner plates.
- Fabrication and installation of one (1) new conical shaped Primary Mill Discharge Trommel Overflow Chute complete with drainage panel and liner plates.
- Fabrication and installation of one (1) new conical shaped Primary Mill Discharge Hopper complete with overflow, drain and rubber lining.
- Refurbishment and re-installation of two (2) existing heavy duty centrifugal Primary Mill Discharge Pumps complete with liners, impeller, coupling and guard, drive motor and baseplate. The pumps will be relocated from the Tailings Area.
- Supply and installation of one (1) new cluster of two (2) Primary Cyclones complete with isolation valves, adjustable spigots, feed distributor, overflow hopper and underflow hopper.
- Refurbishment of one (1) existing Secondary Mill No. 1 Feed Chute, complete with inspection door and rubber liners.
- Refurbishment of one (1) existing Secondary Mill No. 1. Includes inching drive, open gear lube system and trunnion bearing lube system.
- Refurbishment of one (1) existing Secondary Mill No. 1 Discharge Trommel.
- Refurbishment of one (1) existing conical Secondary Mill No. 1 Discharge Hopper complete with overflow drain and rubber lining.
- Removal, modification and re-installation of two (2) existing heavy duty centrifugal Secondary Mill No. 1 Discharge Pumps complete with rubber liners, impeller, coupling with guard, drive motor and baseplate.
- Supply and installation of one (1) new cluster of six (6) Secondary Cyclones complete with isolation valves, manually adjustable spigots, feed distribution, overflow hopper and underflow hopper.
- Refurbishment of one (1) existing Secondary Mill Charge Kibble Hopper.
- Refurbishment of one (1) existing Secondary Mill No. 2 Feed Chute complete with inspection door and rubber liners.

- Refurbishment of one (1) existing Secondary Mill No. 2 includes main drive gearbox and motor, lubrication systems, inching drive, couplings, feed spout trunnion bearings and liners.
- Refurbishment of one (1) existing Secondary Mill No. 2 Discharge Trommel/Screen.
- Installation of one (1) new Secondary Mill No. 2 Discharge Underflow Chute complete with drainage panel and bisalloy liner plates.
- Installation of one (1) new Secondary Mill No. 2 Discharge Trommel Oversize Chute complete with drainage panel and bisalloy liner plates.
- Refurbishment of one (1) existing Secondary Mill No. 2 Discharge Hopper complete with overflow, drain and rubber lining.
- Refurbishment of two (2) existing heavy duty, centrifugal Secondary Mill No. 2 Discharge Pumps complete with rubber liners, impeller, coupling with guard, drive motor and baseplate.
- Removal, relocation, refurbishment, modification and re-installation of one (1) existing Fine Ore Bin as the Emergency Feed Hopper complete with static grizzly and plug welded liner plates.
- Removal, relocation, refurbishment, modification and re-installation of one (1) existing belt type Emergency Feeder complete with carcass, head pulley, tail pulley, rollers and drive.
- Removal, relocation, refurbishment, modification and re-installation of one (1) existing Emergency Feeder Head Chute complete with inspection door and liner plates.
- Supply and installation of one (1) new heavy duty vertical spindle centrifugal Primary Mill Area Sump Pump complete with drive and abrasion resistant liners.
- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Secondary Mill Area Sump Pump complete with drive and abrasion resistant liners.
- Supply and installation of one (1) new Primary Mill Area Safety Shower complete with shower and eyewash station.
- Supply and installation of one (1) new Secondary Mill Area Safety Shower complete with shower and eyewash station.

3.4.4 Copper and Zinc Rougher Circuits

The new copper and zinc rougher circuit comprises the following:

- Supply and installation of one (1) new open-top cylindrical Copper Rougher Conditioning Tank complete with overflow, drain, flat base and baffles.
- Supply and installation of one (1) new single open impeller type Copper Rougher Conditioning Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.

- Supply and installation of one (1) new bank of five (5) self-aspirated conventional Copper Rougher Flotation Cells complete with feed box, discharge box, concentrate launders, diversion boards, launder spray system, agitator, drive mechanism, drive motor, drive support, air header and diverter plates.
- Fabrication and installation of one (1) new Copper Rougher Concentrate Hopper complete with overflow, drain and rubber lining.
- Supply and installation of two (2) new heavy duty centrifugal Copper Rougher Concentrate Pumps complete with impeller, split casing, casing liners, mechanical seal direct coupling with guard and reverse overhead mounted drive motor.
- Supply and installation of one (1) new cluster of five (5) Copper Rougher Concentrate Cyclones complete with isolation valves manually adjustable spigots, feed distributor, overflow hopper and underflow hopper.
- Supply and installation of one (1) new open top cylindrical Copper Rougher Concentrate Storage Tank complete with overflow, drain, flat base and baffles.
- Supply and installation of one (1) new single open impeller Copper Rougher Concentrate Storage Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Supply and installation of two (2) new heavy duty centrifugal Copper Regrind Mills Feed Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Fabrication and installation of one (1) new compartmentalised Copper Regrind Mill Splitter Box complete with lid and rubber lining.
- Supply and installation of two (2) new stirred media detritor type Copper Regrind Mills complete with main shaft, agitator, liners, reducer, main drive motor and automatic lubrication system.
- Fabrication and installation of one (1) new conical shaped Ultrafine Copper Rougher Feed Pump Hopper complete with overflow, drain and rubber lining.
- Supply and installation of two (2) new heavy duty centrifugal Ultrafine Copper Rougher Feed Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Fabrication and installation of one (1) new conical shaped Copper Rougher Tailings Hopper complete with overflow, drain and rubber lining.
- Supply and installation of two (2) new heavy duty centrifugal Copper Rougher Tailings Pump complete with impeller, split casing, casing liners, mechanical seal direct coupling with guard and reverse overhead mounted drive motor.

- Supply and installation of one (1) new open top cylindrical Zinc Rougher Conditioning Tank complete with overflow, drain, flat-base and baffles.
- Supply and installation of one (1) new single open impeller type Zinc Rougher Conditioning Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Supply and installation of one (1) new bank of six (6) self aspirated Zinc Rougher Flotation Cells complete with feed box, discharge box, concentrate launders, diversion boards, launder spray system, agitator, drive mechanism, drive motor, drive support, air header and diverter plates.
- Fabrication and installation of one (1) new conical Zinc Rougher Concentrate Hopper complete with overflow, drain and rubber lining.
- Supply and installation of two (2) new heavy duty centrifugal Zinc Rougher Concentrate Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Supply and installation of one (1) new cluster of five (5) Zinc Rougher Concentrate Cyclones complete with isolation valves, manually adjustable spigots, feed distributor, overflow hopper and underflow hopper.
- Supply and installation of one (1) new open top, cylindrical Zinc Rougher Concentrate Storage Tank complete with overflow, drain, flat base and baffles.
- Supply and installation of one (1) new single open impeller type Zinc Rougher Concentrate Storage Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Supply and installation of two (2) new heavy duty centrifugal Zinc Regrind Mills Feed Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Fabrication and installation of one (1) new compartmentalised Zinc Regrind Mill Splitter Box complete with lid and rubber lining.
- Supply and installation of two (2) new stirred media detritor type Zinc Regrind Mills complete with main shaft, agitator, liners, reducer, main drive motor and automatic lubrication system.
- Fabrication and installation of one (1) new Regrind Mill Media Charging Hopper.
- Fabrication and installation of one (1) new conical Ultrafine Zinc Rougher Feed Pump Hopper complete with overflow, drain and rubber lining.
- Supply and installation of two (2) new heavy duty centrifugal Ultrafine Zinc Rougher Feed Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.

- Supply and installation of one (1) new gantry type Rougher Flotation Area Service Crane complete with dual speed wire rope hoist, motorised travel, transverse lift and roving pendant.
- Fabrication and installation of one (1) new conical Zinc Rougher Tailings Hopper complete with overflow, drain and rubber lining.
- Supply and installation of one (1) new heavy duty centrifugal Zinc Rougher Tailings Pump complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Relocation of one (1) existing compartmentalised Copper Rougher ISA Feed Box from existing Copper Flotation circuit.
- Installation of one (1) new single open impeller type Copper Rougher ISA Feed Box Conditioning Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Installation of one (1) cross-cut Copper Rougher ISA Feed Box Sampler.
- Relocation of one (1) existing compartmentalised Zinc Rougher ISA Feed Box from existing Zinc Flotation circuit.
- Installation of one (1) new single open impeller type Zinc Rougher ISA Feed Box Conditioning Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Installation of one (1) new cross-cut Zinc Rougher ISA Feed Box Sampler.
- Supply and installation of one (1) new heavy duty vertical spindle centrifugal Copper Rougher Flotation Area Sump Pump complete with drive and abrasion resistant rubber lining.
- Supply and installation of one (1) new heavy duty vertical spindle centrifugal Zinc Rougher Flotation Area Sump Pump complete with drive and abrasion resistant rubber lining.
- Supply and installation of one (1) new heavy duty vertical spindle centrifugal Copper Regrind Flotation Area Sump Pump complete with drive and abrasion resistant rubber lining.
- Supply and installation of one (1) new heavy duty vertical spindle centrifugal Zinc Regrind Area Sump Pump complete with drive and abrasion resistant rubber lining.
- Supply and installation of one (1) new Rougher Flotation Area Safety Showers complete with shower and eyewash stations.
- Supply and installation of one (1) new Regrind Area Safety Showers complete with shower and eyewash stations.
- Supply and installation of one (1) building structure including access walkways, platforms, roof and cladding.

3.4.5 Ultrafine Copper Circuit

The new Ultrafine Copper Circuit comprises the following:

- Refurbishment of one (1) existing open top cylindrical Ultrafine Copper Rougher Conditioning Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing single open impeller type Ultrafine Copper Rougher Conditioning Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Refurbishment of one (1) bank of eight (8) existing conventional Ultrafine Copper Rougher Flotation Cells complete with feed box, discharge box, dual concentrate launders, diversion boards, launder spray system, agitator, drive mechanism, drive with support, air header and diverter plates.
- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Ultrafine Copper Rougher Concentrate Pump No. 1 complete with drive and abrasion resistant rubber lining.
- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Ultrafine Copper Rougher Concentrate Pump No. 2 complete with drive and abrasion resistant rubber lining.
- Refurbishment and modification of one (1) bank of six (6) existing conventional Ultrafine Copper Cleaner Flotation Cells complete with feed box, discharge box, dual concentrate launders, diversion boards, launder spray system, agitator, drive mechanism, drive with support, air header and diverter plates.
- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Ultrafine Copper Cleaner Concentrate Pump complete with drive and abrasion resistant lining.
- Refurbishment of one (1) existing compartmentalised Ultrafine Copper Cleaner Concentrate ISA Feed Box complete with rubber lining.
- Refurbishment of one (1) existing cross-cut Copper Cleaner Concentrate ISA Feed Box Sampler.
- Refurbishment and installation of one (1) existing single open impeller type Copper Cleaner Concentrate ISA Feed Box Conditioning Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Refurbishment of one (1) existing Ultrafine Copper Cleaner Tailings Hopper complete with overflow, drain and rubber lining.
- Refurbishment of one (1) existing heady duty vertical spindle centrifugal Ultrafine Copper Cleaner Tailings Pump complete with drive and abrasion resistant lining.
- Refurbishment and installation of one (1) existing cross-cut Ultrafine Copper Rougher Tailings Sampler.

- Removal, refurbishment and re-installation of one (1) existing gantry type Ultrafine Copper Area Hoist with dual speed wire rope hoist, motorised travel, traverse lift and roving pendant.
- Fabrication and installation of one (1) new compartmentalised Copper Concentrate Thickener Feed Box complete with lid and rubber lining.
- Refurbishment and re-installation of one (1) Copper Concentrate Thickener (existing Zinc Thickener).
- Refurbishment and installation of one (1) existing heavy duty centrifugal Copper Concentrate Thickener Underflow Pump complete with impeller, split casing liners, mechanical seals, direct coupling with guard and reverse overhead mounted drive motor.
- Refurbishment of one (1) existing open-top cylindrical Copper Concentrate Storage Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing single open impeller type Copper Concentrate Storage Tank Agitator complete with impeller shaft, reducer, drive coupling drive and baseplate.
- Refurbishment of one (1) existing heavy duty centrifugal Copper Concentrate Filter Feed Pump complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Supply and installation of (1) new Copper Concentrate Filter complete with frame, sidebars, filter plates, sliding mechanism, leakage collector, control system and the following ancillaries:
  - One (1) new cylindrical pressure vessel type Copper Concentrate Filter Filtrate Separator complete with ends, nozzles and supports.
  - One (1) new Copper Filter Pressing Water Tank.
  - One (1) new vertical multi-stage centrifugal Copper Filter Pressing Water Pump complete with impeller, mechanical seal, direct coupling with guard, drive motor and baseplate.
  - One (1) new Copper Filter Cloth Wash Water Tank.
  - One (1) new vertical multi-stage centrifugal Copper Filter Cloth Wash Water Pump complete with impeller, mechanical seal, direct coupling with guard, drive motor and baseplate.
- Refurbishment of one (1) existing monorail type Filter Area Hoist.
- Fabrication and installation of one (1) new Ultrafine Copper Rougher Tailings Pump Hopper complete with overflow, drain and rubber lining.
- Refurbishment of one (1) existing heavy duty centrifugal Ultrafine Copper Rougher Tailings Pump complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.

- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal type Ultrafine Copper Area Sump Pump complete with drive and abrasion resistant rubber lining.

- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal type Copper Thickener Area Sump Pump complete with drive and abrasion resistant rubber lining.

- Refurbishment of two (2) existing Ultrafine Area Safety Showers complete with shower and eyewash stations.

3.4.6 Ultrafine Zinc Circuit

The new Ultrafine Zinc Circuit comprises the following:

- Refurbishment of one (1) existing open top cylindrical Ultrafine Zinc Rougher Conditioning Tank complete with overflow, drain, flat base and baffles.

- Refurbishment of one (1) existing single open impeller type Ultrafine Zinc Rougher Conditioning Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.

- Refurbishment of one (1) bank of five (5) existing conventional Ultrafine Zinc Rougher Flotation Cells complete with feed box, discharge box, dual concentrate launders, diversion boards, launder spray system, agitator, drive mechanism, drive motor with support, air header and diverter plates.

- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Ultrafine Zinc Rougher Concentrate Pump complete with hopper, drive and abrasion resistant rubber lining.

- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Ultrafine Zinc Rougher Concentrate Pump Standby complete with drive and abrasion resistant rubber lining.

- Supply and installation of one (1) new heavy duty vertical spindle froth centrifugal type Ultrafine Zinc Rougher Tailings Pump complete with drive and abrasion resistant lining.

- Supply and installation of one (1) new conical type Ultrafine Zinc Rougher Tailings Pump Hopper complete with overflow, drain and rubber lining.

- Refurbishment of one (1) bank of six (6) existing conventional Ultrafine Zinc Cleaner Flotation Cells complete with feed box, discharge box, dual concentrate launders, diversion boards, launder spray system, agitator, drive mechanism, drive with support, air header and diverter plates.

- Refurbishment of one (1) existing heavy duty vertical spindle, froth centrifugal Ultrafine Zinc Cleaner Concentrate Pump complete with drive and abrasion resistant lining.

- Refurbishment of one (1) existing heavy duty vertical spindle, froth centrifugal Ultrafine Zinc Cleaner Concentrate Pump Standby, complete with drive and abrasion resistant lining.

- Fabrication and installation of one (1) new conical Ultrafine Zinc Cleaner Tailings Hopper complete with overflow, drain and rubber lining.
- Supply and installation of two (2) new heavy duty vertical spindle, centrifugal Ultrafine Zinc Cleaner Tailings Pumps complete with drive and abrasion resistant rubber lining.
- Refurbishment of one (1) existing cross-cut Ultrafine Zinc Rougher Tailings Sampler.
- Refurbishment of one (1) existing monorail type Ultrafine Zinc Area Hoist complete with motorised travel, motorised lift and roving pendant.
- Refurbishment of one (1) existing monorail type Filter Area Service Hoist complete with motorised travel, motorised lift and roving pendant.
- Refurbishment of one (1) existing compartmentalised Ultrafine Zinc Cleaner Concentrate ISA Feed Box complete with lid and rubber lining.
- Refurbishment of one (1) existing cross-cut Ultrafine Zinc Cleaner Concentrate ISA Feed Box Sampler.
- Refurbishment and installation of one (1) existing single open impeller type Ultrafine Zinc Cleaner Concentrate ISA Box Conditioning Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Refurbishment of one (1) existing compartmentalised Zinc Concentrate Thickener Feed Box complete with lid and rubber lining.
- Supply and installation of one (1) new Zinc Concentrate Thickener.
- Refurbishment of one (1) existing heavy duty centrifugal Zinc Concentrate Thickener Underflow Pump complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Refurbishment of one (1) existing open top cylindrical Zinc Concentrate Storage Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing single open impeller type Zinc Concentrate Storage Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Refurbishment of one (1) existing heavy duty centrifugal Zinc Concentrate Filter Feed Pump complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Refurbishment and installation of one (1) existing Larox brand pressure Zinc Concentrate Filter complete with continuous belt, plate and frame.
- Refurbishment and installation of one (1) existing Zinc Filter Pressing Water Tank.
- Refurbishment and installation of one (1) existing vertical multi-stage centrifugal Zinc Filter Pressing Water Pump complete with impeller, mechanical seal, direct coupling with guard, drive motor.

- Refurbishment and installation one (1) existing Zinc Filter Cloth Wash Water Tank.
- Refurbishment and installation of one (1) existing vertical multi-stage centrifugal Zinc Filter Cloth Wash Water Pump complete with impeller, mechanical seal, direct coupling with guard, drive motor.
- Refurbishment of one (1) existing heavy duty vertical spindle centrifugal Zinc Area Sump Pump complete with drive and abrasion resistant rubber lining.

3.4.7 Tailings Circuit

The new Tailings Circuit comprises the following:

- Demolition of redundant tailings hopper and tailings pumps.
- Fabrication and installation of one (1) existing compartmentalised Tailings Thickener Feed Box complete with lid and rubber lining.
- Supply and installation of one (1) new Tailings Thickener.
- Supply and installation of one (1) new open top cylindrical Tailings Tank complete with overflow drain, flat base and baffles.
- Supply and installation of one (1) open impeller type Tailings Tank Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Removal, refurbishment and re-installation of two (2) existing heavy duty centrifugal Stage 1 Tailings Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Supply and installation of two (2) new heavy duty centrifugal Stage 2 Tailings Pumps complete with impeller, split casing, casing liners, mechanical seal, direct coupling with guard and reverse overhead mounted drive motor.
- Refurbishment and installation of one (1) existing heavy duty vertical spindle, centrifugal Tailings Area Sump Pump complete with drive and abrasion resistant rubber lining.
- Refurbishment and installation of one (1) existing compartmentalised Tailings Thickener ISA Feed Box complete with rubber lining.
- Refurbishment and installation of one (1) existing cross-cut Tailings Thickener ISA Feed Box Sampler
- Refurbishment and installation of one (1) relocated single open impeller type Tailings Thickener ISA Feed Box Conditioning Agitator complete with impeller shaft, reducer, drive coupling, drive and baseplate.
- Removal, refurbishment and re-installation of one (1) diesel driven Tailings Decant Waterline Priming Pump.

3.4.8 Reagents

The new reagents area comprises the following:

- Demolition of redundant equipment and structures.
- Relocation of re-used reagent tanks for Copper Sulphate and MBS
- Refurbishment of one (1) existing Lime Silo complete with access ladder-ways, exhaust vent, slide gate isolation valve, manhole and fill pipe.
- Refurbishment of one (1) existing insertable bag type Lime Silo Dust Collector complete with reverse pulse system.
- Supply and installation of one (1) new Lime Silo Bin Activator.
- Refurbishment of one (1) existing Lime Silo Rotary Valve complete with reducer and drive motor.
- Refurbishment of one (1) existing screw type Lime Silo Reclaim Feeder complete with reducer and drive motor.
- Refurbishment of one (1) existing open-top cylindrical Lime Mixing Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing single impeller type Lime Mixing Tank Agitator complete with impeller shaft, reducer, drive motor, baseplate and lining.
- Refurbishment of one (1) existing heavy duty centrifugal Lime Slurry Transfer Pump complete with rubber liners, impeller, coupling with guard and drive motor.
- Refurbishment of one (1) existing open-top cylindrical Lime Storage Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing single impeller type Lime Storage Tank Agitator complete with impeller shaft, reducer, drive motor, baseplate and lining.
- Refurbishment of one (1) existing heavy duty centrifugal Lime Slurry Circulation Pump complete with rubber liners, impeller, coupling with guard and drive motor.
- Refurbishment of one (1) existing Lime Area Safety Shower complete with shower and eyewash stations.
- Supply and installation of one (1) new air diaphragm Frother Drum Pump.
- Supply and installation of two (2) new cylindrical closed roof type UPVC Frother Header Tanks.
- Supply and installation of eight (8) new diaphragm type Frother Dosing Pumps.
- Supply and installation of one (1) new Xanthate Drum Tipper.

- Supply and installation of one (1) new Xanthate Dust Collector complete with Exhaust Fan.
- Supply and installation of one (1) Xanthate Mixing Tank.
- Removal of one (1) existing dual impeller Xanthate Mixing Tank Agitator complete with impeller shaft, reducer, drive coupling, drive motor, baseplate and lining.
- Supply and installation of two (2) new progressive cavity Xanthate Transfer Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of two (2) new cylindrical closed roof type UPVC Xanthate Header Tanks.
- Supply and installation of three (3) new progressive cavity Xanthate Dosing Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of one (1) new vertical spindle centrifugal Reagent Area Sump Pump complete with drive.
- Supply and installation of one (1) new Xanthate Area Safety Shower complete with shower and eyewash stations.
- Supply and installation of one (1) new Copper Sulphate Bag Splitter complete with self closing doors, cutter, roof access ladder and discharge chute.
- Supply and installation of one (1) new Copper Sulphate Dust Collector complete with Exhaust Fan in corrosion resistant material.
- Refurbishment of one (1) FRP cylindrical, open top Mixing Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing dual impeller Copper Sulphate Mixing Tank Agitator complete with impeller shaft, reducer, drive motor, baseplate and lining.
- Supply and installation of two (2) new centrifugal Copper Sulphate Transfer Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of one (1) new FRP cylindrical, open top Copper Sulphate Storage Tank complete with overflow, drain, flat base and baffles.
- Supply and installation of one (1) new progressive cavity Copper Sulphate Dosing Pump complete with impeller, coupling with guard and drive motor.
- Supply and installation of two (2) new Copper Sulphate Area Safety Shower complete with shower and eyewash stations.
- Refurbishment of one (1) existing Flocculant Mixing Package complete with bulk delivery unloading system, mixing tank with agitator, in-line mixers, storage tank, dosing pumps and control system.
- Supply and installation of one (1) new progressive cavity Flocculant Dosing Pump complete with impeller, coupling with guard and drive motor.

- Refurbishment of two (2) existing progressive cavity Flocculant Dosing Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of one (1) new monorail type Flocculant Area Hoist.
- Supply and installation of one (1) new CMC Mixing Package complete with bag delivery unloading system, mixing tank with agitator, in-line mixers, storage tank, dosing pumps and control system.
- Supply and Installation of two (2) new cylindrical open top CMC Storage Tanks complete with overflow, drain, flat base and baffles.
- Supply and installation of one (1) new centrifugal CMC Transfer Pump complete with impeller, coupling with guard and drive motor.
- Supply and installation of two (2) new cylindrical UPVC, closed roof CMC Header Tanks.
- Supply and installation of three (3) new progressive cavity CMC Dosing Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of one (1) new vertical spindle centrifugal CMC Area Sump Pump complete with drive.
- Supply and installation of (1) new CMC Mixing Tank Dust Collector complete with Exhaust Fan.
- Supply and installation of one (1) new MBS Bag Splitter complete with discharge chute.
- Removal, refurbishment and re-installation of one (1) existing FRP cylindrical MBS Mixing Tank complete with overflow, drain, flat base and baffles.
- Refurbishment of one (1) existing dual impeller MBS Mixing Tank Agitator complete with impeller shaft, reducer, drive motor, baseplate and lining.
- Supply and installation of two (2) new centrifugal MBS Transfer Pumps complete with impeller, coupling with guard and drive motor.
- Supply and Installation of one (1) new FRP cylindrical MBS Storage Tank complete with overflow, drain, flat base and baffles.
- Supply and installation of one (1) new centrifugal MBS Distribution Pump complete with impeller, coupling with guard and drive motor.
- Supply and installation of two (2) new UPVC cylindrical, closed roof MBS Header Tanks.
- Supply and installation of three (3) new progressive cavity MBS Dosing Pumps complete with impeller, coupling with guard and drive motor..
- Supply and installation of one (1) new MBS Mixing Tank Dust Collector complete with Exhaust Fan in corrosion resistant material.
- Refurbishment of one (1) existing monorail type MBS Area Hoist.

- Supply and installation of one (1) new air diaphragm Copper Collector Drum Pump.
- Supply and installation of two (2) new UPVC cylindrical, closed roof Copper Collector Header Tanks.
- Supply and installation of four (4) new progressive cavity Copper Collector Dosing Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of one (1) new air diaphragm Zinc Collector Drum Pump.
- Supply and installation of one (1) new UPVC cylindrical, closed roof Zinc Collector Header Tanks.
- Supply and installation of three (3) new progressive cavity Zinc Collector Dosing Pumps complete with impeller, coupling with guard and drive motor.
- Supply and installation of one (1) new gantry type Reagent Area Hoist complete with motorised travel traverse, lift and roving pendant.

3.4.9 Services – Water

The new water services plant comprises the following:

- Refurbishment of one (1) existing cylindrical, flat base, concrete Potable Water Tank complete with pump suction, overflow and drain,
- Refurbishment of one (1) existing cylindrical, flat base, concrete Fire Water Tank complete with pump suction, overflow and drain,
- Refurbishment of two (2) existing flat base, concrete Process Water Tanks (one cylindrical, one square) complete with pump suction, overflow and drain,
- Supply and installation of one (1) new cylindrical, flat base, concrete Raw Water Tank complete with pump suction, overflow and drain.
- Supply and installation of one (1) new centrifugal Process Water Pump complete with impeller, seal, coupling with guard, drive motor and baseplate.
- Refurbishment of two (2) existing centrifugal Process Water Pump complete with impeller, seal, coupling with guard, drive motor and baseplate.
- Supply and installation of one (1) new diesel driven centrifugal Fire Water Pump complete with impeller, seal, coupling with guard, drive motor, baseplate and electric driven jacking pump.
- Supply and installation of two (2) new vertical multistage centrifugal Gland Water Pumps complete with impeller, seal, coupling with guard, drive motor and baseplate.
- Refurbishment of one (1) existing fibreglass Gland Water Storage Tank.

- Refurbishment of two (2) existing vertical multistage Gland Water Pumps complete with impeller, seal, coupling with guard, drive motor and baseplate.
- Refurbishment of one (1) existing Plant Run-off Water Tank.
- Refurbishment of one (1) existing Site Run-off Water Tank.
- Refurbishment of one (1) existing submersible centrifugal Run-off Water Pump.
- Supply and installation of one (1) new Water Treatment Package for potable water.

3.4.10 Services – Air

The new air services plant comprises the following:

- Refurbishment of one (1) existing high pressure rotary screw Plant Air Compressor. Duty complete with direct coupling, oil and water separator, drive motor, oil heater, motor heater, enclosure heater and noise enclosure.
- Refurbishment of one (1) existing high pressure rotary screw Plant Air Compressor. Standby complete with direct coupling, oil and water separator, drive motor, oil heater, motor heater, enclosure heater and noise enclosure.
- Supply and installation of one (1) new high pressure rotary screw Plant Air Compressor complete with direct coupling, oil and water separator, drive motor, oil heater, motor heater, enclosure heater and noise enclosure.
- Refurbishment of one (1) existing Plant Air Receiver.
- Refurbishment of one (1) existing cylindrical pressure vessel type Primary Mill Area Air Receiver complete with dished ends and support structure.
- Refurbishment of one (1) existing cylindrical pressure vessel type Secondary Mill No. 1 Area Air Receiver complete with dished ends and support structure.
- Refurbishment of one (1) existing cylindrical pressure vessel type Secondary Mill No. 2 Area Air Receiver complete with dished ends and support structure.
- Refurbishment of one (1) existing Copper Filter Air Receiver.
- Supply and installation of one (1) new cylindrical pressure vessel type Zinc Filter Air Receiver complete with dished ends and support structure.
- Refurbishment of one (1) existing Instrument Air Pre-Filter.
- Refurbishment of one (1) existing refrigerant type Instrument Air Drier complete with pre and post filters.
- Supply and installation of one (1) new refrigerant type Instrument Air Drier complete with pre and post filters.

- Refurbishment of two (2) existing Ultrafine Copper Flotation Low Pressure Air Blowers.

- Refurbishment of one (1) existing Ultrafine Zinc Flotation Low Pressure Air Blower.

3.4.11 Power Supply

- High voltage power – site power will originate at high voltage from a nominal 7 Megawatt diesel fuelled power station. The power station, 2 x high voltage feeders, 2 x high voltage switchboards and 4 x transformers will all be supplied and installed to the site under a fixed term contract power arrangement. Tenders have been called and the submissions have been reviewed. The specified contract term was for an initial 5 year contract with options of an additional 3 years and a further 2 years. The costs resulting from this tender process have been incorporated into the operating costs.

- High voltage feeders – with the connection arrangements as designed the load is approximately balanced across the two high voltage feeders. Also, the major surge loads are confined to a single feeder. This has the advantage that it will minimise the volt drop effect of these surge loads onto the other feeder which supplies the majority 415V small motor loads and building general power requirements.

- New switchroom – centrally located in the new plant area is a new switchroom and transformer compound.

  The transformer compound will be constructed adjacent to the new switchroom and will house the 3 x transformers dedicated to the new 415V crushing MCC, the new 415V regrind mills MCC, and the new general 415V plant MCC No2. A fourth transformer dedicated to the existing 415V plant MCC will be located in a separate compound adjacent to the existing switchroom.

  The new switchroom will be transportable style with elevated ceiling, air conditioned, high floor loading capacity with cutouts for cable access to switchroom equipment, and elevated approximately 1.5m to allow bottom cable access. This building will house the 2 x new high voltage switchboards, 3 x new 415V MCCs, new variable speed drives (wall mounted), marshalling and PLC panel, and distribution panels.

- New motor control centres (MCCs) – these will generally be Form 2 standard with incomer and bus zones separated from Form 1 fixed starter modules. To achieve practical dimensions of the new switchroom there are advantages to have the 2 new MCCs for the crushing and regrind mills as single sided and the new plant MCC as double sided.

- Duty and standby pumps – where there is a duty and standby motor arrangement there has been only one starter assigned in the MCC and one motor circuit cable. The two motors will be fitted with decontactor plugs that can be selected to connect to either of two decontactor sockets. One socket will connect to the active circuit and the other will be a dummy socket. This saves MCC costs, cable and control costs.

Some of these circuits incorporate a variable speed drive. In this case a single VSD unit will be used in conjunction with the single starter.

- Variable speed drives – these will be vendor standard and wall mounted rather than incorporated into the MCC. This reduces the size and cost of the MCCs.

- Control system – the PLC input and output standards set for the existing plant will be carried through to the new plant. The existing standard is relatively basic so the plant will not have a sophisticated level of automated control but will rely to a large degree upon operator control and local control loops for specific applications such as variable speed control. With the exception of controls and interlocks for the complex load groups of the larger plant (eg mills) the PLC functionality will be restricted to only significant interlocking functions and as a front end collection for the new "Citect" plant operations control and reporting system.

  The PLC will be selected to ensure the backpanel of the remote racks can be accommodated in the locations currently occupied by the existing PLC racks and to allow direct transfer of the wiring from the old to the new racks.

- Crushing control panel – a new crushing plant operator room will be installed overlooking the new primary crusher. This will have a crusher control panel to allow the operator to group start and group stop the crushing plant, control the variable speed plant feeder, and to view ready/fault status of the crushing plant. The FOB bin level will also be displayed here with level alarms.

- Lighting will be provided to illuminate the approach to the ROM bin to allow 24 hour operation dumping by the front-end loader. Lighting at the stockpile end will need to be by a mobile lighting tower or similar.

  "Traffic" indicating lights will also indicate the dump status to the FEL operator; ie GO (dump ok) or STOP (bin level full). These dump traffic lights can also be controlled directly from the crushing control panel.

Reagent sumps – although there will be a quantity of collection sumps in the two main reagents areas these will be located in close proximity. This will allow, for each area, a single sump pump arrangement, single MCC starter and single circuit with manual control. Source and destination of pumping is achieved by appropriate positioning of suction and discharge pipes.

3.5 Control Philosophy

3.5.1 General

- Start and stop control of plant is generally achieved by using the local start / stop control station. In the crushing area though it will also be possible to group start and group stop the crushing system components from the crushing control panel located in the crushing control room. In the main process plant area the local start / stop station will be used as it will not be possible to start and stop the majority of plant from the main control room via the Citect and PLC.

3.5.2 Crushing Circuit

Refer to Drawing No. 1344-PF-001

- Pull wire switches, belt drift switches and zero speed switches are fitted to all conveyors. Belt rip switches are fitted to the crusher discharge conveyor and the transfer conveyor. Blocked chute switches are fitted at all transfer chutes. Detection by these devices will cause the plant item to stop and to cascade stop all interlocked upstream plant.
- Level indication and alarms are fitted to indicate the level status of the ROM bin and the Fine Ore bin.

3.5.3 Grinding Circuit

Refer to Drawing No. 1344-PF-002

- The new mill will adopt all the controls and protections as recommended by the vendor.
- To ensure a controlled feed rate into the primary mill, variable speed motor controllers will be fitted to both the primary mill feed conveyor and the emergency feeder. The emergency feeder will be manually controlled but the primary mill feed conveyor will have both an automatic and manual mode. In the automatic mode the feed rate can be manually set at the VSD or on Citect. The feed rate is automatically controlled by using the feedback signal from the weightometer mounted on the primary mill feed conveyor.
- The mill discharge pumps will be variable speed controlled based on maintaining the process slurry at a predetermined level in the respective mill discharge hopper.

3.5.4 Copper and Zinc Rougher Circuits

Refer to Drawing No. 1344-PF-003

- ISA probes will be fitted to the copper rougher ISA feed box.
- Concentrate discharge from the copper rougher cells to the copper rougher concentrate hopper is manually adjusted. A copper rougher concentrate fixed speed pump transfers the concentrate to the copper rougher concentrate cyclones.
- The cyclone underflow passes to the copper rougher concentrate storage tank.
- The above comments are also relevant to the zinc rougher circuit.
- A copper regrind mills feed pump transfers the concentrate to the copper regrind mills. This pump is variable speed controlled based on maintaining the level in the copper rougher concentrate storage tank.
- Discharge from the copper regrind mills passes to the ultrafine copper rougher feed pump hopper and pumped to the ultrafine copper rougher feed ISA box using a variable speed pump that is controlled based on maintaining the level in the ultrafine copper rougher feed pump hopper.

- The above comments also apply to the zinc regrind mills feed pump and ultrafine zinc rougher feed pump.
- To balance the flows in and out of the copper rougher concentrate hopper the level in the hopper is automatically maintained by the addition of makeup process water as needed.
- The above comment also applies for the zinc rougher concentrate hopper.
- The flow of tails from the copper rougher flotation cells to the copper rougher tailings hopper is automatically controlled based upon the level in the flotation cells.
- An ISA station is also located in the zinc rougher feed stream.
- Lime is automatically added to the zinc rougher conditioning tank to maintain a set pH.
- The flow of tails from the zinc rougher flotation cells to the zinc rougher tailings hopper is automatically controlled based upon the level in the flotation cells.
- All the sumps in this area are manually controlled with low level switch cut out.

3.5.5 Ultrafine Copper Circuit

Refer to Drawing No. 1344-PF-004

- The flow of tails from the ultrafine copper rougher flotation cells is based upon level detection to maintain the level in the cells.
- Concentrate from the ultrafine copper rougher flotation cells passes to the ultrafine copper cleaner flotation cells.
- Concentrate from the ultrafine copper cleaner flotation cells passes to the copper cleaner concentrate ISA box. This is fitted with the third set of ISA probes.
- The copper concentrate thickener underflow pump is variable speed controlled based upon the head pressure from the thickener which is representative of the bed level in the thickener.
- All the sumps in this area are manually controlled with low level switch cutout.

3.5.6 Ultrafine Zinc Circuit

Refer to Drawing No. 1344-PF-005.

- The zinc cleaner concentrate ISA box is fitted with the fourth set of ISA probes.
- Lime is automatically added to achieve a set pH of the process slurry within the ultrafine zinc rougher conditioning tank.

- The control philosophy for the ultrafine zinc circuit is otherwise identical to the ultrafine copper circuit.

3.5.7 Tailings Circuit

Refer to Drawing No. 1344-PF-006.

- The tailings thickener ISA feed box is fitted with the fifth set of ISA probes.
- In automatic operation, the tailings thickener underflow rate is controlled to maintain a preset thickener bed pressure (bed level). The second stage tailings pump speed is adjusted to maintain the tailings transfer tank between defined levels. If the pump speed falls below a minimum level then process water is added to the transfer tank to maintain line velocity.

3.5.8 Reagents

Refer to Drawing Nos 1344-PF-007 and 1344-PF-008.

- High level cutout switches are fitted to remote storage tanks to interlock operation of transfer pumps.
- Pumps are started manually locally at the pump.
- Where tank low level is not obvious during local operation of pumps drawing from the tank then low level switches are fitted to provide Citect warning and pump cutout protection.

4. CAPITAL COST ESTIMATE

4.1 Type of Estimate

The capital cost estimate provides sufficient information and support to allow for the arrangement of project finance.

4.2 Estimate Accuracy

The estimate has been compiled to an accuracy of plus or minus ten (10) percent.

4.3 Estimate Methodology

A visit was undertaken to the Benambra site by Metplant personnel to assess the implication of integrating the new plant into the existing plant.

A detailed report assessing the condition of the existing plant was prepared by Metplant in March 2000.

Subsequently, order of magnitude cost estimates were prepared to upgrade the processing facility. These were tabled in May 2001.

From the above work, base engineering was completed to provide sufficient information to complete the estimate.

Process flow diagrams, mass balances and process design criteria were prepared against which plant and equipment were sized and enquiries issued to the market place for fixed and firm quotations.

A general arrangement drawing was produced to define the new works. Quantity take-offs were prepared for the various engineering disciplines and costing accumulated based on Metplant's database.

4.4 Estimate Philosophy

The capital cost estimate philosophy is based on undertaking the work with an EPCM approach. It is proposed that Metplant Engineering Services will provide engineering, procurement and construction management services for the upgrade and recommissioning of the plant.

Contracts will be let in separate packages where subcontracts are let by major work disciplines such as concrete, structural steelwork/platework, mechanical, pipework and electrical installation. Using this approach Austminex will spend the maximum amount of project funds on direct plant and equipment, rather than on contractors risk and profit.

4.5 Capital Costs

4.5.1 Summary

The capital cost estimate in third quarter 2001 Australian dollars to complete the work is AUD20,629,127. A breakdown of the direct costs is given in Appendix D.

4.5.2 Summary by Area

Table 2 - Capital Cost Estimate by Area

Area	Value (AUD)
Crushing Area	4,147,719
Grinding Area	3,773,564
Rougher Flotation and Regrind Area	4,908,672
Ultrafine Copper Flotation and Dewatering Area	1,127,077
Ultrafine Zinc Flotation and Dewatering Area	994,492
Tailings Area	935,171
Reagents Area	1,382,227
Services Area	378,723
Electrical Reticulation	2,981,483
Total Project Estimate	**20,629,127**

Note that indirect items such as EPCM, Site Establishment, Project Insurances and Contingency have been distributed across the facility items on a pro-rata basis.

4.5.3 Summary by Commodity

Table 3 - Capital Cost Estimate by Commodity

No.	Item	Value (AUD)
1.	Equipment Supply, Refurbishment and Installation	8,825,505
2.	Platework Supply and Installation	1,385,162
3.	Bulk Earthworks Construction	408,614
4.	Concrete Construction	1,156,789
5.	Structural Steelwork Fabrication and Erection	926,779
6.	Platform Flooring Supply and Installation	171,259
7.	Stairways Supply and Installation	151,733
8.	Handrailing Supply and Installation	142,439
9.	Architectural Supply and Installation	94,402
10.	Pipework Supply and Installation	1,001,794
11.	Electrical Supply and Installation	2,380,536
12.	**Direct Total**	**16,645,013**
13.	EPCM	2,323,101
14.	Site Establishment	339,685
15	Temporary Works	61,440
16.	Project Insurances	84,000
17.	Contingency	1,175,888
18.	**Total Project Estimate**	**20,629,127**

4.6 Estimate Basis, Assumptions and Qualifications

4.6.1 General

The estimate has been prepared on the basis of providing Austminex with adequate information to allow the assessment of the viability of the project, identify areas of significant capital expenditure and to provide adequate information to sustain initial scrutiny by a financing institution.

The estimate assumes that the project will be executed on an EPCM basis with Austminex engaging into the direct procurement of equipment and sub-contracts necessary to execute the project.

Implementation of the project on an EPCM basis will transfer the risks of cost over-run, engineering warranty, process warranty and schedule over-run to Austminex.

4.6.2 Commodity Cost Evaluation

4.6.2.1 General

Capital costs for each area were assembled using the following basic methods of calculation.

4.6.2.2 Equipment

All major equipment has been sized and reputable Australian suppliers contacted to submit firm price quotations. Some minor equipment items were priced from Metplant's database based on Australian equipment suppliers.

Some equipment items have been sole-sourced to maintain commonality with existing plant.

Final selections have been made based on equipment that has an established track record in base metal processing facilities.

It has been allowed that all equipment is new other than that equipment which is being reused or relocated.

4.6.2.3 Platework

Platework quantities were developed by in-house quantity take-offs from drawings and sketches of similar plant and equipment constructed by Metplant.

A package was issued to the market-place for budget price quotations to supply typical platework. The pricing submitted was used as the basis for the estimate.

4.6.2.4 Bulk Earthworks, Concrete, Structural Steelwork, Platform Flooring, Stairways, Handrailing and Building Cladding

Quantities were developed by in-house take-offs from drawings and sketches of similar works constructed by Metplant

Unit rates were based on pricing received from the market-place for industry rates for experienced contractors.

4.6.2.5 Erection and Installation

Erection and installation rates for mechanical equipment, platework, bulk earthworks, concrete, structural steelwork, platform flooring, stairways, handrailing, building cladding and construction equipment were based on an assessment of manhours for the site works multiplied by hourly rates obtained from pricing received from Western Australian companies.

Allowances have been made in the notional gang rate for each element of construction to provide an average rate.

A 65 hour work week has been assumed.

The labour hourly rate provides for wages, allowance, consumables, small tools, safety equipment, construction equipment, cranage, supervision, indirect labour, overheads and profit.

Productivity has been assumed to be consistent with other remote but established locations within Western Australia.

4.6.2.6 Piping and Electrical

Quantities were developed by in-house quantity take-offs from drawings and sketches of similar plant constructed by Metplant. Unit rates were developed from Metplant's database.

4.6.3 Temporary Facilities, EPCM and Miscellaneous Costs

4.6.3.1 Construction Facilities

Project wide construction facilities such as offices, crib, ablutions, communications, security, temporary services, warehousing, medical and first aid facilities have been included with due allowance for the established plant site.

Access to existing facilities by the construction personnel will be provided by Austminex at no cost to the project.

Allowance has been made to make use of the existing facilities up to six (6) weeks before practical completion. At that time temporary facilities (offices only) will be provided for construction personnel to provide access to Austminex to the existing facilities.

Following practical completion, the temporary facilities will be demolished and commissioning personnel accommodated within the existing facilities.

4.6.3.2 Accommodation and Messing

Accommodation and messing facilities for construction personnel have been included in the notional labour rate.

Pricing was obtained to accommodate personnel on-site or at a regional centre.

The establishment of a temporary construction camp was investigated but pricing received was cost prohibitive.

The rate allowance provides for accommodation and messing of construction personnel out of Omeo, Victoria. The rate includes a travelling allowance to the project site.

4.6.3.3 Engineering, Procurement and Construction Management

The EPCM costs for the project have been compiled based an assessment of manhours for each discipline.

The estimate makes provision for the EPCM engineers services such as design, drawings, specifications, work scopes, procurement, expediting, inspection, site supervision, management, scheduling, cost control, accounting, monitoring, reporting, commissioning and associated expenses.

4.6.3.4 Project Incidentals

All project insurance costs have been included factored from the processing facilities direct costs.

Professional indemnity insurance costs are included in the engineering manhour rates.

4.6.4 Qualifications

The estimate has been prepared on the basis of the following:

a) No allowance has been made for the following items:

- Owner costs.
- Costs of finance.
- Exchange Rate fluctuations after 10 August 2001.
- Feasibility study costs.
- Royalties and technology costs on overseas components not included in Supplier tenders.
- Metallurgical testwork.
- Environmental impact study costs.
- Costs for initial spares.
- Costs for first fill consumables.
- Escalation within the overall project costs.
- Costs for EPCM engineers risk allowance inclusive of warranties, performance guarantees and liquidated damages.
- Cost for diesel fuel, communications, water and power during the construction of the works.
- The Goods and Services Tax.

b) Demolition of the existing crushing plant will be by others at no cost to the project.

c) Power to the plant will be contract supplied.

d) The existing tailings pipeline and dam distribution system is inadequate for the new processing rate. Allowance has been made to upgrade the system.

e) The existing decant water return pumping and pipeline system is adequate for the new processing rate.

f) The components of the estimate shall not be taken in isolation.

g) Adequate spare capacity and physical space does not exist in the area motor control centres to accept the new facilities local to the MCC. Allowance has been made to provide new facilities to receive the new equipment.

h) No geotechnical work has been completed to confirm that the existing ground can receive the new facilities given that new plant soil loadings are not expected to exceed that of existing plant. Allowance has been made in the estimate to complete a survey to confirm design data.

i) Refurbishment of existing plant and equipment has been based on a rudimentary overhaul to bring the plant into service. Self evident defects will be rectified. Once in service, the equipment will be monitored with any failures being rectified by Austminex personnel. No costs have been allowed for the latter.

j) Allowance has been made to reinstate the existing diesel fuel farm, sewage facility, ablution and communication. Once re-commissioned, responsibility for their maintenance will pass to Austminex.

k) The soil in the designated works areas is free-digging. No rock excavation is required. Plant layout has been based on minimising excavation with plant levels established by backfilling.

4.6.5 Assumptions

The following assumptions have been made for the purpose of the preparing the estimate:

- No major buried services exist in the designated work areas.
- All rates applied are "all in" rates and include painting, rolling, margin, waste, testing, surveys etc.
- All construction contractor distributables include overheads, overtime burden, profit, mobilisation, demobilisation, supervision, vehicles, accommodation, tools and other expenses necessary to complete the activities.
- All rates supplied by Metplant will be applicable at that time that construction proceeds.

4.7 Contingency

In addition to the calculated amounts for direct and indirect costs, allowances were incorporated for contingencies, based on an assessment of the degree of definition available for each main cost centre.

The overall project contingency is 6.0%.

The purpose of the contingency is to make specific provision for uncertain elements of costs within the project scope and thereby reduce the risk of costs over-run to a pre-determined acceptable level. Contingencies do not include allowances for scope changes, escalation or exchange rate fluctuations.

The contingency forms part of the EPCM Engineers estimate and is not to be separated out and included as part of the owners costs.

4.8 Capital Cost Reduction Options

There are several potential options to reduce the project capital cost as follows:

4.8.1 Crushing Plant

4.8.1.1 Contractor Designed Plant

An alternative to a well-engineered purpose designed crushing plant is the supply of a facility that is designed and installed by a contract crushing operator.

A contractor style crushing plant would have the advantage of a lower capital cost, but it is important to recognise that there is a risk of operating compromises associated with this style of plant. Some of these compromises may include:

- The major items of equipment would most likely be second hand and their long-term reliability would need to be assessed.
- The attention given to plant layout for operability, housekeeping and maintenance access would need to be assessed.
- Dust control measures would need to be examined.

Two contract crushing operators have expressed an interest in designing and installing a crushing plant for the Benambra project. One operator, Crushing Services International Pty Ltd, has provided a proposal (refer to Attachment Eleven).

The contract lump sum, including mobilisation to site is AUD2,212,000. This compares to the capital cost estimate for an engineered crushing plant of AUD4,147,719 (excluding electrical), which is based on new equipment and designed as fit for purpose.

4.8.1.2 Contract Crushing

Three contract crushing operators have prepared proposals for the contract crushing of ore. A summary of these proposals is shown in Table 4. Refer to Attachment Eleven for the proposals which include terms and conditions.

Table 4 – Contract Crushing Costs

Contractor	Unit Cost AUD/tonne	Mobilisation AUD	Demobilisation AUD
1	4.47	263,400	118,100
2	4.72	360,000	-
3 *	6.60	436,000	-

* Budget price only.

Note that contractor 2 also provides purchase options after three and five years (refer to Attachment Eleven). It should be noted that the unit costs in Table 4 are nett rates and are not directly comparable to the Metplant operating cost of AUD2.74/tonne. These costs variously require power, front-end loader and diesel costs to be added for making a true economic comparison.

4.8.2 Equipment Selection

There are further opportunities to reduce the project capital costs by using alternative equipment to that selected for the study. Some of these opportunities are as follows:

- Primary Feeder

  An alternative to the selected apron feeder is a vibrating grizzly feeder. Vibrating grizzly feeders have been successfully used in many similar applications.

  The cost savings on equipment only is AUD210,000. There will be differences in structural requirements and installation costs for each type of feeder. This would also need to be taken into account for a true comparison and determination of the nett savings. It is likely that the realisable savings will be less than AUD210,000.

  There are also potential operating cost savings arising from reduced wear on the primary crusher wear liners.

- Secondary Crusher

  An alternative to the selected secondary crusher which has been sourced from a recognised and reputable vendor is a low cost Chyi Meang cone crusher. The quality of these alternative crushers is unknown but should they prove to be acceptable, there is a potential cost saving on equipment only of AUD30,000.

- Regrind Mills

  The stirred media mill selection has been based on four only 185 kW units for regrinding, two for the copper and two for the zinc rougher concentrates. Selection of two mills for each application does provide operational flexibility but carries an additional capital cost as compared to the selection of an additional single larger mill.

  Comparative costs of the stirred media mills are:

  – Four off, 185 kW mills @ AUD290,000 each AUD1,160,000

  – Two off, 355 kW mills @ AUD440,000 each AUD880,000

  Purchase of the larger mills represents a potential cost saving on equipment only of AUD280,000. There will be additional cost savings in concrete, structural steelwork, piping and electrical works if the larger mills are adopted.

- Thickeners

  Thickener selection has been based on the industry standard practice of using a conventional unit for flotation concentrates and hi-rate unit for tailings. The thickeners have been sized from the results of laboratory settling tests.

  A recent development in mineral processing de-watering practice has been the introduction of rakeless thickeners. These units have recently been successfully installed and operating in South African and Australian

operations. They represent a lower capital cost option to traditional thickeners but before they can be recommended, testwork would need to be completed to assess their suitability for the proposed applications.

The dimensions of suitable rakeless units as shown below, have been selected by the vendors as equivalent to the respective conventional and hi-rate thickeners.

Comparative costs of the thickeners are:

Zinc Concentrates:

– Conventional (8 m diameter x 2 m wall height)	AUD181,000
– Rakeless (2 m diameter x 9.3 m wall height)	AUD80,000

Tailings:

– Hi-rate (11 m diameter x 2.4 m wall height)	AUD259,000
– Rakeless (2 m diameter x 12.6 m wall height)	AUD180,000

Purchase of the rakeless thickeners represents a potential cost savings on equipment only of AUD180,000.

- Filters

Filter selection has been based on the use of a dedicated filter for each of the two concentrates. An alternative would be to install a single filter and to batch filter the two concentrates. The compromises arising from this practice would need to be assessed and compared to the potential capital cost savings.

New filter for zinc concentrate	AUD406,000
Refurbish existing filter for copper concentrate	AUD185,000
Total cost for two filters	AUD591,000
New single filter for two concentrates (budget price)	AUD508,350

4.8.3 Second-Hand Equipment

Secondhand equipment has not been considered in the capital cost estimate for this study.

A search of the secondhand market has been completed. The objective of the search was simply to identify major items of equipment that may be suitable for the Benambra project. Equipment condition and the refurbishment costs have not been estimated at this stage.

Several suitable items of equipment have been identified, but of course, their future availability cannot be guaranteed. A listing of the items is shown in Table 5.

Table 5 – Potential Secondhand Equipment

	Item	Description	Price AUD	Comment
1.	Apron Feeder	Burmac Heavy Duty 7000 mm long x 1200 mm wide with power pack and hydraulic drive	85,000	"as new" condition
2.	Jaw Crusher	Minyu 42" x 30" with motor, on frame AMC 42" x 30" Jaques 42" x 30" on frame	108,000 111,000 130,000	"excellent" order unused
3.	Ball Mill	Allis Chalmers 4 m dia x 5.2 m EGL with 1275 kW motor	395,000	
4.	Thickener	Conventional 9 m diameter with rake and mechanism	33,000	excludes bridge structure
5.	Filter	Larox PF19-25, Year 1987	200,000	

5. OPERATING COST ESTIMATE

5.1 General

The operating cost for the Benambra concentrator has been estimated at AUD20.48/tonne for a 600,000 tonnes per year operation. This cost excludes a provision for mill liners and major maintenance material items which averages AUD0.57/tonne over a ten year mine life.

A one off cost of AUD100,000 (AUD0.17 per tonne) should also be allocated to the first year of operation as a commissioning expense. This allowance includes the provision of two additional production personnel and three additional trades personnel for the first three months of operation.

The operating cost breakdown is summarised in Table 6 and the annual provision for ten year mine life are in Table 7.

The cost estimates are based on the process flowsheets as shown in 1344-F-001 to 1344-F-010. In addition it has been assumed that all management, production and maintenance personnel associated with concentrator operations will be Austminex employees.

The battery limits for the operating cost estimates are as follows:

- Inclusive of the front end loader feeding the ROM bin.
- Inclusive of the front end loader loading concentrates into trucks and the weighing of trucks on site.
- Inclusive of potable and process water supply from the “Southern cut-off drain” and the “driller’s dam”.
- Inclusive of generated power provided by a contractor.

Table 6 - Process Operating Costs (excluding provisions)

Annual Operating Cost	Crushing AUD	Grinding AUD	Flotation AUD	Cons Thick, Filt'n & Stor AUD	Tail Thick & Disposal AUD	Services, Utilities AUD	Laboratory AUD	Total AUD
Personnel - Supervision	75,075	125,125	125,125	75,075	50,050	50,050	0	500,500
Personnel- Production	491,400	218,400	218,400	117,000	23,400	23,400	117,000	1,209,000
Personnel - Maintenance	125,125	150,150	100,100	70,070	30,030	20,020	5,005	500,500
Power	305,248	2,309,318	1,440,600	288,120	83,053	340,843	47,888	4,815,071
Reagents & Consumables	0	1,000,800	2,396,610	480	28,800	0	27,100	3,453,790
Equipment Hire	330,000	0	0	55,000	0	15,000	0	400,000
Diesel Fuel	91,060	2,500	2,500	17,260	33,500	6,400	0	153,220
Wear Materials	103,800	0	265,600	0	0	0	0	369,400
Minor Purchases	5,000	5,000	5,000	2,000	2,000	2,000	2,000	23,000
Contract Labour (Maint)	35,000	70,000	20,000	5,000	2,000	5,000	0	137,000
Maintenance Materials	83,000	200,000	250,000	104,000	45,000	45,000	2,000	729,000
Total	1,644,708	4,081,293	4,823,935	734,005	297,833	507,713	200,993	12,290,481

Unit Cost	Crushing AUD/tonne ore	Grinding AUD/tonne ore	Flotation AUD/tonne ore	Cons Thick, Filt'n & Stor AUD/tonne ore	Tail Thick & Disposal AUD/tonne ore	Services, Utilities AUD/tonne ore	Laboratory AUD/tonne ore	Total AUD/tonne ore
Personnel - Supervision	0.13	0.21	0.21	0.13	0.08	0.08	0.00	0.83
Personnel- Production	0.82	0.36	0.36	0.20	0.04	0.04	0.20	2.02
Personnel - Maintenance	0.21	0.25	0.17	0.12	0.05	0.03	0.01	0.83
Power	0.51	3.85	2.40	0.48	0.14	0.57	0.08	8.03
Reagents & Consumables	0.00	1.67	3.99	0.00	0.05	0.00	0.05	5.76
Equipment Hire	0.55	0.00	0.00	0.09	0.00	0.03	0.00	0.67
Diesel Fuel	0.15	0.00	0.00	0.03	0.06	0.01	0.00	0.26
Wear Materials	0.17	0.00	0.44	0.00	0.00	0.00	0.00	0.62
Minor Purchases	0.01	0.01	0.01	0.00	0.00	0.00	0.00	0.04
Contract Labour (Maint)	0.06	0.12	0.03	0.01	0.00	0.01	0.00	0.23
Maintenance Materials	0.14	0.33	0.42	0.17	0.08	0.08	0.00	1.22
Total	2.74	6.80	8.04	1.22	0.50	0.85	0.33	20.48

Table 7 - Provisions Allocation

	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Total
Annual Cost	AUD	AUD	AUD	AUD	AUD	AUD	AUD	AUD	AUD	AUD	AUD
Ball Mill Liners	108,000	99,000	207,000	130,000	185,000	207,000	65,000	207,000	185,000	130,000	1,523,000
Maintenance Materials	0	0	325,200	289,000	348,000	260,000	122,000	294,200	296,000	0	1,934,400
Total Provisions	108,000	99,000	532,200	419,000	533,000	467,000	187,000	501,200	481,000	130,000	3,457,400
Annual Unit Cost	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore	AUD/tonne ore
Ball Mill Liners	0.18	0.17	0.35	0.22	0.31	0.35	0.11	0.35	0.31	0.22	0.25
Maintenance Materials	0.00	0.00	0.54	0.48	0.58	0.43	0.20	0.49	0.49	0.00	0.32
Total Provisions	0.18	0.17	0.89	0.70	0.89	0.78	0.31	0.84	0.80	0.22	0.57

5.2 Discussion

The operating cost estimate details are as follows. Note that the Tables referred to in this Section are in Attachment Ten.

- Personnel. (Refer Tables 1 and 2).

  Manning levels and salary and wage structure have been taken from on Metplant's database for similar base metal operations of this scale. Twelve hour operating shifts have been assumed, using four production crews. The workforce as shown in Table 1 comprises the following:

Management/supervision	7
Wages – production	20
Wages – maintenance	7
Total workforce	**34**

  It has been assumed that all concentrator management, production and maintenance personnel will be employees of Austminex. This includes laboratory personnel.

  There is a separate allocation (refer to Table 1) for contract labour (maintenance) who will be brought on to site from time to time for mill re-lines, major shutdowns and general maintenance work.

- On Costs. (Refer Table 2)

  An allocation of 30% has been included to allow for workers compensation, insurance, annual leave, sick leave, superannuation payments and long service leave.

- Reagents. (Refer Table 4)

  Reagent usage rates and costs have been provided by Austminex. Costs include freight to the Benambra Stores compound.

- Grinding Media (Refer Table 4)

  Grinding media consumption and the cost of media are based on data provided by vendors.

- Equipment Hire (Refer Table 5)

  An allocation has been included for the hire of a Cat 980 front end loader(s) or equivalent to feed the crushing plant ROM bin, load concentrates and for general clean-up.

  There is also an allocation for larger crane hire and the hire of minor equipment as required.

- Diesel Fuel (Refer Table 6)

  Costs are based on an estimate of operating hours and consumption for the following equipment:

  - Front end loader (s).
  - Gensets for water supply and return water pumping.
  - Crane.
  - Bobcat.
  - Forklift.
  - Light Vehicles (x3).

  Note that fuel costs for the contract power station have been included in the power cost of AUD0.135c/kWh.

  A fuel cost of AUD0.369/litre FIS has been provided by Austminex.

- Wear Material (Refer Table 7)

  Wear rates and costs for crusher liners, ball mill liners and the ultrafine grinding mills have been based on information provided by vendors and are considered as typical for an ore with a low to medium abrasion index.

  Costs relevant to crusher liner and ultrafine grinding mills have been reported in the annual operating costs. For the ball mill liners however, a reline schedule has been prepared and the associated costs transferred to a provisions allocation.

- Minor Purchases (Refer Table 8)

  An allocation has been included for the purchase of minor items for an ongoing operation. Such items include personal protective equipment, general safety items and minor operating consumables.

- Maintenance Materials (Refer Table 9)

  These are based on information provided by the vendor, for major items of equipment including crushers, mills and filters. For other equipment the costs are based on Metplant's database for similar operations. The annual operating costs include routine plant and equipment maintenance but do not include major equipment overhauls. A major equipment overhaul schedule has been separately prepared and the associated costs transferred to a provisions allocation.

- Power (Refer Table 10)

  Power costs have been based on a power schedule and a unit power cost of AUD0.135/kWh. The power schedule was developed from the equipment list. The unit power cost has been determined by Metplant following enquiries from potential power providers on facility fee charges and expected diesel fuel consumption rates. A fuel cost of AUD0.369/ltre was applied.

5.3 Qualifications

- The operating costs are June 2001 and make no provision for inflation, wage increases or consumables, reagents cost escalation. A portion of the total operating costs may be subject to escalation through CPI adjustment, fuel costs, materials costs, transport costs, wages and salary levels.
- No contingency has been applied to any of the operating costs.
- No allowance has been made for the maintenance of mobile equipment.
- No allowance has been made for capital equipment replacement.
- No allowance has been made to carry out tailings dam extensions or earthworks.
- No allowance has been made for rehabilitation works.
- All operating costs are free of interest and financing considerations. Ore and concentrate stockpile adjustment, depreciation and amortisation costs have been excluded.
- Concentrate road haulage and shipping costs have been excluded.
- No allowance has been made for any royalty payments to any government bodies or private groups.
- No allowance has been made for GST.

6. WORKS IMPLEMENTATION SCHEDULE

6.1 Type of Schedule

The implementation schedule provides sufficient information to describe the duration of the works associated with the upgrade project.

The schedule has not been prepared as a detail construction management tool but to indicate to Austminex the time required to implement the works required to upgrade the plant.

6.2 Schedule

The duration to complete the Scope of Works is estimated at fifty eight (58) calender weeks inclusive of project preliminaries. A detailed breakdown of the project schedule is provided in Appendix C.

6.3 Critical Path

The activities determining the overall project duration are summarised in Table 8 below.

Table 8 – Critical Path

Activity (Complete)	Week No.
Specify bid and award EPCM contract.	12
Award Primary Mill Supply	16
Delivery of certified drawings for Primary Mill	22
Delivery of Primary Mill	48
Practical completion	54
Commissioning	58

6.4 Major Equipment Deliveries

For major equipment items, the delivery schedule as advised by the suppliers are given in Table 9.

Table 9 - Equipment Deliveries

Equipment	Time (weeks)
Primary Feeder	22
Crushers	16
Primary Mill	32
Flotation Cells	25
Regrind Mills	28
Filter	16
Power Station	31
Thickeners	20
Transformers	27
HV Switchgear	27
MCC's	20
Pumps, Slurry	16

6.5 Schedule Basis, Assumptions and Qualifications

The following qualifications and assumptions have formed the basis for the preparing the schedule

- the scope of works is defined by the findings of this study
- no allowance has been made for time required to complete additional metallurgical testwork and studies to further define the works
- no allowance has been made for time required to establish project financing, if required

7. REFERENCES

The following references were utilised for the works.

1. "Benambra Mine, Plant Report and Valuation", Report No. 1273/AXR002, prepared by Metplant Engineering Services Pty Ltd, March 2000.

2. "Australasian Mining and Metallurgy, The Sir Maurice Mawby Memorial Volume", Monograph Series, Monograph No. 19, edited by J.T. Woodcock and J.K. Hamilton, 1993.

3. "Benambra Project, Preliminary Capital Cost Estimate", Report No. 1337/AXD072/046, prepared by Metplant Engineering Services Pty Ltd, May 2001.

ALTERNATIVE PROPOSALS

300,000 TPA PLANT – FIT FOR PURPOSE

Following receipt of the capital estimate for the 600,000 tpa plant, Metplant was again requested to review the cost of providing a 300,000 tpa plant incorporating a new crushing section, fine rougher concentrate regrinding and an upgraded reagent handling and distribution facility.

The cost estimate is $11,522,440 +/- 20%; made up as follows:

AREA	VALUE (AUD)
Crushing Area	3,349,350
Grinding Area	1,001,460
Rougher Flotation and Regrind Area	1,487,020
Ultrafine Copper Flotation and Dewatering Area	869,870
Ultrafine Zinc Flotation and Dewatering Area	502,730
Tailings Area	208,940
Reagents Area	1,393,530
Services Area	194,520
Electrical Reticulation	2,515,020
Total Project Estimate	**11,522,440**

300,000 TPA PLANT – BASIC

Metplant was also requested to provide an order of magnitude estimate for recommissioning the existing plant, but including fine grinding and reagent plant improvements.

Metplant estimated that to recommission the plant as is, but incorporating new regrind mills and reagent facilities will be AUD 6.0 to 9.0 million with an accuracy of +/- 30%. This cost makes no allowance for a replacement crushing plant.

Their comments are as follows:

- Crushing Plant – given the original recommendation to demolish the plant, no informed comment can be made as no investigative work was undertaken as part of the study. An allowance of $250,000 could be made to refurbish the plant but is arguable that any real benefit could be achieved.

- Grinding Area – the capital cost allows to resurrect the circuit as is and replace missing equipment. No costs have been provided to upgrade the cyclones.

- Regrind Area – costs have been allowed to provide 2 off regrind mills, one for each stream.

- Copper and Zinc Flotation – the capital cost allows to resurrect the circuit as is and replace missing equipment.

- Tailings – the capital cost allows to resurrect the circuit as is and replace missing equipment.

- Reagents – the costs have been left as per our estimate for the 300,000 tpa case. This is due to the poor state of repair of the existing plant and the comment made that the original plant operation was compromised by the poor operation of these systems. Also, the full suite of reagents identified as required by the study were not installed originally.

- Services – the capital cost allows to resurrect the circuit as is and replace missing equipment. The exception is the provision of separate potable water facilities and a new gland water system for the tailings pumps.

- Electrics – the cost was factored from the 300,00 tpa case with account taken for the deletion of the new crushing circuit.

AUSTMINEX NL

BENAMBRA PROJECT

CONCENTRATE MARKETING
&
LOGISTICS REVIEW

SUMMARY REPORT

CONTENTS

AUSTMINEX N.L

Benambra Mine Feasibility

Marketing & Logistics Review.

1. Summary.

- On the basis of forecast production of approx. **40,000 mt per year** of both copper and zinc concentrates, there will be no constraints on the placement of both materials into the concentrate market;
- On indicated qualities, both the copper and the zinc concentrates are at the **lower end of the respective quality spectrums**; however, there are no substantive issues from a quality perspective that have been identified to date that will materially effect marketing.
- **Logistics are difficult,** and physical arrangements are a significant area of concern/cost. Ocean shipping will be problematic and relatively expensive. The R.O.C.O.N concept is novel, but requires more investigation. The potential for domestic placement of copper (Port Kembla) and/or zinc (Newcastle) should be pursued.
- Realisation costs (treatment charges, etc) will be aligned to prevailing world terms, which are relatively transparent. There is **no potential** to gain market advantage or "premium", due to the relatively low concentrate grades and tonnages. (However, as above, there may be some "locational benefit" by way of logistics costs avoided if concentrate can be delivered domestically).
- Copper terms on the "spot" market have been volatile over the past 18 months, from a low point of US$25/2.5 cents in mid-2000, to the current level of US$70/7.0 cents. The emergence of China as a major importer of copper concentrates has been a major and destabilising factor. However, annual or longer-term frame contract terms, set in the predominant Japanese smelter pool negotiations, settled at/about **US$75/7.5 cents**, are close to the long-term average.
- Zinc terms have been strongly in smelters' favour for the past 2 years, at/about US$190, basis US$1000 zinc price. Some movement back from these levels is expected in the medium term, with long- term business over 3-4 years duration being written in the range **US$180-185, basis US$1000** zinc price.
- Both smelters and traders will consider provision of **pre-financing** to "lock in" concentrate contracts. Smelters will consider equity investment (generally passive, minority stakes) to secure concentrate supply. Traders

tend to focus on shorter term financing (eg working capital or "pipeline" financing).

- The adoption of financing arrangements tied to concentrate supply may appear to be foregoing some **market flexibility**, however it does ensure secure outlet and regular/accelerated cash flow. In any event, Benambra will be a "price taker", so is unlikely to gain any major benefit from being active of its own right in the concentrate market. The relatively small tonnages probably require sale to single customers only.

2. Concentrate Quality.

a) Copper.

The copper content of conventional sulphide copper concentrate varies between 18-38% copper content, with most falling within the range 28-34%. In the event the Benambra product attains a 25% copper grade, then it will be considered as a low-grade material, and ranked relatively low in most smelters' order of preference. Smelters increasingly prefer higher copper grade materials, particularly as sulphuric acid disposal becomes difficult. Also, smelters are generally running at high capacity rates, and it is normally the smelting furnace (as opposed to refining capacity) that is the plant bottleneck.

Penalties are imposed for deleterious elements. A typical impurity/penalty schedule as follows:

Zinc:	US$2.00 per 1.0% > 2.0%, permissible up to 5.0%
Lead:	US$2.00 per 1.0% > 2.0%, permissible up to 5.0%
Arsenic:	US$3.00 per 0.1% > 0.2%, permissible up to 0.4-0.5%
Antimony:	US$3.00 per 0.1% > 0.05%, permissible up to 0.2%
Bismuth:	US$2.00 per 0.01% > 0.01%, permissible up to 0.04-0.05%
Fluorine:	US$1.00 per 100 ppm >150 ppm, permissible up to 500 ppm
Chlorine:	US$0.50 per 100 ppm > 500 ppm, permissible up to 1000 ppm
Mercury:	US$2.00 per 10 ppm > 5 ppm, permissible up to 30-50 ppm
Nickel*:	US$3.00 per 0.1% above 0.5%, permissible up to 1.0%

(* nickel plus cobalt often considered as a composite for penalty purposes).

On the basis of advice provided to date, at the target grade (+/- 25% cu) the Benambra copper concentrate will rank towards the bottom of the quality spectrum for most smelters. However, it has not indicated evidence of any deleterious elements in recent test-work, plus was relatively "clean" in previous production. Also, it is a relatively small tonnage, so will be a minor component of

any receiving smelter's overall feed blend. (For example, Port Kembla, relatively a small scale copper smelter, has a planned requirement for 320-340,000 mt pa of concentrate).

As a result, the Benambra copper concentrate will be readily saleable over a wide range of potential market outlets (smelters and/or traders). However, it has no "premium" aspect, and the low copper grade will place it at/near the bottom of smelter rankings.

b) Zinc.

Most zinc production is derived from the conventional electrolytic process (roast-acid leach-electrolysis from sulphide solution), which has a relatively stringent requirement in respect of concentrate quality. Generally, concentrate grade is in excess of 50% zinc content, and ranges up to 60%.

A threshold of combined copper, lead and/or silica of 5% is normally imposed. In addition, elements deleterious to the electrolysis process (antimony, arsenic, chlorine, fluorine, cobalt, nickel, selenium, tellurium, tin) are penalised significantly, and may prove a "block" if a particular smelter has reached its tolerable loading of particular elements. Mercury is tolerated, as most plants have mercury removal capability in the sulphuric acid stream.

Iron-bearing residue disposal is a major industry issue, so that the contained zinc: iron ratio is becoming critical. A zinc concentrate with 50% zinc & 10% iron is now viewed as a low grade/high iron concentrate, and is penalised accordingly.

In a context similar to copper, sulphuric acid disposal is becoming problematic, so that lower zinc content concentrate, as well as carrying elevated iron, will also have a prejudicial zinc: sulphur ratio.

A physical aspect of concentrate grain sizing is also emerging. As mines adopt fine grind technology, the average grain size fed to the fluid-bed roasters utilised in most electrolytic smelters is diminishing. This causes a higher proportion of zinc entering the off-gas stream, necessitating enhanced dust precipitation capability and higher operating complexity/cost. Sizing at/below 10 microns is considered "fine grind" in a context where smelter would target average grain size at/above 20 microns.

A typical impurity/penalty regime as follows:

Arsenic: US$1.50 per 0.1% > 0.2%, permissible up to 0.4%

Magnesia: US$1.50 per 0.1% > 0.2%, permissible up to 0.5%

Mercury: US$1.00 per 10 ppm > 40 ppm, permissible up to 250 ppm

Manganese: US$1.50 per 0.1% > 0.3%, permissible up to 0.7-0.8%

Silica: US$1.50 per 1.0% > 2.0%, permissible up to 3.0-3.5%

Iron: US$1.50 per 1.0% > 8.0%, permissible up to 11-12%

The indicated quality from Benambra at 50-51% zinc grade in concentrate will define it as a low-grade concentrate. (Iron content in concentrate to be defined as a priority, however assume it is at/below 10%). However, as with copper, it appears not to have any significant deleterious elements, although silica was problematic in previous production. Similarly, the tonnage is small, and the impact on a smelter feed blend not significant.

3. Concentrate Terms.

a) Copper.

Typical payments for metals in copper concentrates:

Copper: Pay for 96.5% of copper content, subject to a minimum deduction of 1.0 unit.

Gold: Pay for gold content, less 1.0 g/dmt

Silver: Pay for 90% of silver content, subject to a minimum deduction of 30 gms/dmt

Treatment charge is an agreed US$ deduction per dmt of concentrate

Refining charge is based on the payable copper content, calculated in lbs, expressed in US cents/lb of payable copper.

In addition, price participation (plus/minus) is applied, generally only to the refining charge (albeit sometimes to the combined TC/RC). Typically, increase/decrease RC by 10% of the copper price variation above/below an agreed threshold (say 85 cents/lb in current market)

Precious metals, if any, will also attract refining charges:

- for gold, US$5.00-6.00 per payable oz.
- For silver, US$0.30-0.40 per payable oz.

Treatment charges in the “spot” market are volatile. Current business is being written in the range US$65-70/6.5-7.0 cents, mainly for Chinese and to a lesser extent Indian destination.

For the Benambra project, the frame contract terms, essentially as established in annual negotiations between major mines located around the Pacific Basin and the Japanese smelter pool, are most likely to prevail in eventual sales contracts, and should be used for project assessment.

Over the medium-long term, these terms revolve around a mid-point of US$80/8.0 cents, and within a range of maybe US$15.0/1.5 cents of that mid-point. Price participation will vary between 7.5%-15.0%, normally 10%, from a price basis set at/about the prevailing copper price at the time terms are settled (for 2001, generally 85 cents/lb).

Benambra will be a small, marginal tonnage, at the lower end of quality preferences. As a result, it is likely that terms will be less favourable than the major mine settlements, say by US$5.0/0.5 cents. The frame market is in reasonable balance, and over the next 4-5 years, there is no substantial deviation in forecast demand/supply balance for copper concentrates.

As a result, a medium to longer term treatment charge of US$85.00/8.5 cents/lb is recommended as an appropriately conservative forecast for achievable treatment charges for the Benambra project.

b) Zinc.

Typical payment for metal in zinc concentrates:

Zinc: Pay for 85% of the agreed zinc content, subject to a minimum deduction of 8.0 units.

Silver: Deduct 3 ozs (93 gms)/dmt of the agreed silver content, and pay for 65%-70% of the balance.

(Note – cadmium no longer payable, in fact now effectively a penalty).

Treatment charge is an agreed charge, expressed in US$, per dmt of concentrate delivered to the receiving smelter.

Price participation is by way of escalator/de-escalator of the treatment charge, in relation to the prevailing zinc price. This aspect has been varied over recent

years, where the escalators have been set at unusually high proportions. However, they are now moderating towards longer term norms of:

- Plus 13.0-15.0 cents/dmt for each US$1.00/mt increase in the zinc price above US$1000/mt
- Minus 11.0 cents/dmt for each US$1.00.mt decrease in the zinc price below US$1000/mt.

As for copper, the Benambra zinc concentrate will be relatively small tonnage, plus at the lower end of quality preferences. Thus, it is reasonable to assume a US$5.00/dmt higher treatment charge may be applied (although if a "life of mine" or say a 5 year long term contract is established, the frame terms TC level should be achieved as a benchmark reference).

The current (2001 calendar year) benchmark TC is generally US$189.00/dmt, basis US$1000/mt zinc price, virtually identical to that which was set for 2000. It is generally considered that this is at the upper end of the long term range for zinc concentrate terms, reflecting the opening/expansion of a number of world-scale mines (Century, Antimina, Red Dog) over the past 12-24 months. With these new productions now largely absorbed into the supply/demand forecasts, it is expected that terms may tighten over the medium term future (from 2003 onwards). Already, long-term business of 3-4 years is being written at US$180-185/dmt, basis US$1000/mt zinc price.

As a result, a longer term forecast of US$185.00/dmt, basis US$1000.mt zinc price, with price participation of plus US$0.14/minus US$0.11/dmt for each US$1.00/mt variation in the zinc metal price above/below US$1000/mt is recommended for assessment of the Benambra zinc concentrate.

4. <u>Logistics</u>

a) Trucking.

The indicative proposal supplied by N.L.C.G provides a robust basis for assessing and costing the logistics methodology and costing.

Road transport ex the mine is the only available option. The equipment proposed (bogie plus rigid trailer, approx. 32 mt payload) is widely available, and suitable for the task. Previous experience with Benambra concentrates provides adequate knowledge of handling characteristics, esp. the need for "slip agents" to facilitate tipping.

Trucking distances ex mine:

- To Morwell: 315 km (*A$25.20/mt)

- To Melbourne (Appleton Dock): 465 km (*A$37.20/mt)
- To Geelong: 540 km (*A$43.20/mt)
- To Port Kembla: 535 km (*A$42.80/mt)
- To Newcastle (est.): 750 km (*A$60.00/mt)

(*road freight estimate basis "rule of thumb" 8 cents/mt/km for one way trip, incorporating cost of assumed "dead" return).

The N.L.C.G road freight indication at A$26.50/mt is acceptably competitive. There may be A$3-5/mt downward flexibility if more efficient equipment (esp. trailers) can be warranted. This would require a minimum 3 year, and preferably 5 year, contract to provide adequate amortisation. Also, substantial cost & efficiency benefit will be delivered if some back-haul can be incorporated, e.g for fuel (whether it be diesel or other).

b) Intermediate Handling.

The issue then becomes whether to have storage/intermodal transfer at Morwell, for direct feed to shipping berth, as proposed by N.L.C.G, or opportunity for direct supply to receiving smelter (only realistic domestic possibilities are Port Kembla and/or Sulphide Corp. at Newcastle).

The estimate provided for cost of handling/storing at Morwell is very competitive, at the risk it may not be sustainable In particular, the current storage facility envisaged for the task is too small to handle 2 qualities (zinc plus copper) simultaneously. Also, the task entails multi-handling:

- Tipping ex truck (may need vibration)
- Into stock-pile, including "ramping up", by payloader
- Maybe requirement to turn over if concentrate not sufficiently dry. The mandatory Transportable Moisture Limit (TML) likely to be 9-10%.
- Retrieve from stock-pile with pay-loader, and load into container
- Transfer container, either direct on to rail, or via intermediate store. Requires "tarping".

The methodology proposed (direct rail haul to port, direct discharge into vessel via rotating container) is novel, and if practicable, provides a good outcome, particularly in that it obviates intermediate storage at/near berth and associated environmental/clean-up issues in the wharf area.

We have not yet had opportunity to observe the R.O.C.O.N in operation. We have obtained comment from other parties who have utilised the

system for other cargoes, such as grains, and while they agree that system does work, it has draw-backs, and is still subject to question as to overall efficiency, cost & reliability.

Theoretically, it should perform satisfactorily with concentrate. Issues to address:

- availability of adequate & suitable open top containers. Likely to require minimum 150, probably 200-250, containers to ensure efficient delivery under hook, and meet 3,000 wmt/day load rate.
- Provision of secure cover-alls or tarpaulins. Concentrate must be protected from rain to ensure transportable moisture level maintained.
- Extent of concentrate "hang up" when containers inverted & discharged. Probably cannot apply vibration.
- Facility to clean out residual concentrate. Presents economic plus environmental issue. Vacuum system should be available.
- Degree to which rail can service the irregular "campaign" nature of the delivery concept.

The rail leg from Morwell to Melbourne dock-side has the capacity to meet the task. Theoretic corridor capacity 2 million tonnes p.a against current load 0.6 million mt. Also, rail has performed effectively for other larger tonnage bulk ex Morwell to Melbourne docks ("Energy-brix" for export to Germany; char for export to Asia).

Rail line haul cost, plus port handling, at the indication of A$10.50/mt, appears very competitive, and unlikely to achieve any significant reduction. The task will be irregular and "on demand" and thus not particularly attractive to a rail operator.

b) Ocean Shipping.

In the event concentrates are shipped off-shore (or ocean haul to domestic smelters such as Risdon and/or Townsville) shipments should be in parcels of minimum 5,000 wmt – maximum 7,000 wmt, for each separate quality. This reflects the capacity of a single hold in a vessel. For bulk concentrates, it is not feasible to have cargo separation within a single hold.

Vessels participating in the concentrate trade ex Australia are normally 25-40,000 mt capacity, with 4-6 holds. Some smaller vessels may present, but these are often specialist "box hold" types, seeking high freight rates. In effect, Benambra requires the opportunity to fill 1 hold, as a part cargo with other bulk materials originating from south-east ports (eg Port Pirie, Newcastle, Burnie, maybe Geelong for wood-chips/logs, etc) and most likely destined for North Asian destinations.

Ocean freight rates will be variable and volatile. There are very limited bulk shipments ex Melbourne, so the vessel will need to "cost" a separate port call to Melbourne for each lift. However, there is a substantial bulk trade deriving from relatively proximate ports, so there is no significant impact on total voyage diversion/duration. In effect, each voyage fixture is "constructed" from a range of different parcels ex maybe 3-4 load ports. In addition, the discharge ports will probably vary for each parcel, so that in total the voyage may consist 3-4 load ports, plus 2-3 discharge ports.

In this context, due to the "parcel" nature of the trade, rates will vary. For budget purposes, a rate in the high US$20's (say US$28.00/wmt) is proposed, however with recognition that significant variation around this figure will eventuate. Rates from low-mid-US$20's may be achieved if particularly favourable circumstances occur; alternatively, rates up to US$35 could be requested if the cargo does not coincide with other compatible parcels.

The vessels envisaged are relatively "cheap" on a daily hire basis, currently at/about US$7,000/day. As a result, loading time is not particularly critical. However, a minimum of 3,000 wmt/day should be targeted. (Note only "weather working days" are calculated – vessels stop loading when raining, and the time lost due to weather delay is not assessed). Ports with specialist loading facilities for bulk concentrates (eg Port Pirie, Burnie) guarantee load rates of 7-10,000 mt/day. Freight rates ex Melbourne will be proportionally higher to reflect the relatively slower loading, however not punitively so (maybe US$2-3/wmt, and other factors may mitigate in some circumstances).

It is normally the role of the mine to have responsibility/cost for shipping. In effect, the sale is normally basis CIF receiving smelter port. This is normally advantageous, as it retains overall integration of scheduling of production, storage, delivery to port in the one hand, and avoids the likelihood of clashes, and especially the incidence of vessel demurrage (cost say US$7,000/day).

(We would recommend an experienced shipping broker to assist in this area For example, a local Melbourne broker F.B.A, who undertook this role for Denehurst, and are very familiar/expert with this style of parcel business. Brokerage rate normally 5% of agreed freight rate, but can be rebated to 2-2.5% - allow up to US$1.00/wmt for this service).

c) Realisation Expenses.

Other realisation expenses that need to be included in overall selling costs:

- Supervision fees at receiving discharge port and/or smelter. Allow US$0.50/wmt. (Can be mitigated by having own technical person attend & oversee)
- Marine insurance at/about 0.15% of concentrate value.
- Export document handling, say A$0.25/wmt.
- Export wharfage, at/about A$2.00/wmt

Note in particular that most transport and realisation costs are expressed per wet metric tonne. Sales calculations, and treatment terms, are expressed basis dry metric tonne. Convert wmt to dmt at/about 9.0% moisture.

5. Domestic Smelter Alternative.

Delivery that entails ocean shipping is estimated to cost a minimum of A$50/wmt from mine onto vessel (ref. N.L.C.G quote), plus say US$28/wmt, or A$54/wmt, ocean freight, for total A$104/wmt cost from mine to receiving smelter port.

It should be possible to deliver to domestic smelters (copper to Port Kembla; zinc to Sulphide Corp. at Newcastle) in the range A$40-60/wmt.

As well as basic freight savings, there are other advantages in domestic direct mine-smelter delivery:

- Minimise handling steps and hence material losses. A common assumption is that a material loss of up to 0.1% occurs per handling stage. Commercially, for material sold basis load port weights, as opposed to the normal provision of receiving smelter weights, it is normal practice to allow a weight "franchise" (essentially a weight loss allowance) of 0.3-0.5%. For export delivery, potential material losses occur up to the point of weighing into the receiving smelter.
- Reduce "pipeline" quantity and storage requirements. Domestic smelters will receive on a truck-by-truck, as produced basis, normally throughout the entire year.
- Accelerate timing of cash realisation. Domestic smelters will pay on a monthly increment basis, irrespective of actual tonnage. If desired, 2 week, or even weekly, progressive payments may be negotiated.
- Reduce or eliminate need for supervision service at discharge and weighing/sampling. (Normal charge for SGS or similar to supervise at receiving port & smelter approx. US$0.50/wmt).

Domestic mines & smelters will assess the "locational benefit" of the savings to the mine in not having to export, and negotiate a "sharing" of this overall benefit. In addition, the savings/benefits to the smelter in not having to import the corresponding material *may* be incorporate into this net benefit assessment.

For example, if the total saving to the mine in delivering to Port Kembla as opposed to off-shore was A$60/wmt, then the smelter would seek a 50:50 "sharing" of the benefit. This would be treated as an increase to the agreed "world terms" base treatment charge.

In this context, the opportunity to sell into domestic smelters should be pursued. The Port Kembla copper smelter is currently struggling operationally, reaching only 40-50% of design capacity. However, it must be assumed that they will be performing satisfactorily by the time Benambra concentrate is available. The quality is acceptable, and the tonnage relatively minor. Port Kembla are scheduled to import over half their feed from off-shore, which could be readily diverted to North Asian smelters. They should be motivated to pursue the Benambra material.

Sulphide Corp. is receiving delivery in excess of 50,000 mt pa of various materials through Newcastle port. This is expensive (over 20 km road haul from port to site) plus environmentally exposed, as discharge procedures at the port are rudimentary. Direct delivery into site would be preferred. Quality will be acceptable. The I.S.F technology utilised at Sulphide Corp. in fact favours lower grade material.

(For assessment of the economics of these domestic deliveries, recommend that contact be made with Heggies/Chemtrans. HBL are responsible for copper concentrate deliveries into Port Kembla, plus participated in the delivery of concentrate from the Woodlawn mine into both Port Kembla and Newcastle up until that mine's closure. They are specialist hauliers of coal and bulk materials, headquartered in Port Kembla).

There is risk attached to both the domestic smelters. Port Kembla is struggling operationally, and the Japanese owners (essentially Furukawa) may become exasperated with the slow progress, and seek to withdraw. The plant has been "mothballed" previously, when it was a joint venture between CRA & Furukawa. However, over A$300 million has been expended on revamping the plant, and it would be a major "loss of face" for the Japanese to retreat.

Similarly, Sulphide Corp. is not healthy, financially or operationally. It continues to be a loss-making operation. Also, it fails to achieve operational plans, plus has a major environmental challenge. The parent, Pasminco, is also under severe pressure. Sulphide Corp. is one of the non-core assets that Pasminco would sell if a buyer could be found.

However, closure at either plant would be well telegraphed. It would be a planned closure, or sale, not an abrupt bankruptcy, or similar. It is unlikely a concentrate supplier would face any financial loss. In this context, contracts with either, or both, of these smelters is an acceptable risk. In the event of contract failure, adequate time would be available to place the respective concentrate into alternative outlets.

6. Marketing Candidates.

The following market candidates are recommended. All have a generally open interest in potential new-production concentrate sources, such as Benambra..

They will all be capable/interested in purchasing the total tonnage, probably on a long term (5 year) or life-of-mine basis.

Formal approach should not be initiated until there is better definition of concentrate specifications, quantities particularly scheduled availability.

a) Merchants/Traders.

1. Glencore (via Sydney office, Michael O'Keefe)**.**
2. Mrich/Novarco (via London office, Mark Forsyth)
3. Traffigura (via Brisbane office, Craig Walters)
4. Sogem (via Bangkok office, Keith Johnstone)
5. Brandeis (via USA office, Jeff Beck)
6. Newco (via Hong Kong office, Damien Chung)

b) Smelters

- Copper - Port Kembla Copper
Furukawa, Japan
Dowa, Japan
LG Nikko, Japan/Korea
Birla, India
Sterlite, India

- Zinc - Sulphide Corp (Pasminco)
Korea Zinc, Korea/Townsville
Mitsui, Japan
Toho, Japan
Padaeng, Thailand.
Binani, India.

7. Financing.

a) Equity.

Smelters are often interested to provide some equity input to new mine project, with the objective of securing the mine output for feed into their own plant, and/or an increment to their overall book.

Japanese smelters in particular may take relatively small (up to 15-20%) equity stakes. (Recall that Furukawa did this with Denehurst, although they are probably not a candidate now due to bad experience with both Denehurst and now PKK). In this context, they are relatively passive, although require detailed procedural, reporting, etc. regimes. They may also provide an avenue for competitive loan funds, as typically their cost of money is minimal. Japanese smelters are interested in copper and/or zinc potential.

Indian smelting companies (Sterlite & Birla) are active in pursuing equity involvement in copper mines. Their highly protected domestic market enables them to pay “premium” levels for concentrate sourcing. To date, they have sought to take total or dominant positions in the target mine, and would probably prove intrusive in operations: however, a well constructed joint venture can circumvent this issue.

For zinc, as well as the Japanese smelters, Korea Zinc is a potential equity investment participant. They have looked at several opportunities in Australia (eg Elura) but to date have not come to fruition. The Townsville project has dominated their investment portfolio. Now that it is complete, there is some renewed interest and balance sheet capability.

Traders may also consider equity involvement. Where they do participate (eg Glencore in Cobar) they are intrusive, and will seek additional advantage in unduly favourable marketing & commercial arrangements. However, their normal avenue is via provision of long-term off-take contracts encompassing an up-front payment facility.

b) Financing.

Traders (and some merchant banks such as Macquarie, ABN Amro) will participate in the provision of loan funds, particularly if can be secured against physical mine product. In this context, it is most effectively utilised in financing working capital.

Typically, the merchant will seek a long-term concentrate off-take contract, which will include an early/progressive payment facility against product delivered into (say) an interim storage facility. Sometimes this can extend back to payment for concentrate “as produced” at the mine site against simple statement of production/availability from the mine.

Loan funds will normally be made available in US$, up to a 80-90% of the estimated value of the subject concentrate. Interest rates are basis LIBOR, plus 150-200 points. This would be a revolving type facility, with advances closed out

when the relevant deliveries are eventually completed to the end-use smelter, and final values established. This is normally a 3-4 month period.

Alternatively, the trader may seek to “cash and carry” the concentrate, by pricing early (eg month of estimated production, or even month prior to month of estimated production), and utilising a hedging operation on the LME to gain the contango (if available) against his eventual sale QP to the end-use smelter. The end sale QP will normally be month following arrival at the receiving smelter. Thus, for some material, this may extend up to a 6 month period of potential contango earnings. In this context, the trader may accept the risk of absorbing the financing/interest cost against the potential contango earnings.

8. Attachments.

Attached are several extracts from reputed industry commentators that provide background to the current status of the concentrate markets, plus some forward view as to anticipated demand/supply conditions, etc.

These are provided for your information only.

9. Future Action.

It is recommended that the following issues to be addressed prior to initiating formal contractual discussions with prospective buyers:

- Inspection of ROCON facility, and inspection of port facilities, including container clean-out facility.

  Schedule for week commencing 14th May.

- Review of methodology, actual performance, costs, etc. of prior shipments of Benambra concentrates through port of Melbourne. Data available via FBA, Carlton office.

  Schedule for week commencing 14th May.

- Obtain alternative quotations for trucking services, including for domestic delivery to Port Kembla & Newcastle. This to include visit to Port Kembla (and possibly Newcastle) to observe concentrate handling into smelters. Also, export shipping facilities at Port Kembla.

  Schedule for week commencing 28th May.

- Progress definition of detailed concentrate analysis, quantities and availability schedule.

- Continue to investigate possible alternative logistics avenues, for example port of Eden, rail delivery to Newcastle.

8th May, 2001.

Marketing & Logistics Review.

Supplementary Comments.

Further to the Marketing and Logistics review submitted on 8th May, 2001, the following explanatory comments are provided;

Penalties.

The various penalty elements and scales are indicative of the overall market. The actual level of penalty will reflect the prevailing circumstances at the particular smelter. In some circumstances, a smelter will have "room" to absorb a particular element without material impact on the process. The intent of penalties is that they reimburse actual process costs/penalties incurred in treating the various impurities – they are not intended as revenue avenues.

For example, bismuth penalty in copper concentrates *may* be at a lower threshold if the particular smelter has a feed regime that is otherwise relatively low in bismuth, so that there is still some tolerance to accept additional bismuth without impact on the process. Certainly a penalty provision will be included to protect against unexpected excursions above forecast levels, but the threshold may be set higher. For example, the threshold may be set at 0.02% (rather than 0.01%) if the particular smelter has "room" for additional bismuth without incurring operational problems.

This will also be influenced by the tonnage of concentrates being purchased. In this context, the relatively small tonnage of Benambra concentrate, comprising a relatively small proportion of total feed into a smelter, will assist in negotiating more relaxed penalty regimes.

Zinc Treatment Charge.

A traditional "short hand" method of expressing long term zinc treatment charges has

been to express the TC as a proportion of the total zinc metal content of the

concentrate (alternatively, as a proportion of the payable zinc metal content).

This method is not recommended. There has been a structural shift in the share of

zinc revenue as between smelter and miner over the past 5-6 years. This reflects the

change on the smelter side of significant by-products (especially sulphuric acid and

cadmium) deteriorating from revenue items to now being effectively loss-making

products. As a result, the historic proportions have been moving upward, as the

smelters seek to achieve revenue via TC, to compensate for the significant loss of by-

product revenue. It is unlikely that this structural change will reverse: in fact, with

increasing difficulty in sulphuric acid disposal, there is prospect that smelters will

demand even higher TC revenue to offset costs/losses with acid.

Locational Benefit.

The issue of “sharing” in the “locational benefit” involved in the mine delivering to a domestic smelter, as opposed to more distant & costly off-shore smelter destinations is essentially a negotiable issue. There are no hard-and-fast rules, and it often devolves to a simple resolution, where a 50:50 split is adopted. However, there can be significant variations around this, subject to the prevailing circumstances.

For example, if a domestic smelter needs to import substantial feed due to limited availability from domestic mines, then the mine will stand in a stronger negotiating position (this is the case with both PKK and SC at the present time).

Another factor argued will be the sales pattern for the ultimate metal. If the smelter must sell a substantial proportion of final metal off-shore, and pay the sea-freight accordingly, they will pursue the argument that there are no actual savings – it is simply a transfer of the ocean freight from the miner (in concentrate form) to the smelter (in refined metal form).

Obversely, the mine will argue that where the smelter previously imported concentrate, and paid the costs of vessel discharge and delivery into smelter (normally A$10-15/wmt), then these costs are avoided and should form part of the composite of all “costs avoided” by both mine and smelter, and incorporated into evaluation of the eventual TC adjustment.

For the purpose of the current assessment, the appropriate assumption is that the mine and smelter will share 50:50 in the mine’s identifiable savings in delivery to domestic as opposed to off-shore (North Asian) delivery.

These will be fundamentally:

Ocean Freight	say	US$28.00/wmt (A$52.00)
Port & loading costs	say	A$12.00/wmt

Less differential of inland costs mine-port vs mine-smelter

say A$15.00/wmt.

- Share 50: 50 of net savings = A$25:00 approx.

The mine will adjust (increase) the TC component of the terms calculation for this pre-agreed amount. Normally it is included as a specific line item in the terms sheet.

The less quantifiable benefits (reduced material loss, financing benefit, smaller & shorter pipeline) remain with the mine. These are significant, but are difficult to quantify in the manner of “hard” costs such as transport, freight, etc.

COLKEN PTY. LTD.
A.C.N. 006 974 910
Suite 403, Wellesley House, 126 Wellington Parade, East Melbourne 3002
Telephone: (61-3) 9417 7444 Facsimile. (61-3) 9416 2796

Fax: 9620 2147

For: Austminex, Melbourne.

Att: Mr. Andrew McDougall

From: Michael Kennedy/Russ Colwell.

Date: 11 July, 2001

Re: Domestic Smelters.

The following report is provided in response to your request for an explanation of the relative economics of delivering concentrate into domestic smelters as against export to of-shore smelters.

Potential Domestic Outlets.

The potential outlets for Benambra concentrates are Port Kembla Copper for copper and Sulphide Corporation at Newcastle (Cockle Creek) for zinc. Both provide the opportunity for direct surface transport of material from mine into smelter receival yard.

Smelters such as Risdon (Hobart) and Sun Metals (Townsville) would require ocean delivery, and in that context would not be fundamentally different to off-shore delivery. Ocean freight *may* be slightly lower, although vessel availability and opportunity for combination cargoes (plus the possible residual difficulty of coastal cabotage) make this difficult to predict.

The quality of Benambra concentrates should not present difficulty to either PKC or Sulphide. Both have a good capability to utilise lower grade materials, and in fact the economics of lower grade vs higher grade should attract them to Benambra (this is especially so for Sulphide).

For Sulphide, the appropriate contact route is via Pasminco Melbourne office (Mr. Steve Whitehead). For PKC, the initial contact is via the plant (Mr. Mike Tierney). However, fundamental responsibility is via the Japanese owners, and in particular the trading company participant, Nisshio Iwai.

As discussed previously, both of these plants are "limping", and there is some risk of long term survivability. However, they are continuing to operate in a severe price environment (alleviated by the low A$), and a major factor is the cost of closure of both of these plants. They are unlikely to close suddenly.

Delivery Costs.

We have made preliminary contact with Hoggies Bulk Haulage, whose main activities are centred on the Port Kembla region. They are responsible for the transfer of domestic concentrate (eg North Parkes, Cobar) from the rail head into the smelter. They were involved in haulage ex Benambra to Port Kembla during the Denehurst operation.

Their indicative quotes for the road haulage from mine site into the respective smelter receival yards is:

Port Kemba Copper	***A$70.00/wmt.***
Sulphide Corporation	***A$87.50/wmt.***

These quotations are indicative only. Also, we have not looked at the prospect of any back-haul (eg bio-mass from say Baimsdale region to mine-site, or fuel haulage options).

Terms.

The basic terms applied for domestic delivery will be the same world terms as per export assessment. Looking at the current status of the respective concentrate specifications, there is nothing of particular benefit/prejudice to either of the domestic smelters, otherwise than the benefit of low graded concentrates.

The major variant is that of ***location benefit.*** Simply put, the receiving smelter will assess the financial benefit to the mine on avoiding offshore delivery, and seek some share in that. On the other hand, the mine will look at the savings to the receiving smelter in avoiding the cost of vessel discharge and delivery into the smelter, and seek some recognition of those savings in the overall negotiation.

There are no hard and fast rules in this matter - it is a negotiation. While the outcome often results in a 50:50 share in the aggregate savings, this is not always the outcome. In addition, there are direct costs that are measurable and market determined (sea freight, road haulage, etc), plus indirect, more conceptual costs/benefits that are often of significant value (avoidance of material loss, cash flow acceleration, parcel sizes, minimised stockpiles,etc).

Direct Costs.

Looking at the smelter side , the smelters will save on:

- avoiding import wharfage costs (say A$2.00/wmt);
- vessel discharge A$5-7/wmt;
- transfer into smelter, estimate A$2.50/wmt for PKC; A$6-7/wmt for Sulphide.

In total, savings of ***A$12-15/wmt.*** For confirmation, this is consistent with the actual costs of delivering from wharf into SMC Townsville smelter.

Looking at the Benambra side, if we take the NLC economics as a base case (which we think maybe somewhat under-estimated due to the uncertainty re practical methodology):

- mine into vessel A$48.68/wmt
- export wharfage, allow A$2.00/wmt
- ocean freight current market estimate US$23.00/wmt for 7-8,000 wmt parcel Melbourne to main Japan/Korea port @ 0.52 exchange rate, say A$44-45/wmt

In total, cost of export A$94-95/wmt, versus road delivery to domestic smelters of A$70 & A$87.50 respectively, the direct cost savings range from **A$7-8/wmt to A$14-15.**

In this context, in that there is not a predominant benefit to either party, an appropriate outcome is likely be to allow each party to retain their respective savings.

Indirect Benefits.

There are a number of less quantifiable benefits:

- avoidance of material losses. Export delivery will involve numerous transfer points from mine to receiving smelter - probably 4-6 separate points. As a rule of thumb, we assume a loss of 0.1% of material per transfer point.

- reduced cost and management of stockpile points. Difficult to quantify, but a major benefit to both mine and smelter.

- financing/working capital benefit. The material would be trucked regularly, even to a daily dispatch. In that context, the pipeline stock/time will be minimised. Receiving smelters will normally pay 30 days after the completion of the delivery period. The delivery period will normally be 1 month's delivery.

  However, it may also be reduced to a lot by lot basis, normally 500-1,000 wmt per lot. We have experience as low as 500 wmt/lot, or 1 weeks delivery. In this context, the smelter is also benefiting, in that their normal shipped delivery will be minimum 5,000 wmt, more normally 10,000 wmt.

- reduction/elimination of supervision fees. Mine chemist could provide this service for domestic delivery. This can cost A$0.50/wmt.

- more efficient assay exchange/agreement, expediting receipt of final payment (normally 10-20% of invoice value).

The most significant aspect will be in the financing. For export shipment:

- minimum parcels will be 5,000 wmt. more likely 5-7000 wmt to fill a complete hold. Allow time to accumulate the separate parcels, minimum 6 weeks, probably 8 weeks
- allow 2 weeks to match tonnage available to vessel arrival
- allow 3 weeks vessel sailing (will be part cargo so numerous load & discharge ports, causing lengthier voyage)
- payment normally based date of arrival at receiving port, varies from 0-30 days, and sometimes longer. Allow 15 days, say 2 weeks.

In summary, should allow a period of 12-14 weeks, say 3 months, for financing of export stockpile. Measures can be taken to off-set this financing period (such as pre-shipment financing) however the cost is going to be realised to a large extent in terms, conditions, pricing, etc.

In contrast, domestic delivery could bring forward to as prompt as 4 weeks after production, however more likely 6 weeks.

Please advise if you require any further comments in respect of locational benefit, etc.

(Re contact with the copper smelter, we will be pleased to assist with discussions with Nissho. Nissho are the traditional trading company utilised by Furukawa, and at one point had some minor equity share in the PKC venture. They have fallen on hard times over recent years, especially resulting from wool & textile business, and not sure of their current capabilities.).

Regards,

Russ Colwell.

Austimber Industries Pty Ltd

ABN 16 091 633 091
P.O. Box 491
6 - 8 Grey St.
Traralgon Vic. 3844
Telephone: 03 5176 1888
Fax: 03 5176 2777

Tuesday, November 27, 2001

Mr. Andrew McDougall,
Project Manager,
Austminex

Dear Mr. McDougall,

It gives me great pleasure to submit a price to Austminex for the freight management of the copper/zinc concentrate from the Austminex mine at Benambra to Geelong, Port Kembla and Newcastle. The prices Austimber Industries Pty Ltd ("ATI") submits are:

From	To	Price $ per tonne excluding GST
Benambra	Geelong	$ 59.08*
Benambra	Port Kembla	$ 97.83
Benambra	Newcastle	$104.84

*** Price quoted to Geelong does not include transport from Railhead to store, storage facilities and ship loading costs. Refer attached indicative pricing from Geelong Port**

These prices are indicative prices based on

1. Annual production from the Benambra mine of 100,000 tonnes of copper/zinc concentrates per year.
2. Current cost structure for road and rail transport as at August 2001.

Prices will be subject to rise or fall should there be significant changes in transportation costs and or volume available for transport.

ATI's responsibilities

It is assumed that Austminex requires ATI to be responsible for all aspects of the delivery of the processed product ("concentrate") from the Benambra mine to each of the 3 markets in annual quantities to each market as arranged by Austminex.

Product Quality

It is also assumed that Austminex is required to satisfy strict concentrate product quality criteria at each market, which Austminex would therefore require ATI to strictly adhere to. ATI believes that its handling and delivery processes are the only systems, which will satisfy Austminex's strict product quality requirements.

ATI's Product Handling and Delivery Systems

Features of the ATI product handling and delivery systems include:

- ATI will load the concentrate into containers suitable for safe use on road trucks and on the rail. Once in the container it will be completely sealed and not opened until it reaches it final destination – either Geelong, Port Kembla to Newcastle. The Austminex product will be fully containerised from Benambra to its final destination.
- ATI will put 21 tonnes of concentrate into each container, and then put 2 containers on each B double truck for the road trip from Benambra to ATI's East Gippsland Freight Terminal at Bairnsdale. Assuming 420 tonnes production per day, ATI will use 5 B double trucks, each of which will make 2 trips per day from Benambra to Bairnsdale. Two trips per day means that all trips are in daylight hours, avoiding any black ice problems on the road. ATI will also use local, experienced drivers to ensure the safest possible delivery of each bin by truck to Bairnsdale.
- ATI will weigh each bin at its East Gippsland Freight Terminal as it is unloaded. Austminex may wish a direct computerised link from the weighbridge to monitor deliveries.
- The East Gippsland Freight Terminal will be a fully fenced secure hard stand site adjacent to the railway line to Melbourne, Geelong and on to Port Kembla and Newcastle.
- The East Gippsland Freight Terminal is being developed by ATI only after it had secured arrangements with Freight Australia giving ATI the exclusiv2e rights to handle materials from truck to rail at the East Gippsland Freight Terminal. Freight Australia is the private company operating Victoria's rail freight.
- The East Gippsland Freight Terminal satisfies all requirements of local, state and federal governments.
- ATI will load the full containers onto freight trains for railing to Melbourne under ATI's contractual arrangements with Freight Australia. ATI currently uses Freight Australia to rail approximately 300,000 tonnes of pulpwood quality logs from the East Gippsland Freight Terminal to Geelong for processing and export as woodchips from Geelong. ATI has therefore had experience at product handling and processing to strict quality criteria for export markets.
- Containers for Geelong will be railed directly to Geelong where they will be unloaded from the train, delivered to the product export storage area, opened and unloaded. The empty containers will then be reloaded onto the train for delivery back to the East Gippsland Freight Terminal.
- Containers for Port Kembla and Newcastle will be stored in a secure rail depot in Melbourne until there are sufficient containers for a 2,000 tonne freight train to Port Kembla or to Newcastle.

- ATI will liaise with Austminex, Freight Australia, Geelong export port, Port Kembla and Newcastle throughout the product handling and delivery process to reduce any logistical problems.
- It is usual for logistical problems to arise. Examples are unforeseen truck breakdowns, unexpected rail issues such as derailments. ATI will always have extra containers of concentrate at each point along the handling and delivery system to cater for such logistical problems. This is identical to the system ATI currently uses for its pulpwood log handling and delivery system.

ATI has been in the product handling business for over 50 years, concentrated in the forest products business. The company currently manages harvest and delivery of over 500,000 tonnes per year of logs from forests to sawmills, pulp mills and export markets. It also handles over 300,000 tonnes per year of pulpwood quality logs by road and rail through the East Gippsland Freight Terminal as described above. In addition ATI trucks handle around 540,000 tonnes of woodchips per year from sawmills throughout Victoria to Geelong. ATI has over 165 items of machinery it uses for the handling and delivery of these products, and the company has sophisticated systems to manage this business.

Austminex will benefit from the ATI expertise and experience if it selects ATI for the critical concentrate product handling and delivery aspect of the Austminex copper/zinc concentrate business.

I look forward to a positive response from you in the near future, and I welcome any opportunity to fully discuss any aspect of this quotation with you at your convenience.

Yours sincerely,

Robert Crawford
Managing Director
Austimber Industries Pty Ltd

FILE NOTE

SUBJECT: Austimber Industries – Concentrate Handling – Geelong Port

Robert Crawford of Austimber Industries advised by phone on 16th August 2001 that the fixed annual cost for the storage facility is $576,000, whether the shed is used or not. There is likely to be a sublease market available for the shed, in the event that Austminex has no need for it from time to time.

The cost to handle the concentrate from the storage facility to the hold of the ship is $9.61/tonne.

CONCENTRATE SHIPPING COSTS

DESTINATION	POINT OF LOADING, QUANTITY AND COST USD/TONNE				
	MELB 5KT LOTS US$/TONNE	MELB 10KT LOTS US$/TONNE	GEELONG 5KT LOTS US$/TONNE	GEELONG 10KT LOTS US$/TONNE	PT. KEMBLA 5KT LOTS US$/TONNE
JAPAN	30	27	34	31	35
Nth of and incl. SHANGHAI	33	30	37	34	37
Sth of SHANGHAI	32	29	36	33	36

Fax Message

25 Drummond St, (P.O. Box 434), Carlton, Vic., 3053, Melbourne, Australia
Internet: mail@fbaship.com.au Contact: A21AA67#

FBA SHIPBROKERS
Phone: +61-3-9663-9333
Fax: +61-3-9663-2368

REF: FM26441

DATE: 5/04/01 15:24:38

RECEIVED
- 5 APR 2001

To: N.L.C.
Attn: Jeff Goss

Jeff/Fred

Good day

Re: Freight rates for Benambra

Since our last discussions the market been moving in a slightly upward trend. This has been due to the increase in exports and decrease in imports, therefore creating a shortage of vessels available within close proximity of the Australian Coastline in turn pushing rates up.

Cargo size	Loadport	Dischport	rate
5,000mts	Melbourne	Main Copper Port Japan	USD 32.00
		North of incl Shanghai	USD 35.00
		South of Shanghai	USD 33.00
	Port Kembla or Geelong	Japan	USD 35.00
		North of incl Shanghai	USD 37.00
		South of Shanghai	USD 36.00
10,000mts	Melbourne	Japan	USD 28.00
		North of incl Shanghai	USD 31.00
		South of Shanghai	USD 29.00
	Port Kembla or Geelong	Japan	USD 32.00
		North of incl Shanghai	USD 35.00
		South of Shanghai	USD 33.00

LEAVE MORGL UNCHANGED

All above rates based on 3000mts per wwd shexuu load
and 2500mts per wwd shexuu disch

+ +

Again depending on shipment dates, feel can improve somewhat on market rates..

Thks+Rgds

Fred Mhaya
F.B.A. Melbourne

05-APR-2001 15:42 FROM NLC 10 0-96200147 P.02

AUSTMINEX NL

BENAMBRA PROJECT

WORK PLAN VARIATION
TAILINGS FACILITY
ENVIRONMENTAL STRATEGIES

SUMMARY REPORT

This document contains excerpts from the Work Plan Variation and Tailings Management Facility Rehabilitation documents, which were prepared on behalf of Austminex NL by URS.

The index for the full document is included as an indication of where further detailed information can be obtained.

As the geology, mining areas and methods, mineral processing and services are either separately described in other Feasibility Study documents or are the same, or similar, to those described and approved in the original Work Plan; these items have not been included in this summary document.

The focus in this summary document has been on the proposed closure strategy for the Tailings Management Facility (TMF) and environmental issues, as these are the most crucial aspects for having the project approved and licence to operate issued by the statutory authorities.

I ATX Benambra TMF Environmental Strategies.pdf

Date: **10 May 2001**

To: **Andrew McDougall**

From: Stephen Newman

Subject: TMF raising options

Andrew,

This memo presents a number of options for raising the TMF embankment over the life of the Benambra project. We have based these options on the mining plan dated 18 April 2001 provided by Austminex. We have also estimated costs associated with each of the embankment raising options, including sourcing of borrow for underground backfill.

1. CURRENT STATUS OF THE TMF

Allowing for 1.5 m of dead storage in the TMF (1m storm and wave capacity plus 0.5 metres minimum water cover) the remaining capacity available for tailings deposition with the current embankment crest height is approximately 0.22 Mm^3. Careful management of the discharge will be required to fully realise this capacity i.e. the tailings will need to be placed in the existing low points in the TMF which are primarily on the southern side of the impoundment.

Under the proposed mine plan the 0.22 Mm^3 remaining capacity in the TMF will be filled by the first years tailings production (0.21Mm^3) assuming a settled density of 2.1 t/m^3. Therefore, the embankment must be raised within one year otherwise the freeboard will be compromised.

This will require any investigations and design works to commence virtually immediately after ore processing commences. Construction would be best undertaken in the drier months of the year hence scheduling the construction period will be important.

2. EMBANKMENT RAISING OPTIONS

Five options have been considered:

Two embankment raises over the project life that are primarily focused on minimising capital expenditure early in the project. Both the first and second embankment raises recover fill from within the TMF but only the volume of fill required to raise the embankment. This fill recovery would be undertaken in two periods when the embankment is raised;

Two embankment raises over the project life that are primarily focused on minimising capital expenditure early in the project. Both the first and second embankment raises

recover fill from within the TMF. The volume of fill required to raise the embankment and to provide all the underground fill requirements will be recovered. This fill recovery would be undertaken in two periods when the embankment is raised;

Construct the embankment to final height in years 1 and 2 (i.e. one raise only over the life of the project). Borrowing from within the TMF would be to recover fill for the embankment raise and no recovery of fill for underground would be undertaken.

Two embankment raises over the project life. Construct the first embankment raise in year 1 however, excavate from within the TMF and stockpile material for a second embankment raise in year 3 or 4 of the project. This provides greater tailings storage capacity early in the life of the TMF and reduces the total project cost.

Minimising the final crest height (and hence total project life cost) by recovery of all the borrow from within the TMF required for embankment raising and underground backfill in years 1 and 2 of the project.

2.1 ASSUMPTIONS IN DEVELOPING OPTION COSTS AND CONFIGURATIONS

We have assumed that for the options involving more than one embankment raise, the first raise will provide at least an additional three years of tailings storage i.e. four years of storage since re-commissioning of the processing plant including the 1.5 metres required for minimum water cover and wave and flood security. The three year period is a balance between minimising the amount of capital expenditure required early in the mine life by reducing the height of the embankment, and minimising the total number of raises to reach the final crest height. The more raises to the final crest height the greater total cost.

We have assumed that when the embankment is raised it will have a downstream slope of four horizontal to one vertical (4:1), which is the proposed final rehabilitated profile. This downstream angle could be steepened to 2.5 horizontal to 1 vertical (2.5:1) to reduce the initial cost however, this would result in the difference in borrow volume between the two slope angles (4:1 and 2.5:1) not being recovered from within the TMF before inundation by tailings. For a first lift to 1180 mAHD the volume that could be potentially lost would be about 40,000 m^3. At say \$8.4 per m^3 to recover and place the borrow in the embankment, this represents a cost of approximately \$336,000 deferred to later in the life of the project. A similar cost reduction could be applied to all the options and for the options with higher crest heights the cost deferral will be greater.

The embankment and underground fill required will come from within the future TMF inundation zone and will therefore reduce the final crest height. It is assumed that a 20% swell factor will apply to material excavated from the borrow pit and placed in the embankment.

It is anticipated that borrowing activities will be undertaken up to about the 1190 m contour by year 4. The excavation will require a significant bund of material to be left

between the TMF pond and borrow excavation edge to avoid excessive seepage to the borrow pit and unstable conditions.

It will be necessary to disturb some currently vegetated areas on the margins of the TMF. Embankment raising will require some weathered soil type materials to form a low permeability zone in the embankment. The topsoil from this clearing can be set aside and used in rehabilitating the newly opened areas. The synthetic liner on the face of the embankment will need to be extended up the face of the embankment.

The sourcing of lower permeability materials may be problematic. During the original embankment construction the valley floor was excavated and the soil materials were recovered from this area. In the upper portions of the valley, where the current borrow pits are, the thickness of soil will be less. It is recommended that when on site (currently proposed for the 11 May, 2001) a reconnaissance of likely borrow areas be undertaken to ascertain the types of materials available. Any impacts on the proposed TMF raising plan can then be assessed.

2.2 SOURCING EMBANKMENT AND UNDERGROUND FILL

The embankment fill can be sourced from within the TMF inundation zone however, to recover the entire underground requirement of approximately 750,000 m^3 from the inundation zone for a crest at 1183.5 will not be possible.

To recover the additional volume of fill required for underground fill there are two options:

1. expand the borrow pit at the back of the TMF; or;

2. open new borrow pits closer to the Wilga and Currawong mines.

The option presented here assumes all borrow comes from within the TMF and that this excavated volume is available for tailings deposition. If this does not eventuate the figures presented here will need to be reviewed.

In the original Benambra Mine Works Plan tailings placement up to the 1200 m contour was approved. Therefore no new approvals should be required to expand the borrow area at the back of the TMF. This option will involve some clearing although the topsoil excavated would be stockpiled to eventually rehabilitate the area. If additional ore is identified over the life of the project which requires a higher final crest height then this area may become inundated thereby reducing the rehabilitation requirement. This option avoids opening a separate borrow in a previously undisturbed area.

An alternative to borrowing from the rear of the TMF would be to open borrow areas in the vicinity of the mines. If this was to occur approval would be required from the DNRE and it would need to be included in the work plan. It is likely that flora and fauna surveys would be required prior to approval to open any new borrow areas. If

any rare or threatened species were identified it is unlikely approval would be given. In addition there may not be any suitable borrow areas in the vicinity of the mines.

It is recommended that the additional borrow be sourced from behind the TMF.

3. SUMMARY OF OPTIONS AND ESTIMATED COSTS

A summary of each of the options and estimated costs are summarised in Table 1.

Table 1: Summary of costs

Option	Description	Crest height (mAHD)	Embankment Borrow required (m^3)	Total project cost ($millions)	Expenditure timing	Comments
1	Two embankment raises over the project life that are primarily focused on minimising capital expenditure early in the project. Both the first and second embankment raises recover fill from within the TMF but only the volume of fill required to raise the embankment. This fill recovery would be undertaken in two periods when the embankment is raised.	1st raise: 1180 2nd raise:1187.5 (final)	180,000 m^3 300,000 m^3 (480,000 m^3 total)	5.1	Year 1: $2.2 million Year 4: $2.9 million	No allowance for underground borrow. Would need to open new borrow area or add $3 million for borrow from TMF Can defer approximately $336,000 from Year 1 to final rehabilitation if steepen downstream slope angle to 2.5:1.
1(a)	Two embankment raises over the project life that are primarily focused on minimising capital expenditure early in the project. Both the first and second embankment raises recover fill from within the TMF. The volume of fill required to raise the embankment and to provide all the underground fill requirements will be recovered. This fill recovery would be undertaken in two periods when the embankment is raised.	1st raise: 1180 2nd raise:1183.5 (final)	180,000 m^3 (+203,000 m^3 for u/g) 190,000 m^3 (+550,000 m^3 for u/g)	7.7	Year 1: $3.2 million Year 5: $4.5 million	Careful management of borrow activities required to ensure borrowed volume becomes available for tailings storage.
2(a)	Construct the embankment to final height in years 1 and 2 (i.e. one raise only over the life of the project). Borrowing from within the TMF would be to recover fill for the embankment raise and no recovery of fill for underground would be undertaken.	1186.5	420,000 m^3	4.4	Year 1: $4.4 million	No allowance for underground borrow. Would need to open new borrow area or add $3 million for borrow from TMF
2(b)	Two embankment raises over the project life. Construct the first embankment raise in year 1 however, excavate from within the TMF and stockpile material for a second embankment raise in year 3 or 4 of the project. This provides greater tailings storage capacity early in the life of the TMF and reduces the total project cost.	1st raise: 1180 2nd raise:1186.5 (final)	180,000 m^3 300,000 m^3 (480,000 m^3 total)	5.5	Year 1: $3.1 million Year 6, $2.4 million	No allowance for underground borrow. Would need to open new borrow area or add $3 million for borrow from TMF
3(a)	Minimising the final crest height (and hence total project life cost) by recovery of all the borrow from within the TMF required for embankment raising and underground backfill in years 1 and 2 of the project.	1183.5	370,000 m^3	4.0 (+3.0 for underground fill borrowing)	Year 1: $4.0 million (+3.0 for underground fill borrowing)	Lowest overall final crest height
--	Rehabilitation to a naturally generating wetland.			1.7	Year 10	Assumes that all raising options above are with a final 4:1 slope. Rehabilitation options will vary approximately 15% from lowest crest height to highest crest.

Notes:

1. Monitoring and maintenance costs are provisional estimates based on up to five years of routine works. No specific works to address any non-routine items, if they were to occur, have been included.
2. These costs do not account for any operational costs and re-commissioning of the tailings delivery and distribution systems.
3. These costs do not have any contingency applied. It may be prudent to apply a contingency for unknown items.
4. Rates for construction activities have been based on escalated rates from the original embankment construction and rates for similar more recent works. These are considered provisional until confirmed by contract tender rates.
5. Cost for underground backfill is for excavation and stockpile at TMF and does not include transport or placement undergroun

Benambra Tailings Management

Facility Rehabilitation

URS Costings of Options

Option 3 - Wetland Cover

Item	*Description*	*Unit*	*Quantity*	*Rate*	*Amount*
1.0	**Buttress Embankment**				
1.1	Establishment of Men and equipment	L.sum	1	$100,000.00	$100,000
1.2	Strip Topsoil and stockpile	m^3	1050	$3.00	$3,150
1.3	Raise Embankment to RL1184m	m^3	117339	$8.40	$985,648
1.4	Construct subsurface drain	m	240	$50.00	$12,000
1.5	Move v-notech weir	L.sum	1	$1,200.00	$1,200
1.6	Extension of Liner	L.sum	10448	$24.00	$250,752
1.7	Place Topsoil over embankment	m^3	15000	$6.50	$97,500
					$1,450,250
2.0	**Construct Spillway**				
2.1	Strip Topsoil	m^3	1650	$3.00	$4,950
2.2	Strip unsuitable overburden to stockpile	m^3	1650	3	$4,950
2.3	Trim Excavation	m^2	11000	2	$22,000
2.4	Construct dissipation structure and stilling basin	L.sum	1	50000	$50,000
2.5	Construct drop structure	L.sum	1	5000	$5,000
2.6	Construct low flow channel	L.sum	1	50000	$50,000
	Supply boulders to channel	L.sum	1	40000	$40,000
					$176,900
3.0	**Cover tailings with Wetlands Cover**				
3.1	Supply slurry barge with Material	m^3	50000	$1.00	$50,000
3.2	Placement of material with slurry barge	m^3	50000	$10.00	$500,000
					$550,000
4.0	**Decommission south diversion drain/re-vegetate disturbed area**				
4.1	Decommission south diversion drain	L.sum	1	$250,000.00	$250,000
4.2	Re-vegetate disturbed areas	Ha	2	$600.00	$1,200
					$251,200

5.0	**Anoxic Wetland**	L.sum	1	$200,000.00	$200,000
6.0	**Geotechnical Investigations, instrumentation**	L.sum	1	$50,000.00	$50,000
7.0	**Survey**	L.sum	1	$25,000.00	$25,000
8.0	**Design, Documentation and Supervision**	L.sum	1	$200,000.00	$200,000
9.0	**Monitoring**	Report	60	$3,000.00	$180,000
					$3,083,350

Benambra Tailings Management

Facility Rehabilitation

URS Costings of Options

Option 2 - Permeable Cover

Item	*Description*	*Unit*	*Quantity*	*Rate*	*Amount*
1.0	**Buttress Embankment**				
1.1	Establishment of Men and equipment	L.sum	1	$100,000.00	$100,000
1.2	Strip Topsoil and stockpile	m^3	1050	$3.00	$3,150
1.3	Raise Embankment to RL1184m	m^3	117339	$8.40	$985,648
1.4	Construct subsurface drain	m	240	$50.00	$12,000
1.5	Move v-notech weir	L.sum	1	$1,200.00	$1,200
1.6	Extension of Liner	L.sum	10448	$24.00	$250,752
1.7	Place Topsoil over embankment	m^3	15000	$6.50	$97,500
					$1,450,250
2.0	**Construct Spillway**				
2.1	Strip Topsoil	m^3	1650	$3.00	$4,950
2.2	Strip unsuitable overburden to stockpile	m^3	1650	3	$4,950
2.3	Trim Excavation	m^2	11000	2	$22,000
2.4	Construct dissipation structure and stilling basin	L.sum	1	50000	$50,000
2.5	Construct drop structure	L.sum	1	5000	$5,000
2.6	Construct low flow channel	L.sum	1	50000	$50,000
					$136,900
3.0	**Cover tailings with Permeable Cap**				
3.1	Placement of Geomembrane	m^2	200000	$8.60	$1,720,000
3.2	Anchoring of Geomembrane	m^3	6500	$6.00	$39,000
3.3	Establish crusher and produce 150000m^3 of crushed rock	m^3	150000	$10.00	$1,500,000
3.4	Drill and Blast 150000m^3 of rock and load crusher	m^3	150000	$2.00	$300,000
3.5	Supply dredge with crushed rock	m^3	100000	$1.00	$100,000
3.6	Operate dredge (place rockfill)	m^3	100000	$22.00	$2,200,000
3.7	Dewater	m^3	100000	$2.00	$200,000
3.8	Water treatment Plant	L.sum	1	$1,500,000.00	$1,500,000
3.9	Water treatment operating costs	L.sum	1	$250,000.00	$250,000
3.10	Ongoing water treatment cost	L.sum	1	$250,000.00	$250,000
3.11	Supply and place geotextile	m^3	100000	$8.60	$860,000

3.12	Win and place compacted clay 500mm thick	m^3	50000	$6.50	$325,000
3.13	Place Rockfill 1000mm thick	m^3	100000	$12.00	$1,200,000
3.14	Place topsoil over site	m^3	20000	$6.50	$130,000
3.15	Revegetate Site	Ha	10	$600.00	$6,000
					$10,580,000
4.0	**Decommission south diversion drain/re-vegetate disturbed area**				
4.1	Decommission south diversion drain	L.sum	1	$250,000.00	$250,000
4.2	Re-profile/rough grade disturbed areas	Ha	18	$4,500.00	$81,000
4.3	Replace topsoil over borrow area	m^3	9000	$6.50	$58,500
4.4	Re-vegetate disturbed areas	Ha	2	$600.00	$1,200
					$390,700
5.0	**Anoxic Wetland**	L.sum	1	$200,000.00	$200,000
6.0	**Geotechnical Investigations, instrumentation**	L.sum	1	$50,000.00	$50,000
7.0	**Survey**	L.sum	1	$25,000.00	$25,000
8.0	**Design, Documentation and Supervision**	L.sum	1	$200,000.00	$200,000
9.0	**Monitoring**	Report	60	$3,000.00	$180,000
					$13,212,850

Benambra Tailings Management

Facility Rehabilitation

URS Costings of Options

Option 1 - Dry Cover

Item	*Description*	*Unit*	*Quantity*	*Rate*	*Amount*
1.0	**Buttress Embankment**				
1.1	Establishment of Men and equipment	L.sum	1	$100,000.00	$100,000
1.2	Strip Topsoil and stockpile	m^3	1050	$3.00	$3,150
1.3	Raise Embankment to RL1184m	m^3	117339	$8.40	$985,648
1.4	Construct subsurface drain	m	240	$50.00	$12,000
1.5	Move v-notech weir	L.sum	1	$1,200.00	$1,200
1.6	Extension of Liner	L.sum	10448	$24.00	$250,752
1.7	Place Topsoil over embankment	m^3	15000	$6.50	$97,500
					$1,450,250
2.0	**Construct Spillway**				
2.1	Strip Topsoil	m^3	1650	$3.00	$4,950
2.2	Strip unsuitable overburden to stockpile	m^3	1650	3	$4,950
2.3	Trim Excavation	m^2	11000	2	$22,000
2.4	Construct dissipation structure and stilling basin	L.sum	1	50000	$50,000
2.5	Construct drop structure	L.sum	1	5000	$5,000
2.6	Construct low flow channel	L.sum	1	50000	$50,000
					$136,900
3.0	**Cover tailings with Dry Cap**				
3.1	Dewater Tailings	m^3	250000	$2.50	$625,000
3.2	Maintain Dewatering	months	6	$20,000.00	$120,000
3.3	Water treatment Plant	L.sum	1	$2,500,000.00	$2,500,000
3.4	Water treatment operating costs	L.sum	1	$500,000.00	$500,000
3.5	Ongoing water treatment costs	L.sum	1	$250,000.00	$250,000
3.6	Prepare working bench around dam	m^3	16000	$4.00	$64,000
3.7	Construct working pavement	L.sum	1	$50,000.00	$50,000
3.8	Supply geotextile	m^2	200000	$7.60	$1,520,000
3.9	Lay Geotextile	m^2	200000	$1.00	$200,000
3.10	Anchor Geotextile	m^3	6250	$6.00	$37,500
3.11	Supply and place rockfill 1000mm thick	m^3	100000	$6.50	$650,000

3.12	Supply and place geotextile	m^3	100000	$8.60	$860,000
3.13	Supply and place compacted clay 500mm thick	m^3	50000	$12.00	$600,000
3.14	Supply and place Rockfill 1000mm thick	m^3	100000	$6.50	$650,000
3.15	Place topsoil over site	m^3	10000	$6.50	$65,000
					$8,691,500
4.0	**Decommission south diversion drain/re-vegetate disturbed area**				
4.1	Decommission south diversion drain	L.sum	1	$250,000.00	$250,000
4.2	Re-profile/rough grade disturbed areas	Ha	18	$4,500.00	$81,000
4.3	Replace topsoil over borrow area	m^3	9000	$6.50	$58,500
4.4	Re-vegetate disturbed areas	Ha	2	$600.00	$1,200
					$390,700
5.0	**Anoxic Wetland**	L.sum	1	$200,000.00	$200,000
6.0	**Geotechnical Investigations, instrumentation**	L.sum	1	$50,000.00	$50,000
7.0	**Survey**	L.sum	1	$25,000.00	$25,000
8.0	**Design, Documentation and Supervision**	L.sum	1	$200,000.00	$200,000
9.0	**Monitoring**	Report	60	$3,000.00	$180,000
					$11,324,350

AUSTMINEX NL

BENAMBRA PROJECT

STATUTORY APPROVALS

SUMMARY REPORT

A legal opinion was provided by John McMullan, of John McMullan Solicitors, that the original Environmental Effects Statement (EES), approved in 1988, remains in good standing as does the approval to operate the mine.

All that is required to be done, is for Austminex to prepare a Work Plan and have it approved by the DNRE.

The DNRE indicated at the end of year 2000, that it required the preparation of a Variation to the Work Plan, for submission and approval. The intention being to keep the process simple and that the Work Plan Variation document only had to describe those aspects and processes of the redeveloped Benambra Mine which differed from the previous operation. For those aspects and processes which were unaltered, the existing Work Plan would remain in good standing.

A Work Plan Variation document was prepared (with the assistance of URS – Steve Newman ph. 03 9244 3311), with the focus on: changes in Resources and Reserves, changed mining methods at the approved Wilga and Currawong underground resources, details of changes to the metallurgical process and importantly the proposed changes to mine rehabilitation and the closure strategy for the Tailings Management Facility (TMF).

The currently approved Work Plan is based on a dry cover closure strategy for the TMF. This method of closure, for the long term protection of the environment from the generation of acid drainage, is now recognised by the Authorities and industry as being inappropriate. There is agreement that some form of wet closure is now required, which will prevent oxygen reaching the tailings and which will prevent them from producing acid drainage.

Denehurst in Administration has surrendered its environmental bond, with the DNRE having assumed responsibility for the TMF maintenance and closure. The DNRE have spent the available bond money on monitoring, maintenance and the construction of the northern cut off drain. There is no money left from the bond for the implementation of the final closure strategy.

Whilst the DNRE retains environmental responsibility for the site, it has assigned regular monitoring and water sampling responsibilities to Austminex. It is DNRE's intention that Austminex retain this responsibility for the period that Austminex retains an interest in taking over the property. Austminex through its contract caretaker, Ron Thumerer is required to make weekly observations and take water level readings. In addition there is the requirement for more comprehensive sampling at three month intervals and also the requirement for Austminex to pay for the services of Thiess who maintain and collect data from an automatic stream sampling station. The ongoing cost per month for Austminex is $2,000 to $2,500 per month. The information gathered by Ron Thumerer and Thiess is forwarded to Terry McKinley at DNRE offices Melbourne. Until Austminex takes over the property or the DNRE implements a closure strategy for the TMF, the DNRE is responsible for the dam maintenance and controlled water releases, which are the subject of the approval by the EPA. The DNRE are currently seeking a Works Approval from the EPA for the release of water from the TMF on a needs basis, without having to resort to the current requirement of seeking

permission from the EPA for an “emergency release” on the occasions that a release is required. Since the closure of the mine in 1996 there have been two water releases by means of a syphoning system. The TMF currently doesn’t incorporate a spillway.

The DNRE commissioned environmental consultants, MPA Williams, to investigate the condition of the TMF and to recommend a suitable closure strategy. MPA Williams considered and provided a preliminary costing for a whole range of closure options. Their favoured options were lake (direct water cover) and saturated soil cover. They discounted the previously approved dry cover as being ineffective. The DNRE and EPA did not approve of the lake cover proposal, but favoured the multi layered saturated soil cover. There are alternative expert opinions which suggest that the DNRE favoured method will encounter construction problems, result in environmental disturbance of other areas in the winning of cover materials, may have a negative long term environmental outcome and is likely to cost considerably more that the preliminary cost estimate of around $5.0M.

The final environmental bond will be set at a level which reflects the anticipated cost to implement the approved closure strategy.

If Austminex was to take over the leases from the Denehurst Administrator, it would immediately have to pay the DNRE a preliminary environmental bond of $1.5M and assume all environmental responsibility for the property, including the closure of the existing TMF. Austminex is not prepared to take over the mining lease, which is currently valid until April 2004, until it is assured of having an economic project and the issue of the closure strategy for the TMF is satisfactorily resolved. Austminex has currently factored into its financial model an environmental bond of $4.0M, based on the modified lake style TMF closure option, as recommended by URS environmental consultants. If Austminex is required to comply with the DNRE’s currently preferred closure option, it is unlikely that Austminex will make a decision to proceed with the project, based on environmental outcome concerns, the bond level and the cost of implementation.

Whilst Austminex has submitted its Work Plan Variation document to the DNRE for consideration, procedurally the licence cannot be issued and the Work Plan approved until Austminex formally takes over the leases. The purpose of submitting the Work Plan Variation Document was to seek approval in principle.

The DNRE has formally responded, following the submission of the Work Plan. In their response they have indicated, contrary to previous advice, that they wish to have the original Denehurst Work Plan and the Variation documents consolidated into one document. In addition they have asked for clarification on a number of issues, most of which are easily answered and many of which can be addressed by submission of the final mining proposal report. The major sticking point is the closure strategy for the TMF. Whilst verbally the DNRE have said that it can see some merit in the Austminex water cover closure strategy, it will not go against the requirements of the EPA, which at the moment appears to retain favour with a saturated multi layered soil cover.

Austminex N.L. Benambra

Austminex to date has not replied to the DNRE "response" letter. The focus has been on trying to have the EPA see the merit in our closure proposal and to expedite a favourable decision. The decision process is proving to be tortuous, with current indications that the EPA will not accept the water cover proposal as presented. The first point of contact with the EPA is Garry Kay, ph. 03 5176 1744. He is an advisor, with the decision having to come from others higher up the organization, such as David Mackenzie, Dave Horsman and Rob Joy.

Further to what is written above, there have been recent discussions (23rd November 2001) between URS and the EPA regarding the EPA's closure strategy position. It appears that the EPA has moved a considerable way in accepting a wet closure, but will not accept a straight water cover or the organic waste layer. They are seeking a combination of a water cover and a layer of crushed rock over the entire surface of the tailings. Austminex has still to be formally advised of the requirements and the details. Once these are known the cost of complying with the EPA closure strategy can be established. A file note indicating the position, as understood at present, is attached to this Statutory Approvals summary report.

In respect to the DNRE there has been a change in personnel responsibility and the current contacts for Work Plan negotiations are Geoff Oxley (Environmental Officer) and Mike Mathews (Mines Inspector). Both these people are based in Benalla, ph.03 5761 1504. Other personnel who will be involved in the decision process are, Terry McKinley, Doug Sceney and Robert King. These personnel are based in the DNRE offices at 250 Victoria Parade Melbourne.

The following leases and their status, comprise the Benambra Tenements (verification of status and conditions can be obtained from Graham Robertson – Australian Mineral Services ph. 03 9842 7694):

- Mining Lease ML1865 – held in the name of Denehurst and is current until April 2004, with the process for renewal to commence at least 6 months prior to expiry. At present transfer of this title to Austminex will require an immediate environmental bond payment/facility to be made by Austminex to the DNRE and will also result in Austminex having full environmental responsibility and liability for the cost of the agreed final closure strategy. Austminex has written to George Buckland (DNRE) seeking some relief from these requirements, so that Austminex can take over the title without having to accept full environmental responsibility, until the decision to proceed with mine redevelopment is made and operations commence. A response to this letter is awaited.
- Mineral Licences MIN 4279 and MIN 4281 – these are in the name of Denehurst and are current until 11th April 2004.
- Exploration Licence EL 3458 (Banksia) – this is in the name of Denehurst, is pre native title and has been recently renewed, with currency until 10th August 2002. There is currently a nil expenditure requirement on this licence.

- Exploration Licence Applications ELA 3980 and ELA 3984 (now combined into ELA 4599) – the former ELA's 3980 & 3984 were held in the name of Denehurst. They were scheduled to expire in August 2001. The Denenhurst Administrator indicated that he was not willing to pay the cost of renewal. The DNRE was seeking a reduction in area covered by these two ELAs. Austminex conducted a review of prospectivity and was able to recommend areas for relinquishment and new boundaries. The revised areas have been consolidated into one new ELA 4599 and upon recommendation and agreement of the Administrator, the application has been made directly in the name of Austminex NL. The application process is in progress and is subject to public advertising and native title negotiations. The advertising has taken place but to date there have been no discussions with native title parties.

For additional information, the following documents are attached:

- Legal opinion – John McMullan Solicitors
- DNRE, George Buckland letter, 29th January 2001 re Mining Lease 1865 responsibilities
- DNRE, George Buckland letter, 2nd March 2001 re Mining Lease 1865 responsibilities
- Variation to Work Plan response letter from Robert King, DNRE
- Note on long term containment of potentially acid forming tailings – sent to Dave Horsman, EPA 3rd September 2001
- File note re conversation with Garry Kay, EPA, dated 15th October 2001
- Email to Dave Horsman, EPA, 18th October 2001, re TMF closure strategy
- File note re conversation with Gary Kay, EPA and Geoff Oxley, DNRE, dated 9th November 2001
- File note re URS (Jeff Bazelmans) – EPA discussions re tailings facility closure strategy, 23rd November 2001
- Memorandum from Graham Roberston, dated 1st March 2001, re Bonds
- Letter from DNRE, George Buckland, dated 6th June 2001, re Benambra Tenements Status and approvals process
- Letters and faxes re ground relinquishment and licence renewal for ELAs 3980 and 3984 (replaced by consolidated ELA4599), dated 26th April, 27th April, 30th April, 30th July, 8th August and 13th August 2001
- Letters and faxes re renewal of EL 3458, Banksia, dated 3rd August, 7th August and 20th September 2001
- Letters and faxes re surrender of EL3915, dated 26th March, 18th April and 23rd April 2001

JOHN MCMULLAN
SOLICITORS

Level 46, 525 Collins Street
Melbourne Vic 3000
Tel: 03 9629 6400
Fax: 03 9629 4775
email: [illegible]
ABN 13-004-253-984

8 May, 2001

Attention: Mr R King
Department of Natural Resources and Environment
240 Victoria Parade
EAST MELBOURNE VIC 3002

By Facsimile

Dear Sir

Benambra Mining Lease 1865
Application for Variation to Work Plan
Planning Status under East Gippsland Planning Scheme

I am acting for Austminex NL, in relation to satisfying DNRE as to the planning approval requirements at the Benambra Mine, for an Application to DNRE for a Variation to the Work Plan (paragraph (h) of your letter to Austminex NL dated 13 November 2000). We would like to discuss the DNRE requirements with you.

I enclose a copy of a Memorandum to Austminex NL confirming my view that, on the material before me, no further planning approval is required for the Benambra Mine.

I will telephone you to discuss a possible meeting with the appropriate officer at DNRE. Please call me at any time.

Yours faithfully

JOHN McMULLAN

jm/dave010508

MEMORANDUM

This memorandum is directed to the present planning status of the Benambra Mine, for the purpose of the owner, Austminex NL, seeking Department of Natural Resources and Environment (DNRE) approval to the Work Plan for the mine.

Summary

I have reviewed the following:

- Omeo Planning Scheme (now superseded by the East Gippsland Planning Scheme)
- Amendment L1 to the Omeo Planning Scheme
- East Gippsland Planning Scheme
- Solicitor's report dated 21 July 2000, attached to Austminex Prospectus, prepared by Mr P Hopkins
- Memorandum dated 6 July 2000 from Mr G Peake, of counsel, as to the planning status of the mine (addressing certain non-conforming use right issues)
- Supplementary Memorandum dated 13 July 2000 from Mr G Peake, of counsel (clarifying certain aspects of the earlier memorandum)
- Letter dated 13 November 2000 from DNRE to Austminex
- Letter dated 12 January 2001 from East Gippsland Shire to Austminex

I confirm my view that on the basis of this material, no further planning approval is required for the Benambra Mine.

Discussion

The present position is as follows:

1. The Benambra Mine (the Mine) is within the East Gippsland Planning Scheme. It was previously within the Omeo Planning Scheme (since superseded by the East Gippsland Planning Scheme), and was the subject of Amendment No. L1 to the Omeo Planning Scheme. Amendment L1 was approved by the Minister for Planning and Environment on 23 December 1988.

2. In brief, Amendment L1 designates the area marked on the plans as a Mineral Exploration Area.

3. Clause 6.4 of Amendment L1 provides, so far as relevant, as follows:

Notwithstanding any other provision of the Omeo Planning Scheme, development and use for the purpose of mineral prospecting/exploration and mineral evaluation/development may be undertaken without the consent of the Responsible Authority, provided the provisions of this clause are complied to the satisfaction of the Responsible Authority.

The Amendment sets out the detailed conditions on the use and development of the mine. There is no apparent failure to comply with those conditions.

4. The Omeo Planning Scheme was superseded by the East Gippsland Planning Scheme. The relevant scheme affecting the land at present is the new format East Gippsland Planning Scheme which was gazetted on 26 August 1999.

5. Clause 63 of the East Gippsland Planning Scheme provides, so far as relevant, as follows:

63 EXISTING USES
63.01 Extent of existing use rights
An existing use right is established in relation to use of land under this scheme if any of the following apply:
_ The use was lawfully carried out immediately before the approval date.
_ .....
63.02 Characterisation of use
If a use of land is being characterised to assess the extent of any existing use right, the use is to be characterised by the purpose of the actual use at the relevant date, subject to any conditions or restrictions applying to the use at that date, and not by the classification in the table to Clause 74 or in Section 1, 2 or 3 of any zone.
63.03 Effect of definitions on existing use rights
The definition of a term in this scheme, or the amendment of any definition, does not increase or restrict the extent of any existing use right established prior to the inclusion of the definition or amendment.
.....
63.05 Sections 2 and 3 uses
A use in Section 2 or 3 of a zone for which an existing use right is established may continue provided:
_ No building or works are constructed or carried out without a permit. A permit must not be granted unless the building or works complies with any other building or works requirement in this scheme.
_ Any condition or restriction to which the use was subject continues to be met. This includes any implied restriction on the extent of the land subject to the existing use right or the extent of activities within the use.
_ The amenity of the area is not damaged or further damaged by a change in the activities beyond the limited purpose of the use preserved by the existing use right.
63.06 Expiration of existing use rights
An existing use right expires if either:
_ The use has stopped for a continuous period of 2 years, or has stopped for two or more periods which together total 2 years in any period of 3 years.

In the case of a use which is seasonal in nature, the use does not take place for 2 years in succession.

6. The non-conforming use rights are also affected by section 6(3) of the *Planning and Environment Act 1987 (Vic)* which provides, so far as relevant, as follows:

> *.....nothing in any planning scheme or amendment shall:*
> *a) prevent the continuance of the use of any land upon which no buildings or works are erected for the purpose for which it was being lawfully used before the coming into operation of the scheme .....*

The potential extinguishing of those rights is affected section 6(4) of the *Planning and Environment Act 1987 (Vic)* which provides, so far as relevant, as follows:

> *Subsection 3 does not apply to a use of land –*
> *a) which has stopped for a continuous period of two years; or*
> *b) which has stopped for 2 or more periods which together total two years in any period of three years; or.....*
> *c) in the case of a use which is seasonal in nature, if the use does not take place in two years in succession.*

7. In my view, the mine operations, in a town planning context, have never ceased. It appears that the Mine operations (as an operating mine) ceased mining activity on 31 July 1996. Since that time:

- the plant and infrastructure was placed on care and maintenance
- the plant has not been decommissioned
- the plant has at all times remained on site, pending a recommissioning of the mine
- the plant has been kept secure
- all licenses have been kept current
- the owner of the Mine has never abandoned, or otherwise evinced an intention to cease mining

8. For the reasons set out in Mr Peake's memorandum (I do not repeat the analysis, except to record that I agree with the reasoning and the conclusions), in my view the Mine is still affected by non-conforming use rights.

9. The land is partly zoned Rural Zone, and partly zoned Public Conservation Zone, under the East Gippsland Planning Scheme. Clause 52.08 of the East Gippsland Planning Scheme provides, so far as relevant, as follows:

> ***52.08 MINERAL EXPLORATION AND MINING***
> ***Purpose***
> .....
> *To ensure that mining is not a prohibited land use.*
> *To allow land to be used and developed for mining either subject to a permit (except in*

those areas where a permit is not required) or subject to the preparation of an environmental effects statement under the Environment Effects Act 1978.
To ensure that mining is carried out in accordance with acceptable environmental standards.

.....

S2.08–2 Mining
Permit requirement
A permit is required to use or develop land for mining.
This does not apply if .....

All the following three requirements are met:

> *An environment effects statement has been prepared under the Environment Effects Act 1978 for the work proposed to be done under a licence issued under the Mineral Resources Development Act 1990.*
>
> *An assessment of that statement by the Minister administering the Environment Effects Act 1978 has been submitted to the Minister administering the Mineral Resources Development Act 1990.*
>
> *An authority to commence work has been granted by the Secretary of the Department of the Natural Resources and Environment with the approval of the Minister administering the Mineral Resources Development Act 1990.*

.....

10. It appears that, in fact, each of the three conditions necessary for the land to be used for the purpose of mining without a permit have been met. In particular:

    a) an Environment Effects Statement under the Environment Effects Act 1978 was prepared in 1988;

    b) the EES has been assessed by the Minister administering the *Environment Effects Act 1978* and has been submitted to the Minister administering the *Mineral Resources Development Act 1990*;

    c) An authority to commence work was granted by the Secretary of the Department of the Natural Resources and Environment with the approval of the Minister administering the *Mineral Resources Development Act 1990*.

11. The commencement of mining without a planning permit is also affected by section 42(7) of the *Mineral Resources Development Act 1990*. Section 42(7) provides, so far as relevant, as follows:

> *If .... a planning permit is required to be obtained for carrying out mining on the land covered by a mining licence in accordance with that licence, the licensee is not required to obtained a permit for that work if:*
>
> *a) an Environment Effect Statement has been prepared under the Environment Effects Act 1978 on the work proposed to be done under the licence; and*
>
> *b) an assessment of that statement by the Minister administering the Environment Effects Act 1978 has been submitted to the Minister; and*
>
> *c) An authority to commence work has been granted by the Chief Administrator with the approval of the Minister.*

Section 42(7) of the *Mineral Resources Development Act 1990* is, in fact, is consistent with clause 52.08 of the East Gippsland Planning Scheme.

In all of the circumstances, no further planning approval is required for the Benambra Mine.

John McMullan
7 May 2001

Department of Natural Resources and Environment

240 Victoria Parade
PO Box 500 East Melbourne
Victoria 3002 Australia
Telephone: (03) 9412 4011
Facsimile: (03) 9412 4803
DX 210099

Ref. MI03/0070/6

29 January 2001

Mr Kevin M Tomlinson
Managing Director
Austminex NL
Lower Ground Level
15 Queen Street
MELBOURNE VIC 3000

Dear Mr Tomlinson,

MINING LEASE 1865 - BENAMBRA

I write about our meeting held at noon today concerning the Department's attitude towards, and requirements for, Austminex's continuing involvement at Benambra.

The Department fully understands the reasons surrounding the recent proposed change in project directions. DNRE wants to encourage Austminex in its proposed $1 million exploration program for the key tenements (ML 1865 and MIN 4281) as we believe that new exploration success will lead to a larger mineral inventory with the likelihood of changed project economics. DNRE also believes it is imperative that the sound existing environmental management program covering the site and in particular, the tailings dam, be continued by DNRE and Austminex. I would be pleased if these themes could be conveyed to the Board of Austminex and included in any public announcements.

The *Mineral Resources Development Act* 1990 does have provision for a renewal application to be considered prior to the expiry date of a tenement. While Austminex is actively exploring tenements, such as ML 1865 and MIN 4279, the Department would not propose cancellation. I would be pleased to discuss this matter with you at a later date once the project status is clarified.

The following points and preliminary cost estimates were discussed at today's meeting and I seek your confirmation regarding Austminex's willingness (or otherwise) to agree to the following proposals.

1. **Austminex NL to continue the environmental monitoring program and pay for:**

• B5 site (Wilga-Downstream) continuous monitoring of E.C. and pH plus Cu, Zn and SO_4 each two months.	$8,400 p.a.
• B1 site (Tea Pot Creek Track-Upstream) continuous monitoring of E.C. and pH plus Cu, Zn and SO_4 each two months.	$600 p.a. (another party is paying majority of cost).
• At the tailings dam and V notch weir: pH, E.C., Cu, Zn, SO_4 each two months plus a weekly pH and E.C. measurement at site.	$300 p.a.
• Six monthly monitoring of groundwater conditions (level, Cu, Zn, pH, E.C., SO_4) at borehole sites MB3 and MB17 (suggest February and August)	$100 p.a.

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- Conduct road maintenance including culvert cleaning as requested by DNRE Forests. $5,000 p.a.
- Implement weed control measures as required by DNRE Forests. $3,000 p.a.

Subtotal $19,400 p.a.

- Report all findings to MPV.

2. At the tailings dam, Austminex NL to:

a) Pay for:
- Weekly water level monitoring including the toe of dam V notch weir (WCB8) (0.5 hr at site)
- Inspection of seeps
- Maintenance of dam and associated infrastructure including the skirt drains, as required by MPR (0.5 hr inspection twice weekly).
- Reinstatement of the bund around springs on south abutment and siphon discharge of spring water $3,000
- Reinstatement of the existing coffer dam bund, upstream south side; manage discharge (pump) $1,000 for fuel p.a.

b) Comply with the intent of MPV's tailings dam guidelines – currently being prepared.

c) Report findings to MPV.

d) Reimburse MPR costs for regulatory management $3,000 p.a. (estimate)

3. Austminex NL to write to DNRE seeking an extension to the work plan covering site activities.

Such activities to include the potential for redistribution of tailings.

4. Responsibility for site security to be with Austminex NL.

The Caretaker of the Benambra Mine site is to undertake the following tasks:

- A twice-weekly inspection of the plant site. Ensure that all buildings and plant are secure. Note any vandalism or theft and note these in the site log.
- A twice-weekly inspection of the tailings dam and notation of the water level in the site log.
- A twice-weekly inspection of the site security, including all locked gates. Note any breaches of site security in the site log.
- Measure of the pH and E.C. of the tailings dam water and V-notch water at least once weekly, and note the results in the site log.
- Be responsible for the entry of persons or groups authorised to enter the site by way of providing access. *(Discretion may be exercised with regard to contractors entering and leaving the site. The caretaker does not have to be present at all times, but should ensure that the site security is maintained during and after authorised persons have been to the site.)*
- Keep a log of all persons or groups entering the site, and the reason for entry, in the site log. *(Note: authorised persons only should enter the site. Unauthorised persons should be asked to depart the site immediately).*
- Report to the MPR Contact Officer by telephone at least once per week. The verbal report should contain all events listed in the site log for the week.

- Provide transport for Departmental Officers and other authorised persons requiring entry to the site from time to time. *(This may occur on occasion when visitors have flown to Benambra, and require transport for the time they are in the area.)*
- Other duties as directed by the Contact Officer.
- Cost estimate for site caretaker ($1,000 per month) and vehicle ($500 per month) totals $18,000 p.a.

MPR Contact Officer:
Terry McKinley
8/250 Victoria Parade
East Melbourne 3002

Email: terry.mckinley@nre.vic.gov.au
Ph: 9412 5133
Fax: 9412 5152

5. **Austminex NL to pay any direct cost for DNRE to secure an EPA Works Approval/Licence for Discharge of water from the tailings dam.**

- Licence Discharge—Monitoring and Management (estimate) $5,000
- Contingency funds for laboratory testing, chemical analysis, etc. (estimate) $5,000

6. **DNRE (MPR) to continue with management of regulatory responsibilities at the tailings dam in particular and tenements generally.**

7. **DNRE (MPR) to establish and chair a site management review committee to meet quarterly to oversee site environmental management and related matters.**

   Committee composition to comprise DNRE (MPR), DNRE Forests, EPA and Austminex/consultants.

8. **Technical Reporting**

   In addition to the monthly summary report to the Manager Minerals and Petroleum Tenements, Austminex to report all results of mineral exploration activity and mining feasibility studies to DNRE on an annual basis (similar to Schedule 16 reporting). Austminex to brief DNRE each six months on the status of work activities.

9. **Information Storage**

   Should Austminex not gain transfer of tenements, then all information held by the company is to be despatched (in consultation with GSV) to the DNRE Werribee Core Storage Facility at Austminex's expense.

10. **Tenement Management**

- Regarding EL 3915, I note your undertaking to fax me a copy of your earlier advice to Nick Brooke regarding surrender of the tenement.
- Concerning MIN 5117, I would be pleased if you would mention to Mr Alan Martin his need to justify the proposed renewal application (of Denehurst Limited currently held in abeyance). This is an urgent matter.
- For Exploration Licence Application 3980, I would be pleased to urgently receive a letter and an accompanying map that describes a reduced area for which renewal is sought. A response by the end of April 2001 would be greatly appreciated.
- For Exploration Licence Application 3984, I would be pleased if you would consider a slightly smaller area for renewal and would be pleased to receive your advice by the end of April 2001.

- For EL 3458, I note that this licence is pending renewal.
- For Mining Lease 1865, MIN 4279 and MIN 4281, I acknowledge that the present status of each tenement will be maintained.

I will be in Bendigo tomorrow and will be pleased to discuss these matters with you if required on 0417 389 499.

Yours sincerely,

GEORGE BUCKLAND
Manager Minerals and Petroleum Tenements

cc: Mr Nick Brooks, Partner, PricewaterhouseCoopers (Deed Administrator, Denebourt Limited)

Department of Natural Resources and Environment

240 Victoria Parade
PO Box 500 East Melbourne
Victoria 3002 Australia
Telephone: (03) 9412 4011
Facsimile: (03) 9412 4803
ABN 90 719 052 204
DX 210099

Ref: ML 1865
MI/03/0070/7

2 March 2001

FAXED 2.3.01

Mr Bruce J. Patterson
Company Secretary
Austminex NL
Lower Ground Level
15 Queen Street
MELBOURNE VIC 3000

Dear Mr Patterson

MINING LEASE 1865 – BENAMBRA

I write in relation to your letter of 27 February 2001 regarding Austminex NL's continuing involvement at Benambra. In particular, DNRE notes your agreement in principle with the Administrator of Denehurst Limited about extension of the Benambra Option Agreement for 18 months from 30 April 2001 to 31 October 2002. I confirm that the Department's position relating to ML 1865, MIN 4279 and MIN 4281 is unchanged in that the present status of each tenement will be maintained until 31 October 2002.

Regarding proposals in my letter of 29 January 2001 and your reply of 27 February 2001, I confirm as follows:

1. Agreed by Austminex and the Department. DNRE to be consulted about the choice of analytical laboratory used, including monitoring of sedimentation of the montane swamp below the tailings dam;

2(a). Agreed by Austminex and the Department;
2(b). Not agreed – but the Austminex position is understood by DNRE;
2(c). Agreed by Austminex and the Department;
2(d). Agreed by Austminex and the Department with a cap of $3,000 pa;

3. Agreed between Austminex and the Department about the need for a work plan variation approval for future site activities. However, Austminex's assessment of the tailings as a potential resource for the project to be completed by 31 December 2001 with agreement of Minerals and Petroleum Regulation;

4. Agreed between Austminex and the Department

5. Agreed between Austminex and the Department;

6. Agreed between Austminex and the Department;

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7. Agreed between Austminex and the Department;

8. Agreed between Austminex and the Department;

9. Agreed between Austminex and the Department;

10. DNRE has noted your comments and affirms it's position in relation to dot point 6 and will have further discussions on other tenement management matters shortly.

In terms of the transition from DNRE to Austminex NL site security and environmental management responsibility as detailed in our agreement, I propose that this occur from Sunday 1 April 2001 following an on-site meeting with representatives of Austminex. Such an on-site meeting is planned for 23 March 2001. Please contact Mr Terry McKinley on telephone 9412 5133 to arrange this meeting.

I seek your confirmation regarding points 1 and 3, and the 1 April 2001 transition date. Please contact me on telephone 9412 4774 should you have any queries in relation to this matter..

Yours sincerely

GEORGE BUCKLAND
Manager Minerals and Petroleum Tenements

Cc: Mr Nick Brooke, Partner, PricewaterhouseCoopers (Deed Administrator, Denehurst Limited)

Department of Natural Resources and Environment

31 AUG 2001

240 Victoria Parade
PO Box 500 East Melbourne
Victoria 3002 Australia
Telephone: (03) 9412 4011
Facsimile: (03) 9412 4803
ABN 90 719 052 204
DX 210099

28 August 2001

Mr Andrew McDougall
Project Manager
Austminex
P.O. Box 339
Collins Street West
Melbourne VIC 8007

Dear Andrew

RE: ML 1865 Variation to Work Plan

Officers of the Department have conducted a preliminary assessment of your work plan submission for the Benambra Mine and I now provide the attached comments for you to address prior to the re submission of the work plan. At this stage the Department has not received any comments from other affected government agencies on the variation to the work plan.

In particular, we are aware the EPA has concerns that the rehabilitation concept of "Organic Cover" as described in the work plan may not be in line with EPA policy for waste management. The Department will not approve a Rehabilitation Plan that does not comply with EPA policy.

The work plan appears to have the fundamental framework in place for approval. However, Austminex will have to address the attached comments and any concerns of the other agencies prior to any consideration for approval.

Should you have any queries, or require further information, please do not hesitate to contact Regional Mining Engineer Mike Mathews on 57611501.

Yours sincerely

ROBERT KING
Manager, Minerals and Petroleum Regulation

att.

Victoria
The Place To Be

Mineral Recovery Methods.

26. What vehicles are to be used to move ore from the portal to the crushing plant?
27. How will oversize material be dealt with at the primary crusher?
28. What parts of the treatment plant are being decommissioned and what is being added?
29. Elaborate on the method of storage and location of hazardous substances (section 5.9).

Tailings Management.

30. Section 6.3. How will the boundary between the borrow area and the present tailings be dealt with?
31. Section 6.5. How will a pipeline failure/breach be detected and what will be the capacity of the downhill bund? This will be dependent on the potential discharge rate of the breach and the time taken to respond/repair the breach.
32. Section 6.6. How will the floating platform be moved and how often is "regularly"?

TMF Closure and Rehabilitation Plan.

33. Why does the preferred closure option require some beaching?
34. For the fully and partially de-watered options why is there a possible problem with maintaining a minimum water depth over the tailings, but not with the naturally regenerating wetland?
35. The final work plan will need to address in detail potential seepage through the dam wall and methods to contain it.
36. Has a definite source for the organic material been identified and what quantities are required?
37. How will the organic layer be distributed over the tailings and how will it be ensured that it sinks?
38. Provide expected timing for rehabilitation of the TMF from start of the rehabilitation to completion.
39. What monitoring and surveillance program will be in place during and after the rehabilitation works.
40. Discuss the lime dosing procedure.
41. Provide detail on the spillway construction.
42. How long will it take to achieve a naturally regenerating wetland.
43. Expand on the option to pump tailings from the TMF to underground. It would appear that to use these tailings as fill would enable a much greater water cover over the TMF.
44. Is the long-term integrity of the dam wall compromised by revegetation of it.
45. A trial of the "Organic Cover" rehabilitation may be required at the site to demonstrate the method.

Environmental Management Program.

46. The EMP needs to provide much more detail on all those areas that have been outlined.
47. The EMP should also include responsibilities and training.

48. Attention is also drawn to the requirements of the MRD Act 1990, in particular:

- 2.35 Contingency plans
- 2.50 Working practices manual
- 2.55 Ventilation surveys
- 2.71 Locking out of machinery
- 2.90 Equipment brakes
- 2.91 Off highway brakes
- Part 3 Open Cut operations
- Part 4 Underground Work
- Part 6 Electrical.

Andrew McDougall

From: Andrew McDougall

Sent: Thursday, 18 October 2001 6:13 PM

To: 'david.horsman@epa.vic.gov.au'

Subject: Re Benambra Mine Closure Strategy

Dear David,

I have tried to call you by phone, but realise that you are a busy person.

I spoke to Garry Kay earlier in the week regarding progress in the consideration of the closure strategy for the Benambra Mine tailings management facility.

He said that he had almost completed his report for presentation to David Mackenzie, who may wish to make changes or additions, prior to it being forwarded to you for your consideration and before sending EPA's position and conditions onto the DNRE.

In talking to Garry, I formed the impression that Garry is taking a conservative view on the requirements for encapsulation and the need for separation of the tailings from the water cover. He still seems to be very concerned about the concept of having waste material in a water course.

We are seriously concerned that insufficient account has been taken of the significant body of evidence, which indicates that a direct water cover is the best form of encapsulation of the tailings in areas which can ensure the maintenance of a water cover and that the method favoured by the DNRE and I suspect also by Garry, could have a greater potential for the generation of acid drainage, in the short and long term. A water cover provides the additional benefit of leaving the tailings undisturbed and not exposed to a combination of oxygen and water. Besides the method currently favoured by the DNRE having the potential to form acid, it will require large amounts of borrow material, which in itself will cause a significant impact to the environment, elsewhere, on or off the mine site.

If unrealistic, potentially environmentally threatening and expensive methods of closure are proposed, then this will provide a significant disincentive for making a decision to reopen the mine. Ultimately this is likely to leave the Government with the responsibility for the closure and the environmental outcomes arising from the closure method.

From our meeting with you we gained the impression that you had an enlightened approach and were prepared to consider alternatives. We respectfully ask that you use your judgement and influence to endorse the water cover method of closure.

Yours Sincerely

Andrew McDougall

Project Manager

LONG TERM CONTAINMENT OF POTENTIALLY ACID PRODUCING SULPHIDE TAILINGS

The Canadian Mining Industry and the governments of Canada have for many years recognized the effects on the environment from uncontrolled acid forming waste materials. The MINE ENVIRONMENT NEUTRAL DRAINAGE (MEND) PROGRAMME was established in the mid 1980s to investigate and establish the best methods for prevention, remediation and containment of acid drainage. Many studies have been completed and reports written since the formation of MEND which are recognized as providing the most comprehensive information on acid drainage and its management.

A six volume manual detailing the various aspects and recommendations arising from the research programme has been prepared. Volume 4 specifically addresses the PREVENTION AND CONTROL of acid drainage.

Where appropriate topographic and climatic conditions exist, the MEND recommendations are that the application of a water cover, over the potentially acid forming waste materials, provides the most effective barrier in the prevention of oxygen reaching the waste material.

Quotations from the MEND manual Volume 4 which support the use of a water cover are provided below:

- Preface – Prevention, page xiv: "In Canada, the use of water covers and underwater disposal are being confirmed as the preferred prevention technology for unoxidised sulphide containing wastes" and "Underwater disposal of mine wastes (tailings and waste rock) in man made lakes is presently an option favoured by the mining industry to prevent the formation of acidic drainage"
- New Ideas – Research Strategy, Preface page xviii: "In 1992, a task force was formed to solicit and nurture innovative new ideas. An additional goal was to encourage researchers from outside the general area of mining environment to becoming involved in acid drainage research. The resulting technology would need to be reliable, inexpensive, permanent and widely applicable."
- Project Approvals – Preface page xx: "In large part as a result of MEND, it was shown that new mines are able to acquire operating permits faster and more efficiently than before since there are now accepted acid drainage prevention techniques. As an example, Louvicourt mine in northwest Quebec adopted MEND subaqueous (**lake closure, currently under 1m of water cover, but trialing 0.3m cover, with no organic cover and situated in a natural watercourse**) tailings disposal technology and has been able to progress from the exploration phase to an operating mine within 5 years"
- Risk Assessment and Water Cover – Water Covers page 4-5: "Water covers have been applied at many sites, but are not universally applicable. Related issues such as the ability to maintain a water cover over the long term, the integrity of the containment structures, locality and site specific potential risks due to seismic events, severe storm events, etc. can negate

the use of this technology. However, under suitable conditions, the present state of knowledge is sufficient to allow for the responsible design, operation and closure of waste management facilities using water covers for both fresh and oxidized tailings and wastes" For Benambra the URS Study has addressed and demonstrated that the related issues of water cover retention, wall stability and lack of seismic events and mitigation against storm events are assured. Therefore for Benambra responsible design is the use of a water cover for the encapsulation of the tailings.

- Saturated Tailings – page 4-5: If a water cover is not used, but there is an attempt made to retain saturated tailings under a solid stratified cover MEND says – "The use of an elevated water table by itself does not prevent acid generation, as there may be zones of near surface exposed and drained tailings that remain available for oxidation." This is of significant concern and a major reason for Austminex advocating a water cover for Benambra.
- Dry Covers – page 4-6: Mend extensively researched the use of dry covers (including multi layer and geomembranes). "Supplying suitable and sufficient quantities of cover materials can be challenging and cost prohibitive at some sites."
- Water Covers – page 4-12: "Research has demonstrated that the oxidation of sulphides in mine tailings and waste rock is inhibited by placing the mine wastes under a water cover. The suitability of this method is subject to site specific factors and as such is not universally applicable. However, for sites where water covers can be used" (as demonstrated for Benambra), "the method offers one of the best solutions for preventing sulphide oxidation and acid generation over the longer term."
- Water Cover – page 4-16: "The present state of knowledge is sufficient for the responsible design, operation and closure of a waste management facility with a water cover."
- <u>Storage Under a Water Cover – page 4-18: "…storage under permanent water cover is perhaps the single most effective measure that may be taken to inhibit acid generation from sulphidic tailings.</u>"
- The Effect of Natural Sediments – page 4-23: "Time itself is an effective component in allowing the establishment of a physical barrier, which prevents the release of metals to the overlying lake waters. The accumulation of a veneer of natural sediments (A few centimeters thick) effectively isolates the tailings. Subaqueous disposal is at worst a relatively short term risk that decreases with time to yield a stable, passive but effectively final control system."
- Sufficiency of Water Cover Alone – page 4-60: "The range of oxygen fluxes predicted in modeling suggests that for most sites, a simple well maintained water cover, without additional measures, would be sufficient to suppress sulphide oxidation while maintaining the discharge from the water cover in compliance" with statutory requirements.
- Louvicourt Mine Experience – page 4-66: "It shows with little argument that reactivity of unaltered pyrite and other metal sulphides in situ under 30cm

of water is at worst minor on time scales of a few years. The body of evidence now available demonstrates compellingly that water covers are effective in preventing sulphide oxidation and acid generation."

If the impounded tailings have been exposed to oxygen and allowed to oxidize and acidify, a water cover can be supplemented by the placement of an organic layer directly onto the surface of the tailings and beneath the surface of the water. The organic layer serves to act as an oxygen consuming barrier and provides a reducing environment at the oxidized surface of the tailings. It has been amply demonstrated in the evidence above that for non oxidised sulphide tailings the additional organic layer is unnecessary. **The Benambra tailings are in pristine, non oxidized condition and as such only require a water cover for encapsulation. The proposal to incorporate an organic cover at Benambra is not technically necessary or warranted but has been proposed as a belts and braces solution**.

FILE NOTE

CONVERSATION WITH GARRY KAY, EPA, 15TH OCTOBER 2001

TMF CLOSURE STRATEGY

I spoke with Garry Kay this morning to see if he or the EPA had finalised their consideration of our closure strategy and whether they had developed a position.

The following notes are made from the conversation:

- Garry is just finalising his response which is intended to go to the NRE. Garry expects to pass on his response to his superior, David Mackenzie, tomorrow morning. David may or may not make changes. The response letter is expected to go out under the signature of Dave Horsman, Director Operations Directorate. Dave can change Garry's recommendations if he chooses.
- Garry accepts that a closure strategy based on wet closure in place of the previously approved dry cover is appropriate.
- Garry is still concerned about the lack of a barrier between the water and the tailings.
- Garry is still concerned about how the forest waste/organic cover will work. He acknowledges that verbally it has been stated that the forest waste/organic will be stored within the catchment zone of the TMF prior to placement over the tailings but that this verbal statement hasn't been included in the Work Plan Variation document. He is concerned about how effective the organic material will be if it dries out prior to placement.
- Garry is still concerned about the potential for the tailings to be scoured and disturbed during high runoff water inflow events. He acknowledges that provision has been made in the design to prevent scouring, but remains concerned about the effectiveness of the provisions in the long term. He suggested that another form of barrier is required – clay? I said that clay is scarce, would involve significant environmental disturbance to win it and poses construction issues in placing it over the tailings. He seemed quite vague about forms of cover and admitted that he and others in the EPA do not have construction experience.
- Garry says that the current water quality is not suitable for discharge to the environment due to elevated zinc and TDS levels. There will need to be provision made for water treatment until the water quality consistently meets the discharge criteria. The Work Plan Variation document makes no provision for water treatment.

- Garry says that he is concerned that the previous operators of the mine stored other waste materials; solids and chemicals in the TMF. This will not be allowed in the future.
- I said to Garry that if the mine reopening didn't proceed wasn't he concerned that the NRE closure strategy could have some construction problems and there could be a chance that the tailings could turn acid in the short and long term. He said that the tailings would need to be smoothed and the water level reduced and that the consultants advising the NRE had proposed the solid cover as being a feasible solution.
- I said the NRE's consultants had also indicated that a straight water cover over the tailings was an effective means of closure. Garry said that this was not an acceptable strategy for the NRE and EPA. When asked why, Garry said that because there is waste material in a water course.
- I asked Garry about the fact that there is evidence, with numerous examples, of tailings being stored under a direct water cover as the long term closure strategy. He said that these examples are overseas and that he didn't have all the details on water flows etc. I said that Mt Lyell is an example where direct water cover has and in the future will provide an acceptable final closure strategy. He said that Doug Sceney has some concerns and that some parts of the Mt Lyell facility worked well, whilst other parts didn't. Garry didn't know the specific details.
- I said to Garry that the Benambra facility was designed to overflow at times of high run off when the waterways have high flow. I said that even if there were slightly elevated zinc and TDS levels in the TMF that the effects of these would be lost in the total flow. He said his concern about this was that the streams had short sharp flows and the inference was that the streams could end up having diminished flows whilst the water was still discharging from the TMF and I assume Garry means that any elevated metal or TDS levels in the TMF water would have an impact on the stream water and the downstream environment. He said that the Mt. Lyell environment had higher flows which I infer he means have a greater capacity to flush the system and leave clean water discharging to the streams and rivers.
- Garry said there will need to be frequent monitoring of the TMF in the lead up to closure and for an extended period after closure.
- In summary Garry inferred that our closure strategy in its present form will not be acceptable.

Andrew McDougall

FILE NOTE

CONVERSATION WITH GARRY KAY (EPA)

AND GEOFF OXLEY (DNRE BENALLA), 9TH NOVEMBER 2001

TMF CLOSURE STRATEGY

GARRY KAY

- He has prepared a letter of recommendation regarding his preferred Tailings Management Facility (TMF) closure strategy for David Mackenzie and Dave Horsman to consider. David Mackenzie has still to consider the recommendation and has yet to discuss it with Dave Horsman.
- Garry is recommending some form of physical barrier over the tailings in addition to the water cover. Garry was not forthcoming regarding the form of this barrier.
- He understands that the NRE are supposed to be conducting a risk assessment study of the EPA proposal versus the Austminex proposal plus the previously advanced options. He believes that this process has not progressed very far and that Geoff Oxley has been assigned the responsibility in place of Terry McKinley.
- I asked Garry to keep me informed of the status of the deliberations of David Mackenzie and Dave Horsman.

GEOFF OXLEY

- Geoff confirmed that he has been asked to conduct a risk assessment study of the various closure strategies and to report his findings to the EPA. To date there has been little progress and has not been given priority since the announcement that the project is on hold for the present.
- Geoff says the risk assessment study could take some time. I said that I was trying to finalise the Mine Reopening Feasibility Study and that the TMF closure strategy is an integral part of the study and is significant in the decision process for the project to proceed in the future. I indicated that I had to present the finalised study by the end of November. He said he would try to get his study completed in time, however I don't hold out much hope.
- Geoff says that there are many people in both the DNRE and EPA who hold varying views as to what is an acceptable closure strategy for the TMF and that it will take some co-ordination to arrive at a consensus position, which may or may not be acceptable to Austminex.

Andrew McDougall

Austminex N.L. Benambra

FILE NOTE

URS (JEFF BAZELMANS) – EPA DISCUSSION 23RD NOVEMBER 2001

TMF CLOSURE STRATEGY

Dr Jeff Bazelmans, Senior Principal at URS is a former 3IC at the Victorian EPA.

At the suggestion of Andrew McDougall, URS agreed to have Jeff meet informally with Garry Kay and David Mackenzie of the EPA to establish the status of their considerations and any concerns they may have in respect to the Austminex TMF closure strategy proposal.

Jeff was acquainted with the URS TMF closure strategy report, prior to the meeting.

The discussions were held off the record.

A subsequent meeting was held between Dr Bazelmans, Steve Newman (URS) and Andrew McDougall to record the outcomes of the URS/EPA discussion.
The outcomes are as follows:

- Jeff held discussions with Garry Kay and David Mackenzie for over 2 hours.
- The considerations of Garry Kay and David Mackenzie were at an advanced stage and had reached the point that a draft letter, outlining EPA's requirements, had been prepared for the consideration of Dave Horsman (EPA Operations Director).
- It was clear that Garry Kay had spent considerable time analysing the closure strategy.
- David Mackenzie is very knowledgeable of Benambra, having been associated with the mine site for over 20 years.
- Jeff was permitted to read the draft letter. Following suggestions made by Jeff the letter was withdrawn for redrafting, to incorporate some of the outcomes resulting from the discussions.
- The EPA have accepted that a wet cover is the preferred solution for tailings closure, in place of the previously accepted dry closure strategy.
- The EPA is happy with the concept of the formation of an artificial (engineered) wetland.
- The EPA don't want the water removed from above the tailings during the cover application and tailings dam rehabilitation.
- The EPA now accept that the issue of the tailings dam becoming acid following closure in 1996, was due to the addition of acid to the process water by Denehust, for operational purposes, and that the acidification did not result from the tailings themselves forming acid whilst submerged by water and protected from contact with oxygen.

- The EPA are now classifying the tailings as a resource and not waste material. This appears to overcome the policy concerns of having waste in a water course.
- The EPA will not accept a straight water cover over the tailings as being sufficient protection. They are concerned about the possible scouring and mobilisation of the tailings during storm events and also the need for an adequate water cover to remain at all times.
- The EPA do not accept that the organic layer, as proposed by URS, is acceptable on the basis of its lack of resistance to scouring, the possibility of it not providing a porous medium through which water can pass down to the tailings and the possibility of importing diseases which may attack the surrounding forest. They require the placement of a crushed rock cover over the entire tailings surface in place of the organics. They see the rock providing a scour resistant cover, will be porous and will ensure that the tailings remain wet even through protracted dry periods.
- The EPA believe that waste rock can be quarried from the rear of the tailings impoundment, which will provide additional storage capacity. The thickness and nature of the rock cover and how it can be applied was not specified or discussed. This will be the subject of further discussion and study.
- The EPA accept that in the longer term, natural run off sediments and leaf litter will accumulate in the tailings dam and will provide an additional barrier above the tailings.
- The EPA did not express any concern about the long term effectiveness of a spill way to control the water level and for the protection of the wall.
- The Government is keen to have a private enterprise organization take over the responsibility of the tailings facility and its closure.
- The EPA is under pressure from the DNRE, for the EPA to make a decision as to an acceptable closure strategy and to provide approval. The DNRE will not make a decision in its own right. There is a political power play going on with the DNRE being represented by a minister in Cabinet, where as the EPA is represented by a junior minister. NSW and Victoria are the only states with a separate EPA. In the other states there have been moves to combine the functions under an umbrella resources department. This may be starting to happen in Victoria.
- From an earlier meeting between the DNRE and EPA it was agreed that the DNRE would conduct a Benambra Project risk assessment, in parallel with the EPA specific consideration of the TMF closure strategy. The EPA says that they understand that the DNRE have not commenced the risk assessment and they doubt that it will be done. Geoff Oxley of DNRE Benalla was assigned the task of preparing the risk assessment.
- Jeff doesn’t believe that there will be anything gained by Austminex mounting a formal appeal to VCAT against any rulings made by the EPA, as the EPA will then take a harder position, which is likely to result in the tribunal compromising back to the position that the EPA has currently

reached. This would be a costly and time consuming process with little if any beneficial effect.

- The approvals process from here is as follows: The Garry Kay/David Mackenzie letter will be redrafted (softened) by 30th November → draft letter submitted to Dave Horsman (EPA) for his executive consideration and possible modification (1 to 2 weeks) → EPA final draft letter sent to DNRE for consideration (time with DNRE unknown) → letter returned to EPA with or without comments → EPA considers any possible comments and prepares letter stating its requirements → EPA letter sent out under the signature of Dave Horsman to DNRE → DNRE incorporate the letter into its requirements and writes to Austminex advising it of the licence conditions. At best this process will take 4 weeks, but could well take longer.
- Austminex to establish the cost of complying with the requirements and establish/agree with the DNRE the quantum of the environmental bond. Austminex in establishing the cost of compliance may have to further engage the services of URS. This should amount to no more than $10,000. URS have reviewed a previous cost estimate for the closure of the TMF. Based on what they currently believe the EPA will require, in the way of a rock cover, it appears that the existing allowance for the application of the organic cover may be sufficient for the substitute rock cover. That is the previous cost estimate for the formation of a wetland may remain the same. This will have to be formally reviewed, designed and costed once the licence conditions are known.
- As the EPA don't appear to have provided detailed requirements and specifications for the rock cover, URS believes that after receipt of the final letter, via the DNRE, that there may be opportunities for proposing to the EPA that there be a perimeter rock cover applied to the margins of the dam and not over the entire surface. This would cut down on the quantity of crushed rock required and make for easier placement of the cover.

On another matter the EPA says that it cannot permit any more water discharges from the tailings dam (as has occurred on 2 occasions since cessation of operations, when the DNRE have syphoned off water to prevent overtopping of the non spillway wall), under the Section 30A Emergency Discharges provisions. The DNRE (Terry McKinley) is supposed to be preparing a submission to the EPA for a works approval to discharge water on a needs basis and when conditions permit. It would appear that the EPA have not received an application from the DNRE. This is likely to become an issue as the water level in the dam rises and there is a need for discharge, to ensure that wall integrity remains through having sufficient freeboard. This is an issue for the DNRE and EPA to resolve, as the DNRE currently has responsibility for the dam. Austminex must not get caught up in this issue or accept any premature responsibility for the dam.

Andrew McDougall

29th November 2001

Austminex N.L. Benambra

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

1 March 2001

Memorandum: **Kevin Tomlinson**
Austminex NL
9620 2147

From: **Graham Robertson**

Bonds

Further to our telephone discussion, it appears that rehabilitation bonds are provided for by S80 of the MRD Act, which requires a licensee to "enter into a rehabilitation bond for an amount determined by the Minister".

The term "bond" is not defined, in the MRD Act, and the Regulations do not refer to the subject, and it appears that S124 of the Act, which provides for the making of Regulations does not allow any Regulations concerning bonds to be made.

Therefore, it seems that the manner and form of the required bond is entirely at the discretion of the Minister and there are no Regulations or Schedules that provide any legislative guidelines or parameters regarding what form the bond may take.

It follows that the DNRE instructions re bonds of 11 March 1999, and the "Rehabilitation Bond" (copies follow) are creations of DNRE, based on policy and simplicity of administration. As you may be aware cash bonds were previously allowed, but were apparently regarded as difficult to administer, resulting in the introduction of the present policy.

As the type and level of the bond on any licence is at the discretion of the Minister, it may be possible in certain particular cases, to waive the policy requirement that a bond must be in the form of a bank guarantee.

For this to occur, DNRE would obviously need to be totally convinced that the alternative form of proposed bond is able to fulfil all of the requirements of Part 7 of the MRD Act (Rehabilitation) as well as the relevant provisions of DNRE's policy on bonds of 11 March 1999.

G.R.

Graham Robertson

Exploration and mining title consultants for the
management of mining interests throughout Australia.
A division of Graham Robertson & Associates Pty Ltd ACN 006 279 643
3 Browning Drive, Templestowe Victoria 3106

to discharge water on a needs basis and when conditions permit. It would appear that the EPA have not received an application from the DNRE. This is likely to become an issue as the water level in the dam rises and there is a need for discharge, to ensure that wall integrity remains through having sufficient freeboard. This is an issue for the DNRE and EPA to resolve, as the DNRE currently has responsibility for the dam. Austminex must not get caught up in this issue or accept any premature responsibility for the dam.

Andrew McDougall

29th November 2001

Natural Resources and Environment

AGRICULTURE • RESOURCES • CONSERVATION • LAND MANAGEMENT

***MINERAL RESOURCES DEVELOPMENT ACT* 1990**
***EXTRACTIVE INDUSTRIES DEVELOPMENT ACT* 1995**

11 March 1999

BONDS ON TENEMENTS & AUTHORITIES

The *Mineral Resources Development* Act 1990 (MRD Act 1990) and the *Extractive Industries Development* Act (EID Act) 1995 require a licensee or holder of a Work Authority to lodge a bond with this Department as a surety for compliance with the conditions of the tenement or authority relating to rehabilitation and stabilisation of the land disturbed by mining, exploration or quarrying operations and the safety of the public.

The **ONLY** form of bond acceptable to the Department is a Bank Guarantee. This Bank Guarantee must:

(a) be a Bank Guarantee from a recognised bank (according to the current list of Individual Authorised Banks in Australia - Australian Prudential Regulation Authority, Reserve Bank of Australia), Building Society or Credit Union (according to the current list of Registered Corporations under the *Financial Corporations Act* 1974, Reserve Bank of Australia)

(b) be in the name of the Minister for Energy and Resources; X

(c) state the licence or work authority number to which it applies;

(d) if more than one (1) licence or work authority is in operation, each must have a separate bond;

(e) be on Bank, Building Society or Credit Union letterhead or have a recognised stamp to indicate authenticity;

(f) not have an expiry date;

(g) be an original document (ie. not a copy)

(h) be *signed* and *dated* by an authorised officer of the financial institution.

***NOTE*: In all cases where a Bond is lodged by a party other than the licensee, this party should be aware that the release of the Bond will be forwarded directly to the licensee.**

REHABILITATION BOND

MINERAL RESOURCES DEVELOPMENT ACT 1990

I, ...
(licence holder

of ..... being a
(address)

*person/company to whom .....
(Title name and number)

has been granted/transferred* and .....
(name of surety corporation)

.....
(address of surety corporation)

jointly and severally bind ourselves to for the payment to the **Minister for Energy & Resources** for the State of Victoria for the sum of $ .....

The condition of this obligation is such that the said licence holder shall duly observe all conditions, limitations and restrictions to which the licence is granted under the provisions of the *Mineral Resources Development Act 1990*. If the licence holder does not observe all conditions, limitations and restrictions to which the licence is granted then the bond may be forfeited to the **Minister for Energy & Resources** to effect reclamation of the site. The bond cannot be released either in part or in whole without the written authority of the Department of Natural Resources & Environment and

.....
(name and address of surety corporation)

undertake to pay the Department of Natural Resources & Environment on demand, of the whole or any part of the bond.

Signed, sealed and delivered by the abovenamed .....
(licence holder)...

in the

presence of .....
(Print name of witness) (Signature of witness)

.....
(address of witness)

The Common Seal of .....
(surety corporation)

was hereunto affixed in the presence of

.....
(Print name of person authorised to sign on behalf of the surety corporation)

.....
(Signature of person authorised to sign on behalf of the surety corporation)

Sealed with our seals this ..... day of ..... 19.....

** Strike out whichever is not applicable*

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Department of Natural Resources and Environment

14 JUN 2001

240 Victoria Parade
PO Box 500 East Melbourne
Victoria 3002 Australia
Telephone: (03) 9412 4011
Facsimile: (03) 9412 4803
ABN 90 719 052 204
DX 210099

ML1865
MI/03/0070-7

6 June 2001

Mr Kevin Tomlinson
Managing Director
Austminex NL
PO Box 339
Collins Street West
MELBOURNE VIC 8007

Dear Mr Tomlinson

BENAMBRA TENEMENTS

I write in reference to the status of the Benambra tenements (see attached map) and advise as to the requirements for the continuity and transfer of the tenements to Austminex NL:

- Mining Lease 1865 - expires 11 April 2004.
- Mining Licence 4279 - expires 11 April 2004
- Mining Licence 4281 - expires 11 April 2004
- Exploration Licence 3458 - tenement in pendency since 10 August 1997. Requires two renewal applications to allow the tenement to be renewed until 10 August 2002.
- Exploration Licence Applications 3980 & 3984 - Written consent in accordance with Section 15(1a)(d) of the *Mineral Resources Development Act* 1990 is required from the Administrator of Deneburst Ltd to allow a subsequent application to be made in the name of Austminex NL. The Administrator would be required to withdraw its application and the subsequent application from Austminex NL would be given priority.

Steps leading to a commencement of work are:

1. Administrator (Deneburst) and Austminex lodge completed "Deeds of Transfer".
2. Rehabilitation bonds on tenements lodged by Austminex - particularly the $1.5 million rehabilitation bond on Mining Lease 1865.
3. Department processes transfer applications.
4. Following registration of transfers, Department can then accept lodgment of a Work Plan (including rehabilitation plan) Variation in the name of licensee/lessee Austminex.
5. Department commences final approval process for the Work Plan Variation request.
6. Approved Work Plan Variation is registered.
7. Site works can commence.

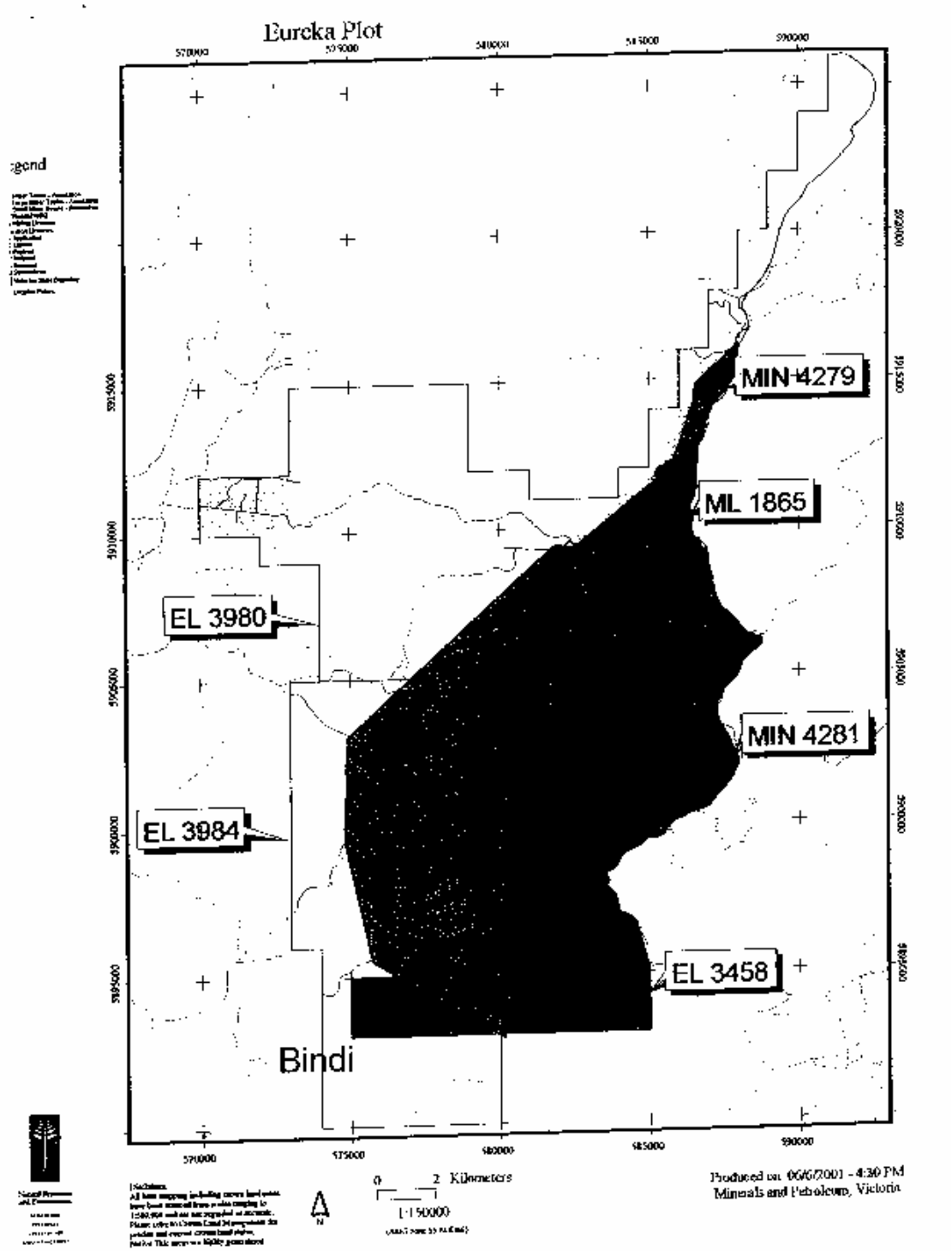
I have attached for your information a copy of the Departments "Deed of Transfer" proforma and should you require any further information please do not hesitate to contact Mr Tony Monardo, Team Leader Northern Region on (03) 9412 5080.

Yours sincerely

GEORGE BUCKLAND
Manager Minerals & Petroleum Tenements

Victoria
The Place To Be

CC: Mr Nick Brooke - Deneburst Administrator
Mr Graham Robertson - Authorised Agent



Austminex N.L. Benambra

	Graham Robertson
	Australian Tenement Management
	03 9841 9361
	Andrew McDougall
	Reductions in ELA 3980 and 3984 Areas
	4
	26 April 2001

Graham,

As you are aware the DNRE have asked Austminex to review the extent of ELAs 3980 and 3984. The DNRE are seeking to have the existing ELAs reduced in area and to have this done by 30th April.

Austminex has conducted an extensive review of the exiting exploration data applicable to these ELAs and has come to the conclusion that the shaded areas, on the attached tenement map, can be relinquished due to them being of low prospectivity.

We are currently reluctant to recommend further relinquishment as we believe the retained areas still contain good exploration potential.

Letters have to be written to the Denehurst Administrator and to the DNRE. Kevin Tomlinson is currently away and will be back in this office on Monday 30th April.

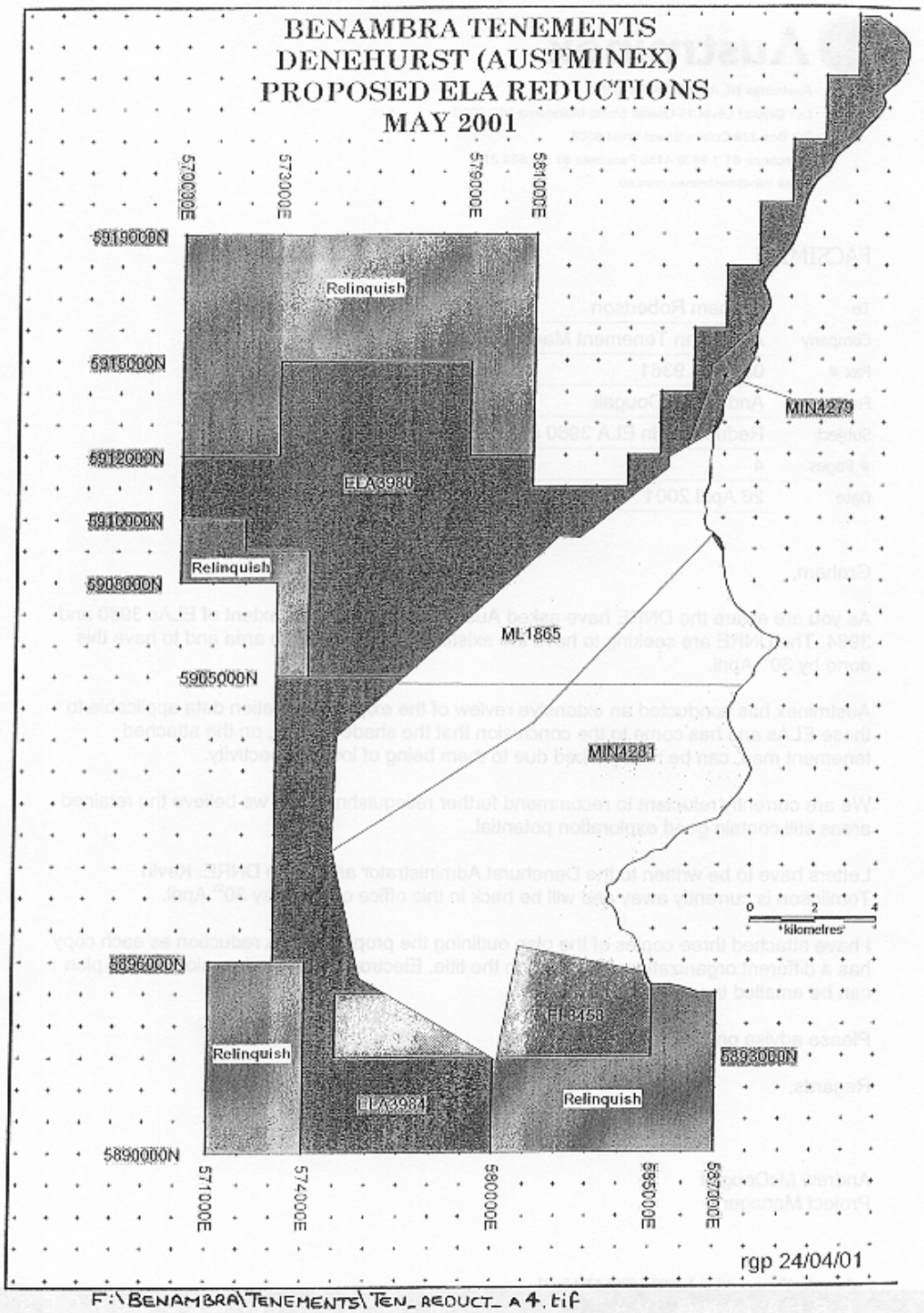
I have attached three copies of the plan outlining the proposed ELA reduction as each copy has a different organization referred to in the title. Electronic, coloured versions of the plan can be emailed to you if required.

Please advise on the process from here.

Austminex N.L. Benambra

Regards,

Andrew McDougall
Project Manager



Austminex N.L. Benambra

	Graham Robertson
	Australian Tenement Management
	03 9841 9361
	Andrew McDougall
	Reductions in ELA 3980 and 3984 Areas
	1
	27 April 2001

Graham,

Further to your fax of 27th April 2001 regarding ELAs 3980 and 3984, I confirm that Austminex wish to proceed with the relinquishment of the shaded areas as depicted in the location plan provided to you yesterday.

Please proceed immediately with the process required to effect the reductions in area.

Regards,

Andrew McDougall

Austminex N.L. Benambra

Project Manager

Austminex N.L. Benambra

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

30 April 2001

Memorandum: **Andrew McDougall**
Austminex NL
9620 2147

From: **Graham Robertson**

ELAs 3980 and 3984

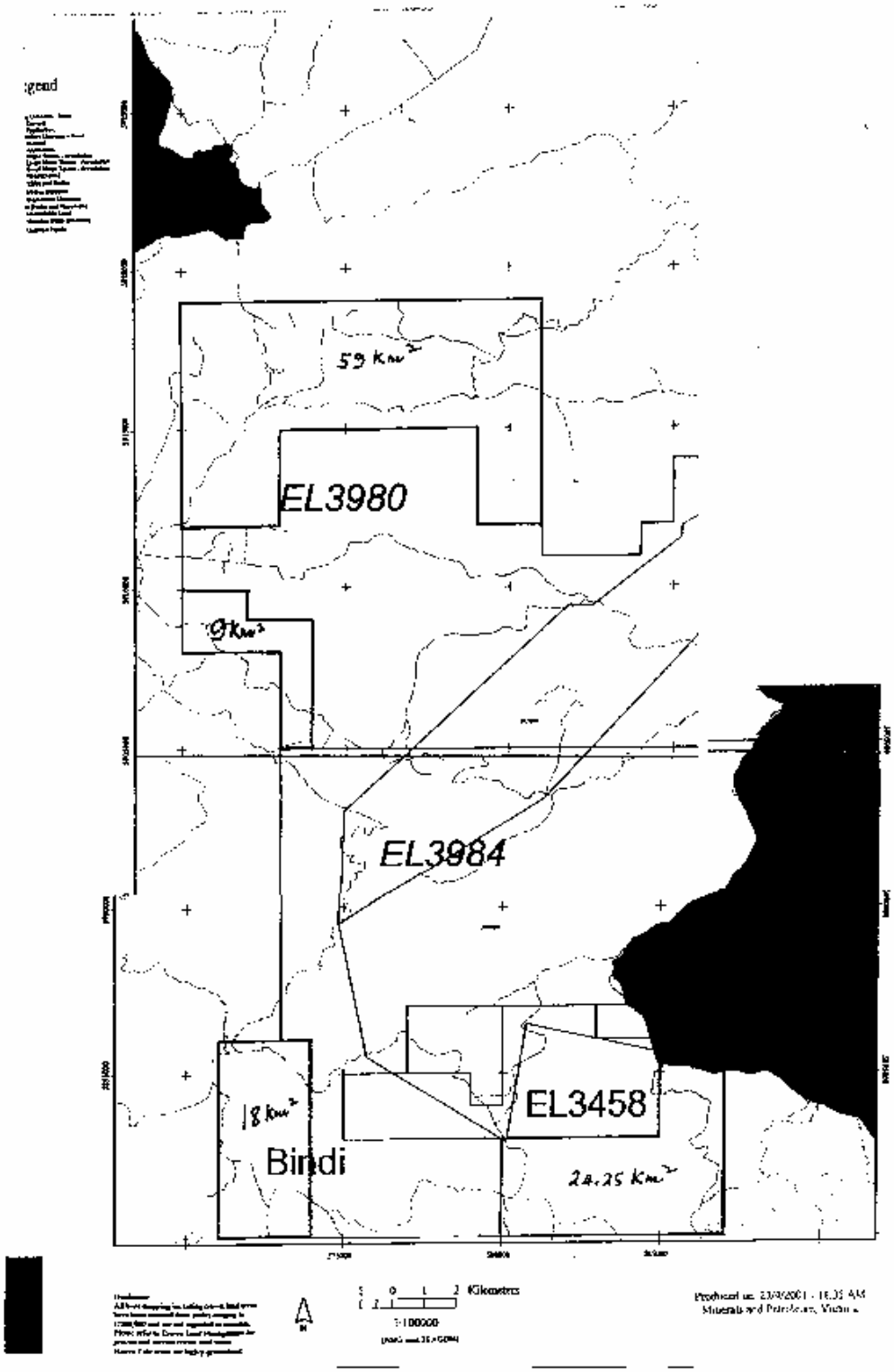
Further to your instructions, I intend to request an amendment of the areas of ELAs 3980 and 3984 to approximately 136 km2 and 135 km2 respectively, by the surrender of 68 km2 from ELA 3980 and 42.25 km2 (approximately) from ELA 3984, as shown on the following diagrams.

These reductions in area will take effect immediately, and will therefore be available for EL/MIN application by other parties, should they wish to do so.

G.R.

Graham Robertson

Exploration and mining title consultants for the management of mining tenements throughout Australia.
A division of Graham Robertson & Associates Pty Ltd ACN 006 279 643
3 Browning Drive, Templestowe Victoria 3106



Austminex N.L. Benambra

Austminex

Austminex NL ABN 56 005 470 799
Level 6 150 Queen Street Melbourne VIC 3000
PO Box 339 Collins Street West 8007
Phone +61 (0)3 9670 5111 Fax +61 (0)3 9670 8111
email info@austminex.com.au
www.austminex.com.au

30 July 2001

The Secretary
Minerals & Petroleum Division
Department of Natural Resources and Environment
240 Victoria Parade
East Melbourne VIC 3002

Dear Sir

Applications for Exploration Licences

Attached is an application for an Exploration Licences to cover the areas of the former Exploration Licences 3980 and 3984.

The fee of $1073 accompanies this application.

Yours sincerely

Graham Robertson
Authorised Agent – Austminex N.L.

F:\Benambra\Tenements\Exp. Licences 3980 - 3984 Appl. Letter – ATM 300701.doc

CREATING VALUE FROM MINERAL RESOURCES

30. JUL. 2001 12:23

PRICEWATERHOUSECOOPERS

PricewaterhouseCoopers
215 Spring Street
GPO Box 1331L
MELBOURNE VIC 3000
DX 77 Melbourne
Australia
Telephone (03) 8603 1000
Facsimile (03) 8603 6044
Website: www.pwcscovery.com

Fax cover sheet

To:	Tony Monardo
Company:	Department of Natural Resources and Environment
Addressee fax No.:	9412 5150
cc:	Andrew McDougall
	Austminex NL
Addressee fax No.:	9670 8111
From:	Nick Brooke
Return fax number:	8603 6044
Date:	30 July 2001
No. of pages:	1 (incl. this page)

If this fax is incomplete or illegible please telephone

Denehurst Limited (Administrators Appointed) ACN: 006 738 576 ABN: 83 006 738 576 (Denehurst)

Dear Tony

I refer to the interests of Denehurst in EL 3980 and EL 3984.

I advise that, as the Deed Administrator of Denehurst, it is appropriate for Austminex NL to lodge applications behind Denehurst for EL 3980 and EL 3984. Subsequently, Denehurst Limited will withdraw its applications on these two EL's.

Yours sincerely

Nicholas Brooke
Deed Administrator
Denehurst Limited

Department of Natural Resources and Environment

35 Sydney Road
PO Box 124 Benalla
Victoria 3672 Australia
Telephone: (03) 5761 1611
Facsimile: (03) 5761 1628
DX 214480

08 August 2001

Austminex NL
C/- Mr Graham Robertson
Australian Tenement Management
PO Box 3194, The Pines
EAST DONCASTER VIC 3109

BY REGISTERED MAIL

Dear Mr Robertson

EXPLORATION LICENCE APPLICATION 4599

The above application which was received on 30 July 2001, has been awarded priority over the area shown on the attached map. This area may be smaller than the original application area if some of the land was unavailable due to the higher priority of another application, the location of an existing licence, or the land being exempt by legislation. If you do not wish to proceed you may withdraw your application by writing to the Department.

As part of the application assessment, details of the applicant/s will be checked against Departmental records regarding any outstanding non-compliance issues relating to previous or currently held tenements. Failure to finalise such matters or to comply with requirements of section 15(6) of the *Mineral Resources Development Act* 1990 may result in your application being refused.

Advertise the application

You must advertise your licence application within 14 days of the receipt of this letter (*Mineral Resources Development Act* 1990 s.15(5)). The advertisements should be according to Schedule 7 (enclosed) and appear as follows:-

- In the Wednesday edition of a daily newspaper circulating generally in Victoria, e.g. The Herald-Sun, The Age, The Australian. There is no requirement for this advertisement to include a map.
- In a newspaper circulating in the locality of the application area. This advertisement must contain a map clearly identifying the area of application (include local road and place names). There is no requirement for this advertisement to be published on a Wednesday.

Copies of each advertisement must be lodged with this Department by **14 September 2001**. If copies of the advertisements are not received by this date, the Department may recommend that this application be refused.

Victoria
The Place To Be

Native title - options

As the application includes Crown land, we cannot grant the licence until you have met the "future act" provisions of the Native Title Act (Cth) (NTA); or alternatively all Crown land (other than roads and road reserves) is excised from the application. The options are:

Option 1: Excise all Crown land (except roads and road reserves which are not subject to Native Title)

Option 2: Retain the Crown land and comply with the Right to Negotiate provisions of the NTA

Option 3: Retain Crown land and reach an Indigenous Land Use Agreement under the NTA.

You must tell us which option you wish to pursue by *31 August 2001* or the application will lapse. This request for additional information is made under section 15(7) of the *Mineral Resources Development Act* 1990. Please address your reply to your Client Service Officer whose contact details are below.

Yours sincerely

GEORGE BUCKLAND
Manager Minerals & Petroleum Tenements

Contact:

Maree Halligan

Client Service Officer
Minerals and Petroleum Tenements
Ph: (03) 57611503
Fax: (03) 57611628
maree.halligan@nre.vic.g

Encl.:
Application Plot
Native Title Act 1993 Advice
Native title letter ILUA or RTN
Schedule 7

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

13 August 2001

Memorandum: **Andrew McDougall**
Austminex NL
9670 8111

From: **Graham Robertson**

EL 4599

DNRE has confirmed the priority of this application and we are now required to advertise in a metropolitan and local newspaper. I will attend to this next week. *The Age* will bill the company directly.

We must also deal with the Native Title situation, options 2 and 3 are the only choices (see p 2 of DNRE's letter).

I suggest at this stage we opt for an ILUA, which means that we will be required to deal directly with the NT claimants. If that does not work out, then we can opt for the RTN procedures, which require advertising and compulsory negotiation.

May I have your instructions re NT as soon as possible.

G.R.

Graham Robertson

13/8/2001

Andrew
I support GR's advice re opting for the ~~UEA~~ ILUA alternative at this time
BJPat...

Austminex

Austminex NL ABN 56 005 470 799
Level 8 150 Queen Street Melbourne VIC 3000
PO Box 339 Collins Street West 8007
Telephone 61 3 9670 5111 Facsimile 61 3 9670 8111
email info@austminex.com.au
www.austminex.com.au

FACSIMILE

To	Graham Robertson
Company	Australian Tenement Management
Fax #	9841 9361
From	Andrew McDougall
Subject	EL 4599
# Pages	1
Date	13 August 2001

Graham,

In response to your fax of today's date, please proceed as per your advice, of adopting the ILUA process in the first instance.

Andrew McDougall

F:\Benambra\Tenements\EL 4599 ATM fax 130801.doc

CREATING VALUE FROM MINERAL RESOURCES

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

3 August 2001

Memorandum: **Kevin Tomlinson**
Austminex NL
9670 8111

From: **Graham Robertson**

EL 3458 Bankala (Denehurst)

This EL was due to expire on 10 August 1998 and a further renewal application was lodged on 18 May 1998. A further renewal application was lodged on 16 March 2000 for a period of two years. These renewal applications also remain pending, and will now expire on 10 August 2001.

This is a pre-native title EL, and should be retained if it is of any interest.

If it is to be retained, we **must** lodge a renewal prior to cob on Friday 10 August 2001 (preferably before). We should also investigate the possibility of transferring the EL to Austminex from Denehurst.

The EL is adjacent to Austminex's new EL application 4599, lodged on 30 July 2001. Denehurst's ELAs 3980 and 3984 have been withdrawn.

May I have your instructions as soon as possible.

G. R.

Graham Robertson

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

7 August 2001

Memorandum: **Andrew McDougall**
Austminex NL
9670 8111

From: **Graham Robertson**

EL 3458

As instructed, I have lodged an application for the further renewal of this Deneliurst EL. We should arrange for the transfer of the title if it is required.

G.R.

Graham Robertson

Exploration and mining title consultants for the
management of mining interests throughout Australia.
A division of Graham Robertson & Associates Pty Ltd ACN 006 279 643
5 Browning Drive, Templestowe Victoria 3106

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

20 September 2001

Memorandum: **Andrew McDougall**
Kevin Tomlinson
Austminex NL
9670 8111

From: **Carolyn McSporran**
for Graham Robertson

EL 3458 Benambra

This EL has been renewed to expire 10 August 2002 with the expenditure commitment set at nil. A copy of the registration certificate follows for your records.

The previous renewal applications were also approved, each with a nil expenditure commitment.

Carolyn McSporran

Exploration and mining title consultants for the
management of mining tenements throughout Australia.
A division of Graham Robertson & Associates Pty Ltd ACN 006 279 643
3 Browning Drive, Templestowe Victoria 3106

MINERAL RESOURCES DEVELOPMENT ACT 1990

INSTRUMENT OF RENEWAL (SECTION 31)

LICENCE TYPE	Exploration Licence
LICENCE NUMBER	3458
DATE OF GRANT OF LICENCE	10 August 1993
NAME OF LICENSEE	Denehurst Ltd
ADDRESS OF LICENSEE	Level 8/250 Victoria Pde East Melbourne Victoria 3002
LOCATION	Omeo Map Sheet
CURRENT AREA	27 graticular sections
AREA RELINQUISHED/REFUSED	0 graticular sections
AREA REMAINING	27 graticular sections
LICENCE RENEWED TO EXPIRE	10 August 2002
STRATUM OF LAND	Not Applicable

NOTES

CONDITION 3.
The licensee must expend on the licensed area **a minimum of $0 per annum**, except where the Minister varies the expenditure requirement under the Act, or suspends the application of the expenditure requirement for a specified period.

This licence is not an approval pursuant to the *Environment Protection and Biodiversity Conservation* Act 1999 (Cth).

Signed by

DAVID LEA
Executive Director, Minerals and Petroleum
Delegate of the Minister.

Date: 4 SEP 2001

Date of Registration
10/09/2001
Time of Registration
2 59 am/pm
MINING REGISTRAR
MRDA 1990 (Section 69)

F11,401

Department of Natural Resources and Environment

240 Victoria Parade
PO Box 500 East Melbourne
Victoria 3002 Australia
Telephone: (03) 9412 4011
Facsimile: (03) 9412 4803
ABN 90 719 052 204
DX 210099

18 April 2001

Mr Graham Robertson
Denehurst Ltd (In Administration)
C/- Australian Tenement Management
PO Box 3194 The Pines
East Doncaster VIC 3109

Dear Mr Robertson

EXPLORATION LICENCE 3915

The Minister has consented to the surrender of the above licence. The instrument of Surrender is enclosed.

Please forward a final technical report by **16 May 2001** (Regulation 216 of the *Mineral Resources (Titles) Regulations* 1991). Please notify us if the last annual technical report should be taken as the final report.

The rehabilitation bond will be returned after we have received a satisfactory report from the environmental officer that all rehabilitation issues have been addressed.

Yours sincerely

Tony Monardo
Team Leader - Northern Region
Minerals & Petroleum Tenements

Tel: (03) 9412 5060
Fax: (03) 9412 5150
Email: Tony.Monardo@nre.vic.gov.au

Encl.
Instrument of Surrender .

Victoria
The Place To Be

R7

MINERAL RESOURCES DEVELOPMENT ACT 1990

NOTICE OF SURRENDER (SECTION 37)

LICENCE TYPE	EXPLORATION LICENCE
LICENCE NUMBER	3915
DATE SURRENDER REQUEST SUBMITTED	26 March 2001
NAME OF LICENSEE	Denahurst Ltd
ADDRESS OF LICENSEE	C/- Department of Natural Resources and Environment 8/250 Victoria Parade East Melbourne Victoria 3002
AREA	455 Graticular Sections
STRATUM OF LAND	Not Applicable

Signed by

GEORGE BUCKLAND
Manager, Minerals & Petroleum Tenements
Pursuant to Instrument of Delegation by the Minister dated 18 November 1999.

Date: 4/4/01

Date of Registration
04 / 04 / 2001
Time of Registration
5 : 00 am/pm
MINING REGISTRAR
MRDA 1990 (Section 69)

F 11209

Australian Tenement Management
Exploration and Mining Tenement Consultants

PO Box 3194
The Pines
Doncaster East 3109

Telephone: 03 9842 7694
Facsimile: 03 9841 9361
Mobile: 04 0805 1140

23 April 2001

Memorandum: **Kevin Tomlinson**
Austminex NL
9620 2147

From: **Graham Robertson**

EL 3915

The following has been received from DNRE. I will advise them that the last annual report should be regarded as the final report.

G. R.

Graham Robertson

Exploration and mining title consultants for the
management of mining tenements throughout Australia
A division of Graham Robertson & Associates Pty Ltd ACN 006 279 443
3 Browning Drive, Templestowe Victoria 3106

AUSTMINEX NL

BENAMBRA PROJECT

CAPITAL AND OPERATING COSTS

SUMMARY REPORT

CAPITAL COST AND FUNDING REQUIREMENTS-600,000 TPY

Capital Cost	Item	$M
	Wilga Contractor Mobilisation and Mine Re-opening	1.20
	Wilga Pre- production development and mining	1.30
	Property and Lease Purchase	0.38
	Plant Construction	20.60
	Infrastructure, Light Vehicles and Housing	1.25
	Project Management, Engineer and Metallurgist	0.30
	Commissioning and Recruitment	0.35
	TOTAL	25.38
Other Funding Requirements		
	Environmental Bond	1.50
	Working Capital (Concentrate Stocks)	4.50
	Corporate Costs (Construction Year)	1.00
	Surplus Working Cash	1.00
	TOTAL FUNDING REQUIREMENTS	33.38

OPERATING COSTS-600,000 TPY

Activity	Comment	Units	Cost
Mining - Wilga	Refer report, no contingency	$/tonne ore	23.1
Mining - Currawong	Refer rep't, incl. Cap. Dev, no contingency	$/tonne ore	25.5
Mill haulage - W	B'Fox Cont. – pre owned trucks	$/tonne ore	3.5
Mill haulage - C	B'Fox Cont. – pre owned trucks	$/tonne ore	2.0
Milling	Refer Eng. Report, no conting'cy	$/tonne ore	21.0
Admin/Lease/Env	Refer attached detailed estimate	$/tonne ore	3.4
Cu conc t'port PK	Refer Colken report – Heggies	$/tonne conc	70.0
Zn conc t'port SC	Refer Colken report – Heggies	$/tonne conc	87.5
Power	Refer report - 36.9 cpl diesel	C/kWHr	13.5
Smelter Charges	Refer Fin. Assump'ts/Colken		
Royalty	Refer Fin. Ass'ts – no Gov/WMC	$/tonne ore	0.8

BENAMBRA MINE – ADMINISTRATION OPERATING COST ESTIMATE

These costs are based on the guidelines from the 1993 AUSIMM Cost Estimating Handbook (suitably escalated to current day costs), current data and a review of the historic Denehurst costs for the 6 months ending December 1995.

ITEM	EST OPEX $	HISTORIC OPEX – PRORATA 12 MTS	COMMENTS
1) Personnel			
Operations GM x1	130,000		Salaries incl. 8% on costs
Admin. Manager x1	100,000		
Accountant x1	60,000		
Invoice Clerk x1	43,000		
Receptionist/Sec. X1	35,000		
HR/Training/Enviro. x1	43,000		
First Aid/Train/Security x1	43,000		
Store Super. X1	65,000		
Storeperson x1	38,000		
SUB TOTAL	557,000	472,000	
2) Payroll Tax	29,800	-	5.35%
3) Workcover	15,000	-	2.70%
4) Light Vehicles	120,000	62,000	8 vehicles: 4 admin, 2 mine, 2 mill – opex = 43% of capex = 8x35kx43% This includes for high KM per year. Denehust had other light vehicles

			built into their mining and processing costs for operator transport
5) Personnel Transport by Bus Contractor	193,000	82,000	3 round trips/day for 5 days/wk + 2 round trips/day for 2 days/wk – 130km/trip x $1.5/km
6) Communications	80,000	33,000	$600/yr/person – 80 people plus facility charge for satellite lines
7) Safety/Training	60,000	65,000	3% of salary and wages cost – based on 40 direct employees at an average of $50k/a
8) Insurance	222,000	222,000	0.5% of plant capex value – say $30M; Plus loss of profits Denhurst was $111k for 6 months – provisions account?
9) Recruitment Fees & Relocation	*(200,000) then12,000/annum	6,000	*(Initial recruitment for 8 principal management and technical staff), then 8% turnover of 40 direct employees – principally local recruitment.
10) Rental Assistance	88,000	106,000	Limited to imported technical staff – 10 x $150/wk plus phone & power subsidy; Denehurst also allowed power/phone/heat subsidy

11) Legal Fees	-	-	Corporate allowance
12) Entertainment – Recreation - Sports	10,000	7,000	Allowance from est. manual for non remote site
13) First Aid – Safety Equip	15,000	10,500	Allowance from est. manual
14) Licence Fees	5,000	-	
15) Computer Support – Software	30,000	24,000	Leasing + allowance for operating costs from est. manual
16) Postage	8,000	6,400	
17) Road Freight	60,000	102,000	3 x 12 tonne trucks x $31/t/week
18) Office Stationery	20,000	20,000	Based on current experience and historic costs
19) Subscriptions	12,000	11,000	
20) Donations	5,000	5,000	
21) Bank Charges	15,000	14,000	
22) Auditing	-	25,000	Allowed in Corporate budget. Denehurst historic $12.5K in 6 months.
23) Admin. Mtce & Cleaning	40,000	35,000	Includes the cleaning and maintenance of the ablutions and administration facilities.
24) Equipt. Hire	4,000	3,000	
25) FBT	43,000	55,000	FBT on light vehicles: 8 x $35,000 x 365/365 x 11% + $10,000 for entertainment etc.
26) Lease Interest Expense	-	62,000	Covered in Corporate allowance and

			light vehicle allowance
27) Magazines & Papers	1,500	1,500	
28) Travel and Accommodation	25,000	28,000	
29) Consultants	12,000	10,000	
TOTAL	1,682,300	1,467,400	

NOTES:

- In 1993 the Denehurst Administration Operating costs were $1.656M which may reflect additional costs associated with the early days of operation.
- In 1995 Denehurst had budgeted for an Administration Operating cost of $1.612M but actual appears to be about $120,000 less.
- The 1993 AUSIMM Estimating Manual indicates, from extensive surveys, that the Administration Operating costs for a small lean mine should be about 10% to 12% of total mine cash operating costs. Based on the Mining and Milling costs in the Model this would equate to $47/tonne x 10% x 600,000tpa = $2.82M. As a partial offset we don't have as high administrative costs associated with mining as we are using a contract mining organization in a partnership relationship. The mining contractor has an administration component built into the mining unit rates.
- The Austminex Financial Model currently incorporates an annual Administration Operating cost of $1.08M plus $0.16M in lease costs. This allowance is insufficient.

Andrew McDougall

31st May 2001

AUSTMINEX NL

BENAMBRA PROJECT

VALUATION AND MODELLING

SUMMARY REPORT

BENAMBRA PROJECT VALUATION AND MODELLING

1. VALUATION METHOD

The discounted cash flow analysis ("DCF") has been used to derive a net present value ("NPV") for an operation at 600,000 tonnes per annum based on the July 2001 ore reserve estimate for Wilga and Currawong which comprise the Benambra Project.

The Ore Reserves of the Wilga and Currawong deposits, as estimated by McArthur Ore Deposit Assessments Pty Ltd (MODA) in July 2001 are:

TABLE 1

WILGA ORE RESERVES

(all Probable)

Lens	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%Zneq
Wilga	1636	2.4	0.51	6.1	35	0.50	12.0
TOTAL	**1636**	**2.4**	**0.51**	**6.1**	**35**	**0.50**	**12.0**
rounded	**1.6Mt**						

TABLE 2

CURRAWONG ORE RESERVES

(all Probable)

Lens	'000 Tonnes	%Cu	%Pb	%Zn	g/t Ag	g/t Au	%Zneq
A	93	1.5	1.2	6.2	50	1.9	10.0
B	253	1.9	1.3	4.3	44	1.9	9.0
J	24	2.7	0.3	2.0	25	0.6	8.7
K	195	1.1	1.6	6.3	51	2.0	9.0
M	3890	2.2	0.7	4.3	36	1.2	9.6
TOTAL	**4455**	**2.1**	**0.78**	**4.4**	**37**	**1.3**	**9.6**
rounded	**4.5Mt**						

2. VALUATION

At present, it is considered that the data is known in sufficient detail to provide a reliable estimate at a recognised "feasibility study" level and that most inputs are known within an accuracy of around plus or minus 10%.

The Base Case model has a NPV @ 10% of approximately $10 million and a positive rate of return of 18%. But the model does not provide sufficient debt capacity to advance the project. It should be noted that this estimate is based on no State or WMC royalty.

Base Case Metal Prices & Exchange Rates

Zinc	Yr 0 to 2	US c/lb	43.5	Bankwest 27 Mth Forecast (10th August 2001)
	+Yr 2	US c/lb	52	Long Term Forecast (Average of 15 forecasters)
Copper	Yr 0 to 2	US c/lb	72.1	Bankwest 27 Mth Forecast (10th August 2001)
	+Yr 2	US c/lb	90	Long Term Forecast (Average of 15 forecasters)
ER	Yr 0 to 2	$A/$US	0.5	Bankwest 27 Mth Forecast (10th August 2001)
	+Yr 2	$A/$US	0.6	Long Term Forecast (Average of 15 forecasters)

Austminex - Benambra Project - (Base Case)

Production, Revenue & Commercial Assumptions

		2001/02	2002/03	2003/04	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12
Mill Treatment	- tonnes		550000	600000	600000	600000	600000	600000	620000	640000	640000	640000
	- % Cu		2.2	2.6	2.3	2.0	2.2	2.5	2.4	1.7	2.1	1.7
	- % Zn		6.7	6.2	5.1	4.5	4.7	4.1	4.8	4.5	3.5	4.7
	- % Pb		0.5	0.5	0.6	0.9	0.7	0.7	0.7	0.9	0.7	0.8
	- g/t Ag		36	36	34	41	41	33	37	39	30	39
	- g/t Au		0.5	0.5	0.6	1.4	1.1	1.4	1.2	1.3	1.3	1.1
Metallurgical Recoveries	% Zn to Zn Conc		85	85	84	80	80	80	80	80	80	80
	% Cu to Copper Conc		83	83	83	83	83	83	83	83	83	83
	% Ag to Copper Conc		31	36	33	24	26	35	31	21	33	21
	% Au to Copper Conc		15	14	15	20	16	21	18	19	19	16
Concentrate Production												
Zinc	- tonnes		62183	62947	51302	42078	44364	38345	46379	45395	35157	47283
	- % Zn		50.00	50.00	50.20	51.00	51.00	51.00	51.00	51.00	51.00	51.00
Copper	- tonnes		40432	51535	44672	39088	42757	46990	48093	33862	42008	34164
	- % Cu		25	25	25	26	26	26	26	26	26	26
	- g/t Ag		150	150	150	150	150	150	150	150	150	150
	- g/t Au		1.1	0.8	1.3	4.3	2.4	3.8	2.9	4.8	3.7	3.4
Revenue Statement		A$M	A$M	A$M	A$M	A$M	A$M	A$M	A$M	A$M	A$M	A$M
	Net Smelter Rev	0.0	48.7	54.9	50.4	46.0	48.3	49.0	53.0	44.3	44.2	44.8
		0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
	Transport Road, Storage, Insurance	0.0	9.5	10.5	8.8	7.4	7.9	7.7	8.6	7.3	7.0	7.5
	NRV Production	0.0	39.1	44.4	41.6	38.5	40.4	41.3	44.5	37.0	37.2	37.2
Commercial Assumptions:												
Metal Prices												
Zn	$US c/lb	43.5	43.5	43.5	52.0	52.0	52.0	52.0	52.0	52.0	52.0	52.0
Cu	$US c/lb	72.1	72.1	72.1	90.0	90.0	90.0	90.0	90.0	90.0	90.0	90.0
Ag	$US/oz	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4	4.4
Au	$US/oz	265.0	265.0	265.0	265.0	265.0	265.0	265.0	265.0	265.0	265.0	265.0
Exchange Rate	$A/US	0.50	0.50	0.50	0.60	0.60	0.60	0.60	0.60	0.60	0.60	0.60

Austminex - Benambra Project - (Base Case)

Profit and Loss Statement		2001/02	2002/03	2003/04	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12	2012/13
		$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	
Revenue	Revenue Production	0.0	39.1	44.4	41.6	38.5	40.4	41.3	44.5	37.0	37.2	37.2	
	Revenue Other												3.0
	Mine Revenue	0.0	39.1	44.4	41.6	38.5	40.4	41.3	44.5	37.0	37.2	37.2	3.0
Mining Costs													
	Mining cash cost	0.0	16.3	17.1	14.1	13.3	13.2	14.5	14.9	15.6	15.4	15.2	
	Non Cash Cost												
	Amortisation	0.0	0.8	0.9	1.2	2.2	2.2	2.2	2.3	2.3	2.3	2.3	
	Depreciation	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	
	Total Mining Costs	0.0	17.1	18.1	15.3	15.5	15.4	16.7	17.1	17.9	17.7	17.5	
Milling Costs													
	Ore Treatment	0.0	11.5	12.4	12.8	12.7	12.8	12.8	12.9	13.6	13.6	13.2	
	Depreciation	0.0	3.3	3.6	3.7	3.7	4.3	4.4	4.4	0.8	0.5	0.5	0.0
	Total Mill Costs	0.0	14.8	16.0	16.5	16.4	17.1	17.1	17.3	14.4	14.1	13.7	0.0
Other Costs													
	Mine Administration	0.00	1.84	1.84	1.84	1.84	1.84	1.84	1.84	1.84	1.84	1.84	0.00
	Environment	0.00	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	0.20	3.00
	Royalty	0.00	0.44	0.48	0.48	0.48	0.48	0.48	0.50	0.51	0.51	0.51	0.00
	Interest & Bank Guarantee												
	Total Other Costs	0.00	2.48	2.52	2.52	2.52	2.52	2.52	2.54	2.55	2.55	2.55	3.00
Total costs		0.0	34.3	36.6	34.3	34.5	35.0	36.3	37.0	34.9	34.4	33.8	3.0
Mine Profit		0.0	4.8	7.8	7.3	4.1	5.4	5.0	7.5	2.1	2.8	3.4	0.0
Income Tax Expense @	30 %	0.0	0.2	2.3	2.2	1.2	1.6	1.5	2.2	0.6	0.9	1.0	0.0
Net Profit after Tax		0.0	4.6	5.4	5.1	2.8	3.8	3.5	5.2	1.5	2.0	2.4	0.0

Austminex - Benambra Project - (Base Case)

Cash Flow & Capital Expenditure

		2001/02	2002/03	2003/04	2004/05	2005/06	2006/07	2007/08	2008/09	2009/10	2010/11	2011/12	2012/13
Cash Flow		$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M	$M
Revenue		0.0	39.1	44.4	41.6	38.5	40.4	41.3	44.5	37.0	37.2	37.2	3.0
Less:	Operating Costs	0.0	34.3	36.6	34.3	34.5	35.0	36.3	37.0	34.9	34.4	33.8	3.0
Earnings Before Tax		0.00	4.81	7.75	7.30	4.06	5.39	4.99	7.47	2.08	2.85	3.45	0.00
Tax Payments		0.00	0.21	2.33	2.19	1.22	1.62	1.50	2.24	0.62	0.85	1.03	0.00
Net Profit		0.00	4.60	5.43	5.11	2.84	3.77	3.49	5.23	1.46	1.99	2.41	0.00
Add:	Depreciation & Amortisation	0.00	4.14	4.56	4.87	5.94	6.48	6.55	6.71	3.11	2.85	2.80	0.00
		0.00	8.74	9.99	9.98	8.78	10.25	10.04	11.94	4.57	4.84	5.21	0.00
Less:	Working Capital Movements	0.00	4.58	0.51	-0.27	-0.30	0.18	0.09	0.30	-0.73	0.02	0.00	-4.39
Surplus from Operating		0.00	4.16	9.48	10.24	9.08	10.07	9.95	11.63	5.30	4.82	5.21	4.39
Less:	Capital Expenditure	25.38	0.20	3.55	5.35	3.18	5.78	1.95	1.93	1.36	0.75	0.00	0.00
Cash Generation		-25.38	3.96	5.93	4.89	5.90	4.30	8.00	9.71	3.94	4.07	5.21	4.39
Cumulative Cash Flow		-25.38	-21.42	-15.49	-10.60	-4.69	-0.40	7.60	17.31	21.25	25.32	30.53	34.91
NPV @ Dec 2002													
[Npv @ 7.5% $M14.7]	Discounted Cash (7.5%)	-25.38	3.68	5.13	3.94	4.42	2.99	5.18	5.85	2.21	2.12	2.53	1.98
[Npv @ 10% $M10.1]	Discounted Cash (10%)	-25.38	3.60	4.90	3.68	4.03	2.67	4.51	4.98	1.84	1.72	2.01	1.54
IRR 17.8%													
Tax payments													
Mine profit pre tax		0.00	4.81	7.75	7.30	4.06	5.39	4.99	7.47	2.08	2.85	3.45	0.00
Add accounting amortisation/depreciation													
Less tax depreciation claim													
Carry forward tax losses acquired													
Taxable income		0.00	4.81	7.75	7.30	4.06	5.39	4.99	7.47	2.08	2.85	3.45	0.00
Tax payable @ 30%		0.00	0.21	2.33	2.19	1.22	1.62	1.50	2.24	0.62	0.85	1.03	0.00
Capital Expenditure													
Underground Development		2.50	0.00	1.15	5.00	2.83	2.53	1.60	1.58	1.06	0.60	0.00	0.00
Exploration													
Equipment & Fixed Assets	Mining	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00	0.00
	Mill - & Other	22.88	0.20	2.40	0.35	0.35	3.25	0.35	0.35	0.30	0.15	0.00	0.00
	Sub total	22.88	0.20	2.40	0.35	0.35	3.25	0.35	0.35	0.30	0.15	0.00	0.00
Total		25.38	0.20	3.55	5.35	3.18	5.78	1.95	1.93	1.36	0.75	0.00	0.00

Since September 2000 base metal prices in general have trended downwards, in reaction to the slowdown in world economic growth, and the outlook for the next twelve months has become increasingly gloomy. Copper prices in US dollar terms have fallen by some 25% during the last twelve months and the fall in zinc prices over the same period is close to 30% and both metals are currently at a 14-year low. The weak Australian dollar has provided some relief, but increasing metal stockpiles, as demand falls away, has resulted in a substantial deterioration in smelter terms for base metal concentrate producers.

The prevailing conditions have a direct adverse affect on the Benambra Project economics and the ability to fund its development

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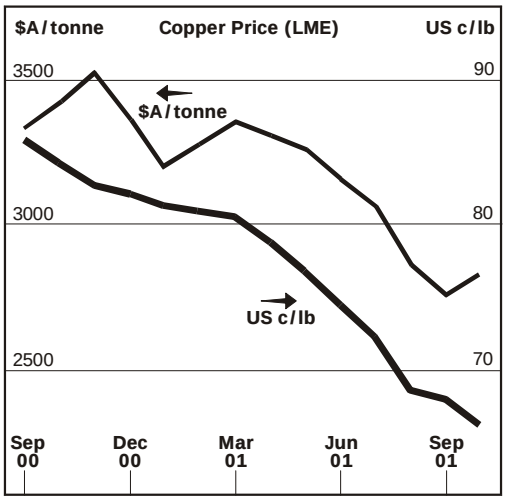
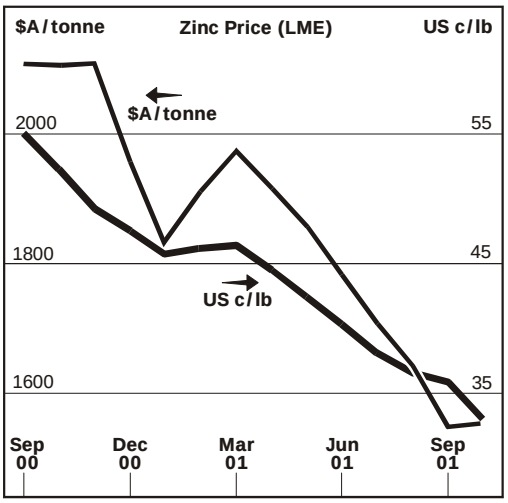
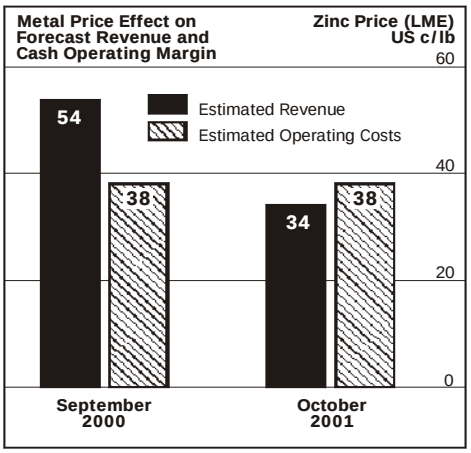
Graphs of Metal Price Trends since September 2000

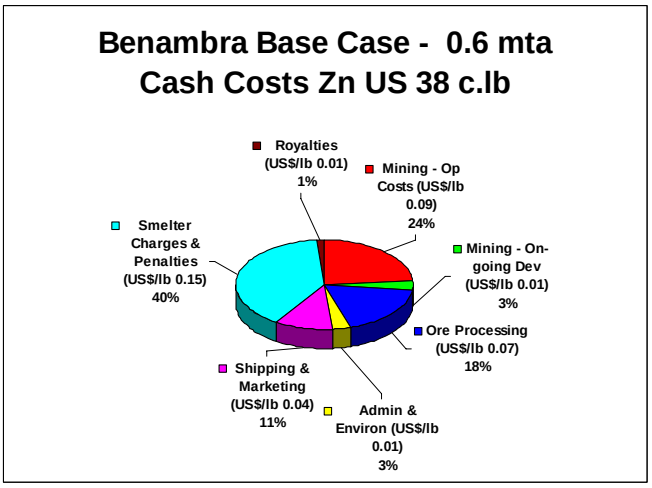




Graph of Metal Price Effect on Forecast Revenue US c/lb and Cash Operating Margin



Graph of Unit Cash Costs



3. SUMMARY TABLE OF COST INPUTS, METALLURGICAL PERFORMANCE AND SEMLTER TERMS AND CHARGES

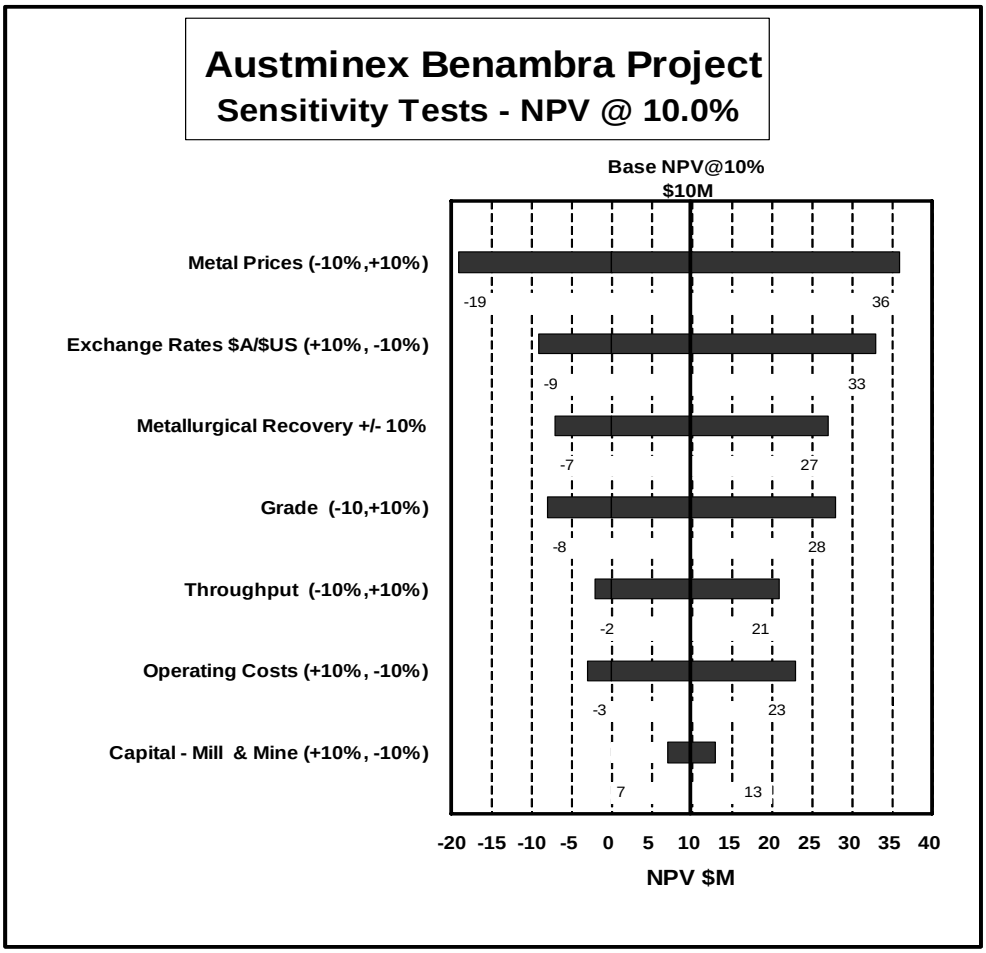
Summary of Project Cost Estimates, Metallurgical Performance and Smelter Terms and Charges

				Base	Comments
Mining Costs					
Wilga			A$/tn	23.1	Based on contract budget prices
Currawong		Op Cost	A$/tn	17.7	Based on contract budget prices
		Op Dev	A$/tn (avg)	4.2	Based on contract budget prices
		Total	A$/tn (avg)	21.9	Based on contract budget prices
		Capital Dev	A$/tn (avg)	3.6	Based on contract budget prices
		Total		25.5	Based on contract budget prices
Surface Haulage to Mill		Wilga to Mill	A$/tn	3.5	Bonney Fox Estimate (Using pre-owned Trucks)
		Currawong to Mill	A$/tn	2.0	Bonney Fox Estimate (Using pre-owned Trucks)
Milling Costs			A$/tn	21	Austminex/Metplant
Admin Costs inc lease and environmental			A$M/yr	3.36	Austminex
Freight & Shipping	Transport Cu to Port Kembla		A$/tn	70.00	Colken/Heggies Transport
	Transport Zn to Sulphide Corp		A$/tn	87.50	Colken/Heggies Transport
Power			c/kWHr	13.5	Based on estimates from Generator Sales & Rentals
Smelter TC; & RC,s	Copper	Treatment	$US/tn	85	Colken
		Refining	US c/lb	8.5	Colken
		RC Ag	US $/oz	0.3	Colken
		RC Au	US $/oz	5	Colken
		Penalty (Zn+Pb)	US $/tn	10	Colken
	Zinc	Treatment	$US/tn	185	Colken
		Penallty (Fe&Bi)	$US/tn	5.2	Colken
Royalties	Vanufo		$/tonne	0.8	Existing Agreement
	Government	% of NSR at mine gate.		0	Assume can be negotiated to 0%, otherwisw 2.75%
	WMC	% Revenue-costs-capital		0	Assume can be negotiated to 0%, otherwisw 5.1%
Capital(Start Up)	Mill	Year 1	A$M	20.60	Final Estimate - Met Plant - August 2001
	Mine	Year 1	A$M	2.50	Based on contract budget prices
	Other	Year 1	A$M	2.30	Lease Purchase, Housing, Light Venicles etc
		Total	A$M	25.40	
Metallurgical	Copper	Wilga	Recovery %	83	Final Estimate - Austminex/Optimet
			Cu Grade %	25	Final Estimate - Austminex/Optimet
			Au Recovery %	14	Final Estimate - Austminex/Optimet
			Au g/t (avg)	0.77	Final Estimate - Austminex/Optimet
			Ag Recovery %	40	Final Estimate - Austminex/Optimet
			Ag g/t (avg)	150	Final Estimate - Austminex/Optimet
		Currawong	Recovery %	83	Final Estimate - Testwork Completed July
			Grade %	26	Final Estimate - Austminex/Optimet
			Au Recovery %	14.3	Final Estimate - Austminex/Optimet
			Au g/t (avg)	2.2	Final Estimate - Austminex/Optimet
			Ag Recovery %	30	Final Estimate - Austminex/Optimet
			Ag g/t (avg)	150	Final Estimate - Austminex/Optimet
	Zinc	Wilga	Recovery %	85	Final Estimate - Austminex/Optimet
			Grade %	50	Final Estimate - Austminex/Optimet
		Currawong	Recovery %	80	Final Estimate - Austminex/Optimet
			Grade %	51	Final Estimate - Austminex/Optimet

4. PROJECT SENSITIVITIES

As can be observed from the sensitivity graph the project is most sensitive to metal prices, exchange rates, metallurgical recovery and grade. Though not as sensitive to throughput and operating costs, a 10% change can still produce a $10M change in NPV. The project is least sensitive to capital as the relative low cost of $28M only produces a variance of +/- $3M.

Unconstrained Sensitivity Analysis



There are various techniques for controlling the down side effects of the model parameters.

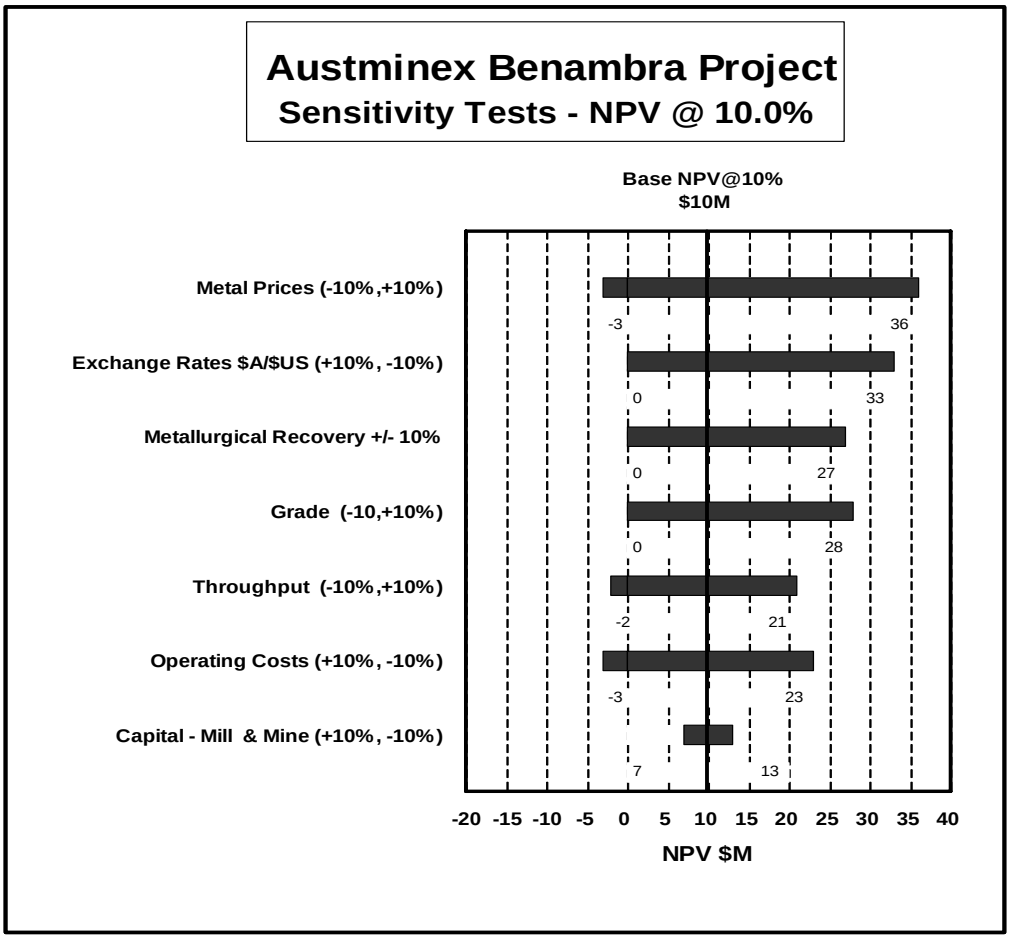
- Metal Prices, the use of hedging can reduce the downside risk of metal prices for the first 3 to 5 years of a project. The effect of hedging for 5 years would be to reduce the downside for a 10% drop in overall metal prices from -$19M to -$3M.

- Exchange rates can be locked in to limit the downside this reduces the effect from around -$9M to $0M.

- The downside risk of grade and recovery can be managed by increasing throughput. A 10% increase in throughput can reduced the individual grade/recovery downsides from -$7 and -$8M to approximately $0M.

The net effect of using these techniques for project management is illustrated in the sensitivity graph below. A project that had a maximum downside of -$19M for a 10% variation in metal prices can be managed to have a downside of -$3M, for a 10% variation in one of the main project parameters.

Constrained Sensitivity Analysis



5. METAL PRICES FOR AN ECONOMIC PROJECT

Two cases were examined to see what metal prices were required to make the project capable of 100% debt funding, or capable of a 50/50 debt/equity mix.

The 100% Debt Funding model requires metal prices for Zinc of ***A$2200/tonne***, and for Copper of ***A$3650/tonne***. The current Cu price under A$3000/tonne is well below the required level but in 1989 and 1995 the copper price was at or above these levels. (See attached price graphs). Also in 1989 the Zn price (current price A$1583/tonne) was close to the required level, therefore there is some historic basis to assume the prices will reach these levels again.

- Zinc $2200/tonne, is equivalent to US60c/lb at an exchange rate of 0.6.
- Copper $3650/tonne, is equivalent to US100c/lb at an exchange rate of 0.6.

The 50% Debt/50% Equity model requires metal prices for Zinc of ***A$2050/tonne***, and for Copper of ***A$3500/tonne***. In September 2000 both Zinc and Copper prices were at these levels, and if the prices and exchange rate had been locked in at this time then the project NPV of $26M would have justified the equity input and loan conditions.

- Zinc $2050/tonne, is equivalent to US55.6c/lb at an exchange rate of 0.6.
- Copper $3500/tonne, is equivalent to US94.5c/lb at an exchange rate of 0.6.

6. PROJECT MODELLING HISTORY

Before arriving at the current model many models were developed and examined ranging from “Boots Strap” models at 0.3 million tonnes per annum (mta), start-up at 0.3mta and scale up to 0.6mta, and full scale start up at 0.6mta.

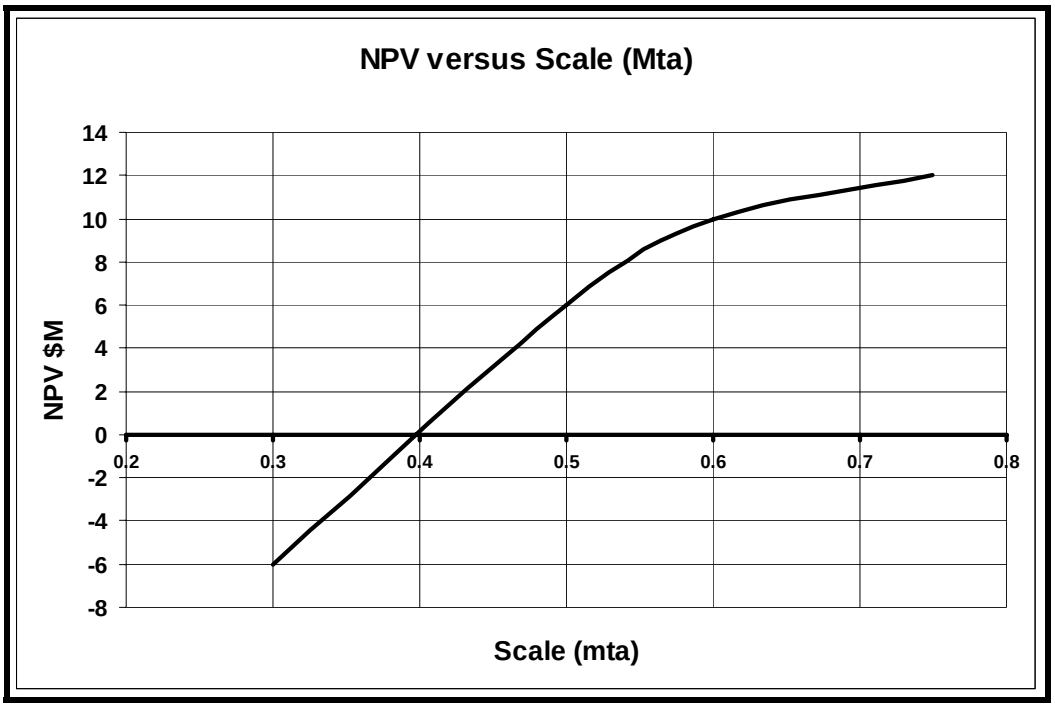
The lower throughput 5year models were based on the higher-grade Wilga deposit and althought the models were showing a positive cash flow, they ran out of tonnage before generating a positive NPV or achieving an economic rate of return. Developing and mining Currawong at realistic grades and at a rate of 0.3 million tonnes per annum always resulted in a negative rate of return.

Invariably bulk mining at 0.6mta produced the best rate of return and NPV.

Additionally models were run from 0.3mta up to 0.75mta. The best returns were achieved for the higher throughput models but the size and disposition of the resource cannot sustain production rates above 0.6mta.

Using the assumptions at the time, the 0.6mta model was adopted as the base case, as this model was capable of close to 100% debt funding. The initial 0.6mta model did not have enough reserves for a 10year life, so a programme of drilling was initiated to expand the resource base and upgrade resources to indicated category. This programme was successful and a 10year ore reserve was established for the 0.6mta model.

Since the early studies, September 2000 to April 2001,metal prices have deteriorated to such an extent that the 0.6 million tonnes per annum model cannot support 100% debt funding.



AUSTMINEX NL

BENAMBRA PROJECT

POWER SUPPLY & SERVICES

SUMMARY REPORT

POWER SUPPLY

Power for the previous mining operation was provided by on site diesel generators, supplied and operated by Aggreko. Denehurst paid Aggreko a facility fee in cents/kWH for the supply, operation and maintenance of the plant (which included high voltage switchgear and step up transformers) and an operating cost relating to the fuel consumed in cents/kWH. The total cost of power was in the range of 11 to 12 cents/kWH.

For the re-opening study Austminex investigated a range of supply options:

Contractor provided on site diesel generation

Contractor provided reticulated gas generation

Contractor provided on site non reticulated LPG generation

Contractor provided on site biomass fired boiler for steam turbine generation

Wind power generation

Grid power connection

Biomass Steam Generation

Renewable Energy, ph. 03 9820 1322, proposed the installation of a 5 MW generator and associated boiler, supported by diesel backup units. The proposal was to obtain waste biomass material from sources such as the Maryvale pulp mill and cart it to site as back load in empty concentrate transport trucks. Normally plants such as this are associated directly with a process which produces a combustible waste product, ie co-generation.

The indicative cost for supply of power, based on the provision, operation and maintenance of the facility is between 12.0 and 13.5 c/kWH. This is conditional that organic waste fuel (70,000 to 80,000 TPY) can be back loaded in trucks used for the outward shipment of concentrates. If the fuel was to incur transport costs due to the back loading facility being unavailable, the power cost will be considerably more expensive. Any carbon credits available for the use of a renewable fuel are to the account of Renewable Energy and have been built into the power cost structure.

There is currently no water tight guarantee that political pressure wont cause this form of energy to be reclassified as non renewable and hence in that case would not enjoy the benefits of carbon credits. Chris Uren of Renewable Energy expressed the view that they would not be in a position to provide Austminex with a bankable proposal.

An alternative on site steam generation proposal was put forward by Energy Brix (Richard Jackson ph. 03 9565 9836, mob. 0411 467 419). For their proposal they planned to use brown coal briquettes. The indicative all up BOO cost of power was around 17 c/kWH. Again the pricing was contingent on the briquettes being back

loaded in the concentrate trucks. Due to the briquettes having a higher calorific value than organic wastes, the annual consumption of briquettes is around 25,000 to 30,000 TPY.

Currently Benambra based logging contractor, Gil Parker, is investigating the possibility of providing power to the mine, using low grade wood chips obtained from the local forest. This proposal is in its infancy, but preliminary indications are that the power cost will be in the region of 13.5 c/kWH. Gil is aware that the mine reopening is currently on hold, but he is quietly going to continue gathering information and developing the proposal.

On Site Gas Fuelled Generators

Energex who are capable of providing many forms of power generation, as well as grid connection, investigated a range of supply methods including the running of a gas pipeline from the coast to the mine site. Due to the topography of the country and distance the capital cost for the pipeline exceeded the cost of grid power connection.

They also considered using LPG fuelled on site generators, but this proved to be too expensive. Their final conclusion was that on site diesel generation provided the only feasible option.

Westfarmers Group who provide LPG fuelled generators to remote locations in South and West Australia were contacted. They said that they had never been successful in establishing power generating facilities at mine sites as the LPG can't compete with diesel, when the mines are eligible for the diesel fuel rebate. Currently there is no rebate for LPG. LPG only has a calorific value of about 2/3rds that of diesel. LPG cannot be used in conventional diesel engines and as such requires special larger capacity engines, of about triple the cost, in order to generate the same power as a diesel. The efficiency of an LPG engine is in the order of 20%. Therefore an LPG facility, when the customer is able to claim the diesel fuel rebate, cannot compete on both fuel and capital costs. The power cost at Benambra for an on site LPG facility would be up to 22 c/kWH.

Wind Power

There was cursory consideration of this form of power by Energex. It was quickly established that there is not sufficient dependable wind year round for this to be the sole source of supply.

Grid power

There have been numerous discussions with TXU regarding connection to the grid system as they are the owners of the network in this district. With the grid connected, Austminex would be able to obtain a contestable power supply by contracting with a provider who has the best terms and lowest energy cost. TXU have provided an indicative two component energy cost of around 7.6 c/kWH. It is

possible to lock in around 80% of this price for 4 years, with the remaining component subject to rise and fall. The contract would be take or pay.

Whilst the network is at the nearby town of Benambra it is low grade and is incapable of supporting the mine. There is currently very little spare capacity in the system as it is only designed to supply small communities and low density, low load farms.

For the system to have the capacity to supply the relatively large mine load (6MW installed and 5MW operating) and machinery starting currents, it has to be upgraded right back to the substation at Bruthen, a short distance inland from the coastal town of Bairnsdale. This entails the running of a new 66kV line (approx. 130 km) above the existing 22kV line and the installation of heavy duty voltage regulation and harmonics control, so that other customers on the system are not affected by the mine. A back of the envelope estimate from TXU for the line upgrade is $20M; 90% of the cost for the upgrade to the town of Benambra and 5% for the connection to the mine. A survey found that at present there are no existing large or foreshadowed industries in the area which require power and which would help justify the installation of an upgraded system and help with the capital cost.

TXU have indicated that an engineering study is required in order to confirm the capital cost and produce an estimate to within +/- 10%. TXU have indicated that they are not in a position to provide the study without payment from Austminex or to contribute to the capital cost of the line upgrade. The cost of the study is estimated at $133,000 + GST and is expected to take 13 weeks to complete. The upgrade of the line is estimated to take around 12 months from commencement of work. It is likely to follow the existing easement and will require new and double the number of taller poles and will have to be done with the existing 22 kV system live. By using the existing easement it should eliminate the need for extensive and prolonged environmental and native title negotiations.

The TXU contacts are: Energy Supply – Geoff Duke ph. 03 8628 1179 and Network System – Max Rankin ph. 03 5760 2480.

Austminex extensively lobbied both Federal and State Governments, to have them collectively provide the upgraded line to the mine. This approach was made on the basis of providing employment in a depressed region of Australia and to save the Victorian Government the cost of having to rehabilitate the Tailings Facility.

The Federal Government has stated that it does not directly fund infrastructure for individual private industry projects and the Victorian Government Treasurer and Minister for State and Regional Development has written to say that his government will not provide funds towards the power system upgrade. Austminex with the current project economics cannot afford to contribute towards or pay for the capital cost.

Contractor provided on site diesel generation

Enquiries were issued to a number of contractors on the basis of providing, operating and maintaining the facilities, with Austminex providing the rebated diesel fuel to the contactor. This is in the same manner as was previously done when Denehurst operated the mine.

The quoted facility fees ranged from 3.74 C/kWh to 6.76 c/kWh. The energy cost is based on the specific fuel consumption of the generators (fuel efficiency) and the landed cost of diesel fuel at the mine site. Fuel prices were obtained from Shell (the previous supplier) and with the diesel fuel rebate, a long term projection, on site, price of 36.9 c/L was used for the study. With the volatility of oil prices and exchange rates, fuel providers will no long provide fixed pricing with simple long term escalators. The price of fuel can vary from one month to the next and may be even at greater frequency. This is a problem for the mine's economics as the cost of power is around 100% higher than grid power and could well rise to levels much higher than this. As this currently is the only practical means of providing power to the project, a power cost of 13.5 c/kWh was used for the study. If grid power could be used it would provide far greater certainty of power cost and would result in operating cost savings of around $3.50/tonne of ore treated.

As a variation to using diesel engines, the possibility of using a secondhand low revving ships generator, running on bunker oil, was investigated. It appears that suitable machines, up to 10MW capacity, are available from wrecked ships. Whilst the basic plant can be reasonably cheap, the set up cost can be significant. The bunker oil is particularly difficult to handle and has to have steam heated storage facilities and heat traced fuel lines (particularly at Benambra where it snows in winter). The tankers bringing the fuel to site also have to be double insulated and heated in order to keep the fuel fluid. The final nail in the coffin is the price, as the bunker oil is not eligible for a rebate from the government and so is considerably more expensive than diesel.

Metplant Engineering produced a specification for Contract Power – Generation and HV Distribution. This specification can be found in the Study Folder designated Plant Engineering.

WATER SUPPLY

Previously Denehurst had a licence from Southern Rural Water to extract 30 ML of water from the upper Tambo River system.

This licence has been renewed in the name of Austminex, licence number 1000977. This licence has to be renewed on an annual basis by payment of the applicable fee.

12 September 2001

Gil Parker
OCB Logging
PO Box 199
Benambra Vic 3900

Dear Gil

Benambra Mine Power Supply

Following our phone conversation regarding the possibility of using chipped hard wood generated power, please find enclosed a copy of the document specifying the power requirements for the Benambra Project.

At this stage there needs to be sufficient work done to credibly establish the practicality, reliability, political sensitivities and order of magnitude costs for the provision of power by this means. The long term availability of fuel, the supply/consumption of water and the requirements for heat exchanging/provision of condensing/recycle need to be established. Previous proposals based on steam generated power, using organic fuels have not provided the savings in costs that may initially have been expected.

The reliability of power is important, with the facility having to meet the production requirements of the mine which is 24 hours per day, 365 days per year and with expected defined shutdown periods for maintenance of between half to one day duration at around eight week intervals. Also it is anticipated that the crushing plant would only operate for around eighteen hours per day so that the power generating plant will require flexibility to turn down and wind up its output. For these reasons consideration needs to be given to the response times and to having redundancy or backup provided in the power supply facility.

In addition there may be a requirement in the future to provide power from the central facility for the development and operation of the Currawong mine. This load is expected to be up to one megawatt.

I will only consider involving our consulting engineers if there appears to be some prospects for the wood chip based power being practical, reliable and economic. Please address any questions you may have, to me in the first instance.

I look forward to hearing your response once you have given further consideration to the proposal.

Yours sincerely

Andrew McDougall
Operations Manager

Grid Power – TXU discussions 26/07/01

Kevin/Bruce,

I rang Max Rankin who is the technical person with whom I have been dealing in recent times. He is currently working out a cost estimate for preparing a definitive capital cost estimate for the line upgrade and installation. This has necessitated him taking into account load details provided by John Callaghan of Metplant and review of maps of the area and the line route. He provided the following answers/comments to my inquiries.

- the new line will follow the existing route and will be a "backbone" arrangement. This means that the new 66kV conductors are run above the existing 22kV conductors. To do this requires the replacement of the existing poles with longer ones of timber or concrete construction. The poles will look similar to the current poles and won't be pylons.
- due to the extra conductors the span between poles will be halved, requiring twice as many poles.
- the work will be done with the existing conductors live and without interruption to existing customers. This adds to the difficulty and cost, but is preferable, quicker and cheaper in this topography, to running a completely separate line on its own route.
- due to the mine's load and the size of the equipment which has to be started, they have reviewed the line requirements and now say that the 66kV line has to be run through to the town of Benambra via Omeo. This will be 95% of the cost.
- the existing customers on the 22kV system will be back fed from a connection to the 66kV line at Benambra.
- the final line run (approximately 16km) into the mine will be 22kV. This will cost about 5% of the capital cost.
- the overall capex is still likely to be $20m+
- there is more involved with the design of the line than Max had envisaged and that the cost for the study will be considerably more than he first thought. He says we might be shocked. He didn't quote a figure and expects to get back to us in about a week.
- I asked about TXU's contribution to the capex. Max said this used to come under the old SEC guidelines but since January of this year comes under the direction of the Regulator General. In determining any contribution by TXU they will be required to take into account the revenue that they will receive, the capital cost of the installation, the operating costs of the system and the system maintenance costs. With long lines to single customers the cost of operating and maintenance will be high in comparison with to revenue received. This means that TXU will not be contributing to the capital cost.
- mines are not considered to be robust customers and have had a history of closing prematurely. Supply organisations, governments etc. tend to be wary/cautious as they have been caught before.
- the existing power infrastructure to the district is adequate for the projected loads for the next 20 to 30 years. On this basis if the mine was to close prematurely and a politian/bureaucrat was to ask if the line was necessary without the mine's load, TXU would say that the original infrastructure was adequate. Technically/politically it would be considered to be a white elephant.
- notwithstanding what has been said in the point above, if the line was left there without a major customer it would provide benefits and options for stiffer more reliable supplies and opportunities for back up supplies. It is a fact that 66kV lines running through the same corridor as a 22kV line have only 50% of the faults/outages due to animals, tree branches etc. The line in being there would provide benefits which over time would be utilised, but the benefits are not sufficient to justify it having been run in the first place.

Andrew

TXU

TXU Networks Pty Ltd
ABN 27 075 826 881
Level 14, 452 Flinders Street
Melbourne Vic 3000
Tel: 03 9229 5923
Fax: 03 9229 6297

12 OCT 2001

21 September 2001

Andrew McDougall
Austminex
PO Box 339 Collins Street West
VIC 8007

Dear Andrew

Re: COSTS TO PREPARE ESTIMATE FOR BENAMBRA MINE PROJECT

Further to your request to "provide me with the cost to Austminex for TXU to establish a definitive (+/- 10%) capital cost estimate for the installation of a system suitable for meeting our full power requirements", I advise the following.

Subject to the reservations and disclaimer we have set out below, we advise that the costs to prepare this estimate in accordance with the following considerations, is $133,000 plus $13,300 GST;

1. As indicated to you, we propose to rebuild the existing 22,000 volt line to 66,000/22,000 volt line and establish a Zone substation at a location which will meet the requirements of the proposed mine (eg Benambra town) as described by John Callaghan in his email of 19 July 2001. Although we anticipate that much of the line will be rebuilt along the existing pole line, the major determinants in developing a cost estimate at this stage are the approvals, including easements, for the route of the line.
2. The estimate will not be based on an actual survey. It will be based on the design and associated estimate by an experienced line design engineer.
3. The majority of these costs are associated with the time taken in negotiations to obtain easements and the most critical government approvals although not all approvals.
4. As you are aware planning approval for major works can take many years (as was the case recently for TXU in obtaining approvals for the underground cable from Mt Beauty to Mt Hotham). As a result we believe that it will take a minimum of 13 weeks to obtain some degree of certainty as to the route selection of the line.

5. If we have not been able to gain the appropriate level of certainty in our negotiations, and the monies received from you have been fully expended, TXU Networks will require to further invoice your Company in order to complete the work. The cost from that stage forward will be the actual cost to TXU Networks.
6. As an alternative for your consideration, TXU has received a quote of $475,000 plus $47,500 GST, for the full survey other than legal fees and costs to register the easements. These fees and costs would be additional.

   If the proposal in point 5 above is not acceptable to you, we are prepared to investigate this option further.

As indicated above the estimate could not be prepared earlier than a 13 week period following payment of the money.

If ultimately you choose to proceed with electricity supply via TXU's network, the works proposed in this letter will offset some of the future works and therefore some of the costs.

If you wish to proceed with the approach above, please contact Max Rankin of 03 5760 2480 who will arrange for a tax invoice to be sent to your Company. The offers contained in this letter will remain open until 31 October 2001. If you wish to proceed on any of the bases set out in this letter, then we shall prepare and forward a contract for execution by your company.

IMPORTANT NOTE & DISCLAIMER
The above estimates of costs and conditions have been prepared in good faith but projects of this nature are undertaken infrequently and on a one off basis. As a result budget costs/estimates are subject to large variations, and accurate estimates can only be undertaken following detailed discussions with affected land owners and government Departments. TXU takes no responsibility for any action taken or not taken as a result of the information set out in this letter.

TXU Networks has compiled this Estimate upon the information available to it as at the date of the Estimate. Nothing in this letter constitutes a legal offer by TXU Networks to contract with your company or otherwise deal with your company in relation to the matters contemplated in this letter, or creates any legal or financial liability on the part of TXU Networks. TXU Networks reserves the right to alter the Estimate at any time.

TXU Networks shall not be liable for any loss damage claim or demand incurred either directly or indirectly resulting from any act or omission which was made in reliance in whole or in part upon the Estimate.

Yours sincerely

Larry Westney
Manager Operations
TXU Network

74017241

Fax Cover

TXU

TXU Pty Ltd
ACN 086 014 968

Level 19
IBM Tower
40 City Road
Southbank VIC 3006

Date:	24 April 2001		
To:	Andrew McDougall Project Manager Austminex NL	Fax:	03 9620 2147
From:	Geoff Duke	Phone:	03 9299 2702
Dept:	I & C Contracts	Fax:	03 9299 2777
Re:	Indicative pricing proposal		

Andrew

Please find following some indicative quotations that can be used for budget estimations for the electricity costs based on the information you supplied to me re: the Benambra Project.

I am still waiting on the network guys to advise me who is following the network estimates up.

If you need to discuss further please let me know.

Regards

Geoff Duke

CUSTOMER CAN NOT HAVE OWNERSHIP OF FACILITIES ON PUBLIC LAND. - CUSTOMER PROVIDES THE CAPITAL BUT TXU RETAIN OWNERSHIP OF THE LINE.

ENERGY CONTESTABLE / HIGH CONTESTABLE METERING

NETWORK/GENT CHARGES REGULATED.

CONTESTABLE CONSTRUCTION ACCORDING TO TXU SPEC.

TXU PROVIDES CENTRALIZED TIGHT NEG 3 YR HOME MEAN TO CONTRACTORS

RUN TENDER PROCESS, EVALUATE & MANAGE CONTRACTS

Number of pages (include cover page) 3

24. Apr. 2001 — 11:12 — -130 01 3 92002770-

COMMERCIAL IN CONFIDENCE

INDICATIVE ONLY

TXU

TXU ELECTRICITY LTD PRICING PROPOSAL (Page 1)

Customer Name: Austminex NL
Site Address: Benambra Project
Omeo

Supply Period: 1 Jul 2002 - 30 Jun 2006

Host Distributor: Eastern Energy
Account No: Budget Estimate Only

RETAIL CHARGES

Energy Charges

	Energy	Rates			$ pa
		Net Rate	Loss Rate	Total Rate	
Peak - 7am-11pm, Mon-Fri	13,992,000 kWh	7.963 ¢/kWh	0.392 ¢/kWh	8.354 ¢/kWh	1,168,900.00
Off-Peak - All other times	15,158,000 kWh	2.896 ¢/kWh	0.099 ¢/kWh	2.995 ¢/kWh	453,966.00
Total Energy Charge	**29,150,000 kWh**				**1,578,566.00**
	Distribution Loss Factor 2.41%			Transmission Loss Factor	0.00

4 YR FIXED RATE

5.4¢/kWh.

Metering Charges

	Meters	Rates	$ pa
Data Forwarding	1 meter @	0 $ p.day	0.00
Provision Metering & Connect	1 meter @	0.9863 $ p.day	350.00
Total Metering Charge			**350.00**

Total Retail Charges 1,578,916.00

Retail Charges shall be adjusted by CPI on 1 January 2003 and each subsequent year of the Contract Period

REGULATED CHARGES

Network Charges

Refer to Page (2) for Details

Total Network Charge 515,132.40

1.77 ¢/kWh.

NEMMCO & Gov. Charges

	Energy	Rates	$ pa
Ancillary Services	29,150,000 kWh	0.1428 $/kWh	41,668.00
NEMMCO Op. & Establ.	29,150,000 kWh	0.0037 $/kWh	1,078.00
NCCA Op. & Establ.	29,150,000 kWh	0.0003 $/kWh	87.00
VIC Smelter Reduction Levy	29,150,000 kWh	0.2312 $/kWh	67,396.00
Total NEMMCO & Gov. Charge			**110,129.00**

0.38 ¢/kWh.

Total Regulated Charges 625,261.40

TOTAL COST OF PROPOSAL (excludes GST) 2,204,177.40
(includes GST) 2,424,595.14

7.56 ¢/kWh.

This Proposal is made in good faith based on information provided to TXU ELECTRICITY LTD and is valid until 07-May-2001 subject to confirmation by TXU ELECTRICITY LTD following your acceptance.

Remember:
1. Network and NEMMCO Charges are as Regulated.
2. Network Charges based on actual rates for 2000-2001
3. Actual costs will depend on energy consumption during contract period

Note: The information contained in this document is confidential and disclosure of its contents to third parties is prohibited.
CPI has not been included for years beyond 30/06/2003. Costs beyond this period will need to be adjusted for CPI for each subsequent year.

20 Apr 2001, 11:2

Network D.O. Sequence No: 4

* INDICATIVE ONLY *

TXU

TXU ELECTRICITY LTD PRICING PROPOSAL (Page 2)

Customer Details

Name	Austminex NL
Address	Benambra Project
Suburb	Omeo
Postcode	3,895
State	VIC
Account No	Budget Estimate Only

Customer's General Network Distributor Information

Network Distributor Name	TXU
Network Distributor Code	NEE61
Network Distributor Tariff Type	High Voltage Tariffs High Voltage Demand KVA

Customer's Actual Consumption Data

Flat 1	0	kWh
Flat 2	0	kWh
Total Flat	0	kWh
Summer Peak	n/a	kWh
Non Summer Peak	n/a	kWh
Peak 1	13,992,000	kWh
Peak 2	n/a	kWh
Peak 3	n/a	kWh
Peak 4	n/a	kWh
Total Peak	13,992,000	kWh
Total Off-Pk	15,158,000	kWh
Total Consumption	29,150,000	kWh
Actual Demand	4,210	kWhpa or kVAhpa
Chargeable Demand	4,210	kWhpa or kVAhpa

Customer's Special Network Charges

	TOTAL RATE	UNIT RATE
Standing Charge	3,215.40 $/pa	3,215.4000 $/pa
Flat 1	0.00 $/pa	0.0000 c/kWh
Flat 2	0.00 $/pa	0.0000 c/kWh
Total Flat	0.00 $/pa	n/a c/kWh
Peak Time Definition	7.00 A.M. to 11.00 P.M. Mon - Fri	
Off Peak Time Definition	All Other Times	
Summer Peak	n/a $/pa	n/a c/kWh
Non Summer Peak	n/a $/pa	n/a c/kWh
Peak 1	206,102.16 $/pa	1.4730 c/kWh
Peak 2	n/a $/pa	n/a c/kWh
Peak 3	n/a $/pa	n/a c/kWh
Peak 4	n/a $/pa	n/a c/kWh
Total Peak	206,102.16 $/pa	1.4730 c/kWh
Off-Pk	97,149.94 $/pa	0.6410 c/kWh
Summer Demand Incentive Charge	n/a $/pa	n/a $/kVA/month or $/kVA/month
Rolling Peak Demand Interruptable Contract	n/a $/pa	n/a $/kVA/pa
Summer Demand Incentive Interruptable Contract	n/a $/pa	n/a $/kVA/month
Rolling Peak Demand Interruptable Contract	n/a $/pa	n/a $/kVA/pa
Demand	238,664.90 $/pa	56.6900 $/kWhpa or $/kVAhpa
Total Network Charge	515,132.40 $/pa	

Based on 22 kV Supply
6MW = 4200
[illegible]

Austminex N.L. Benambra

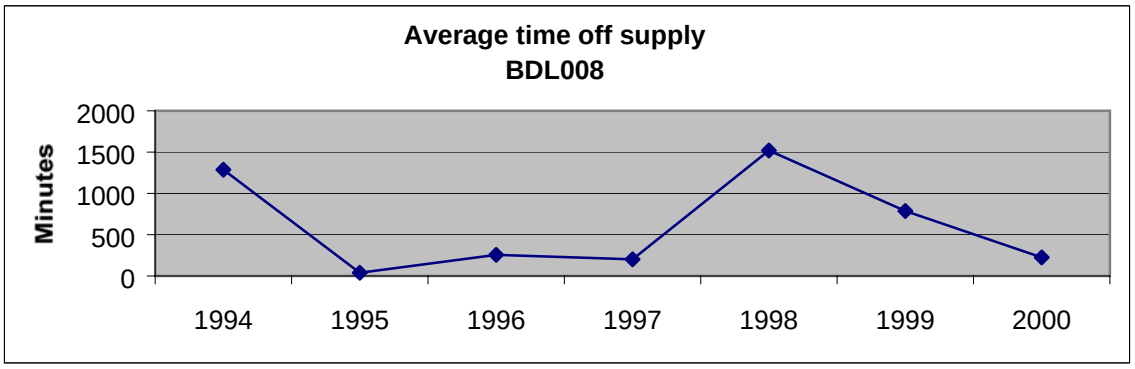
Date:	14 May 2001		
To:	Andrew McDougal- Austminex	Fax:	03 9620 2147
From:	Chris Websdale	Phone:	51539278
Dept:	TXU Networks Bairnsdale	Fax:	51521130
Re:	Proposed Benambra Mine		

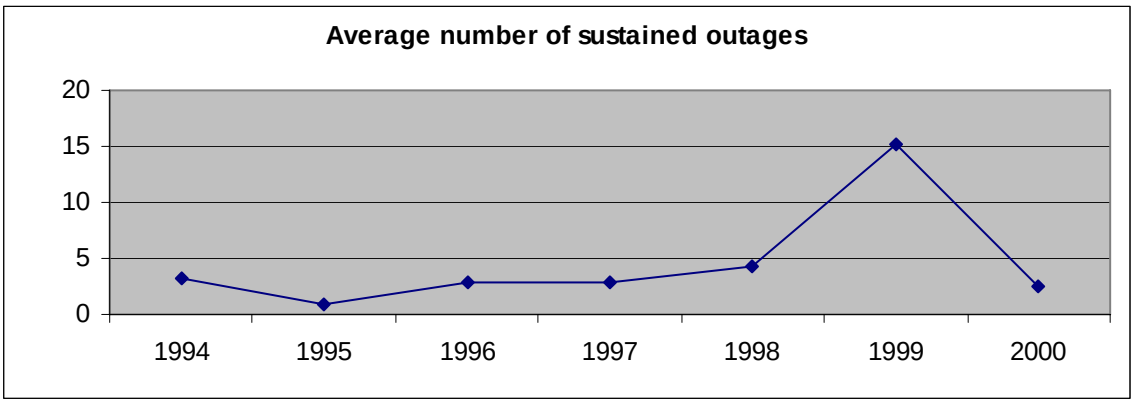
Andrew,

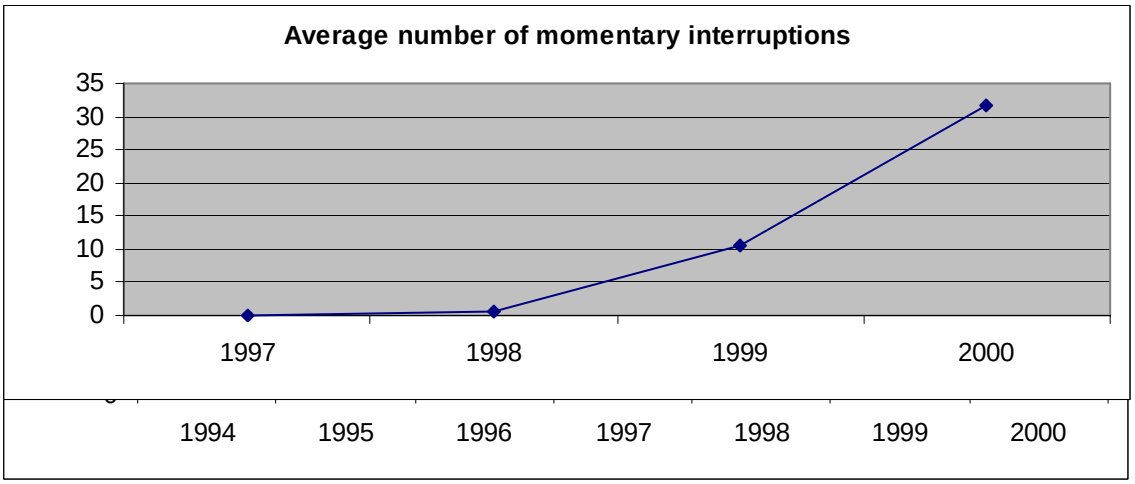
Further to discussion on the 18th April regarding your supply requirement to the Benambra Mine, I have attached a number of performance graphs for the current 22kv feeder that supplies the Omeo and Benambra area. The previous correspondence from Ross Ipenburg indicated that to meet your power requirement the construction of a 66kv line from Bairnsdale would be required. The performance of any future 66kv line from Bairnsdale to Omeo would be of a similar nature to the current 22kv feeder given the terrain and environmental conditions of the area.

A number of Network issues relating to the final demand and starting current characteristics of the Ball Mill must be provided and resolved before any future cost analysis could be undertaken. The capital cost of a new 66kv line following the existing 22kv feeder would be in the order of $20M. The contribution required from you will vary on your usage profile and final demand figure but could be as much as 90%-100% of the total capital cost.

Below is a number of performance graphs relating to the current 22kv line to Omeo







Monitoring of “momentary interruptions” only commenced in 1998 and thus there is little objective data on which to make meaningful comparisons. Nethereless TXU’s records indicate that there was an increase in momentary intereuptionsin 1999 and 2000

In relation to the current Distribution Use Of System tariffs that may apply will vary on whether you are supplied via 22kv distribution system or a 66kv sub transmission. The Tariffs are set out below:

High Voltage Tariffs
NEE81- High Voltage demand

Standing Charge	$3215.40/year
Peak Energy	$0.743c/KWh
Off Peak Energy	$0.443c/KWh
Contract Demand	$56.69/KVA/yr
Minimum demand	1150KVA

NEE95- Sub Transmission Demand
<25000KVA and >20km from Terminal Station

Standing Charge	$18412.20/year
Peak Energy	$0.261c/KWh
Off Peak Energy	$0.031c/KWh
Contract Demand	$8.65/KVA/yr

Minimum Demand 10500KVA.

The Tariffs set out above do not included any retail tariffs that you will incur and could only be estimated after a usage profile was forwarded from yourself.

If you require any further information in relation to the above matter, please do not hesitate to contact myself on 51539278 at the above office.

Yours faithfully

Chris Websdale

Senior Network Consultant

Bairnsdale

XC 7 B/, Att. 7F16

Minister for State and Regional Development

1 Treasury Place
Melbourne Victoria 3002
GPO Box 4379QQ
Melbourne Victoria 3001
Telephone: (03) 9651 6255
Facsimile: (03) 9651 0759
DX 210759

- 8 OCT 2001

Mr Kevin M Tomlinson
Managing Director
Austminex N.L.
PO Box 339
COLLINS STREET WEST 8007

10 OCT 2001

Dear Mr Tomlinson

AUSTMINEX N.L. - BENAMBRA PROJECT CONNECTION TO POWER GRID

I am writing in regard to recent meetings between representatives of Austminex and the Department of State and Regional Development relating to the Benambra project, its connection to the power grid and potential investment opportunities.

I have been advised that connection of the Benambra base metal project in Gippsland to the power grid would cost at least $20 million, with TXU prepared to cover only 5% of this. Austminex has asked the Department to provide up to 50% of the cost. I regret that this level of funding is not available.

There are other energy source options available to Austminex to consider, including biomass, diesel and brown coal briquettes. The Department has discussed these options with Austminex.

In April of this year, I understand that the Department introduced you to a Japanese company which is seeking to become a joint venture partner in a copper project in order to reduce its reliance on purchasing copper supplies on world spot markets. I understand that a confidentiality agreement has been signed and that further business discussions are under way.

The Government is committed to assisting companies establish in Victoria and will continue to work with you to find solutions acceptable to all stakeholders.

Please contact Mr Peter Dudzinski, Industry Specialist – Minerals, of the Department of State and Regional Development on 9651 9577 if you would like to discuss this matter in more detail.

Yours sincerely

JOHN BRUMBY MP
Minister for State and Regional Development

Victoria
The Place To Be

AUSTMINEX NL

Benambra Project
Contract No. 1344
Contract Power – Generation and HV Distribution

Request for Quotation No. 1344-31

Prepared for

Austminex NL

Prepared by

Metplant Engineering Services Pty Ltd
A.C.N. 009 256 508

Ref: 1344/AXD310/07

A	16.07.01	issued for tender	JC			
Rev	Date	Description of Revision	by	SM	DM	Client

CONTENTS*

*Note: This is the “contents” for the full specification which can be found in F:\Benambra\Process Plant Engineering\. It is provided in this document as an indication of the additional information which is available.

4. WORK SPECIFICATIONS

4.1 General

4.2 Design

4.3 Manufacture and Installation Standards

4.4 Substitutions

4.5 Quality Requirements

4.6 Expediting and Reporting

5. COMMERCIAL

5.1 General

5.2 GST

5.3 Payment Schedule

5.4 Damages

A1	ATTACHMENT ONE	-	Schedules
A2	ATTACHMENT TWO	-	Documentation
A3	ATTACHMENT THREE	-	Technical Data
A4	ATTACHMENT FOUR	-	Equipment Data Sheet Questionnaire
A5	ATTACHMENT FIVE	-	Commercial Terms and Conditions

Quotation requirements

PURPOSE OF THIS REQUEST FOR QUOTATION

The purpose of this Request for Quotation is to outline details of the Purchaser's requirements to enable interested parties to prepare a quotation for the supply of contract power:

- Power station – nominal 7 MW complete, one (1) off
- Plant high voltage distribution system – complete, one (1) off

FIRM PRICE

The Supplier is to provide a firm and fixed price to supply contract power initially for a contract period of five years with an option to extend for the following three years and then a further two years. The price shall not be subject to rise and fall or change due to exchange rate variations, or variations in taxes and duties.

The price shall consist of :

- A fixed monthly fee (The Facility Fee); plus;
- A consumption fee (The Energy Fee)

The cost shall include all activities, components, man-hours, design, documentation, supply, fabrication, surface treatment, inspection, assembly, shop testing, packaging, delivery, offloading, construction, installation, commissioning, operational running, servicing, maintenance, supply of parts, reporting and warranty.

VALIDITY

The pricing shall remain valid for a period of sixty (60) days.

SCHEDULES

The Supplier shall complete all schedules contained in Attachment One.

COST OF QUOTATION PREPARATION

All costs incurred by the Supplier in the preparation of the quotation shall be borne solely by the Supplier.

LODGEMENT OF QUOTATIONS

The quotations shall be enclosed in a sealed envelope endorsed with the Quotation number, name of the Supplier, general description of the works and addressed to the attention of Mr John Callaghan, Electrical Engineer, Metplant Engineering Services Pty Ltd.

Quotations shall be lodged at the offices of the Engineer by 1430 hours, Friday, 27 July 2001. The Quotation shall be deposited in duplicate. The duplicate Quotation shall be a complete copy of the original Quotation.

COMPLETION TIME AND PROGRAMME

The Supplier shall submit with its quotation, a time based network or bar chart program in sufficient detail to distinguish the significant elements of the work and the various key dates along with such explanatory notes as may be necessary to identify anticipated problems and restraints.

TENDER INFORMATION

The Supplier shall obtain all necessary information that may affect the Quotation. The Supplier shall obtain clarification for any aspect of the Request for Quotation documentation about which the Supplier requires more information.

The Supplier may seek further information from the Purchaser's representative. All queries shall be in writing and referred to Mr John Callaghan on facsimile 08 9479 3478 or e-mail "johnc@metplant.com.au".

All requests for information and responses provided will automatically be distributed to all other interested parties registered under the project.

DELIVERY

The Supplier shall quote its best delivery in calender weeks.

POINT OF DELIVERY

All equipment and construction materials shall be installed and erected at the designated location for the power station which is approximately 250 metres from the existing main plant switch room at the Benambra project site, Victoria.

ACCEPTANCE OF QUOTATIONS

The Purchaser shall not be bound to accept the lowest of any Quotation and reserves the right to accept or reject any Quotation in whole or in part. A contract shall be formed on issue of the Purchaser's order number accepting the Supplier's proposal or on issue of a Notice of Award if this is given prior to the issue of the purchase order.

CONFORMING AND ALTERNATIVE QUOTATIONS

The Supplier's base Quotation shall conform in all respects with this specification. Suppliers may offer alternatives provided all deviations and exceptions are listed separately and are clearly defined.

If the Supplier does not provide deviations and exceptions, it is deemed that the Supplier has conformed with all aspects and requirements of the specification.

QUALITY ASSURANCE

The Supplier shall supply with its Quotation details confirming accreditation of its Quality System or alternatively, information detailing the progress it has made towards implementing and ultimate accreditation of its Quality System.

SCOPE OF WORK

GENERAL

The scope of this specification includes all activities, components, man-hours, design, documentation, supply, fabrication, surface treatment, inspection, assembly, shop testing, packaging, delivery, offloading, construction, installation, commissioning, operational running, servicing, maintenance, supply of parts, reporting and warranty of the following:

- Power station – nominal 7 MW complete, one (1) off
- Plant high voltage distribution system – complete, one (1) off

All equipment and materials shall be supplied with all standard tools and accessories. All miscellaneous material, minor parts and other such items shall also be supplied whether or not the items are indicated on the drawings or in the specification, and where it is normal industry practice that they be supplied to complete and to operate the equipment.

The equipment shall be supplied in accordance with Section 4.0 "Work Specifications" of this specification.

INCLUDED IN SUPPLY

- One (1) power station building complete with services and concrete floor and external concrete pad areas for equipment located external of the building;

- Seven (7) nominal 1MW engine generator units selected to meet the power requirements of the plant. This quantity includes for a maximum of five units on-line to service the average maximum demand and an allowance for two units unavailable off line.

- One (1) group of ancillary plant to suit, including; exhaust systems, engine cooling and circulation systems, lubricating oil storage and make up systems, fuel storage and transfer systems, engine starting systems, air intake systems, piping, hoses, valves, filters and fittings and base-plates as required;

- One (1) 3.3kV power station switchboard

- One (1) 3.3kV / 415V power station auxiliary power transformer

- One (1) 415V power station auxiliaries MCC

- One (1) power station control and monitoring system

- One (1) 3.3kV plant switchboard;

- One (1) 3.3kV plant ring main unit;

- Four (4) plant distribution power transformers.

- One (1) group of interconnecting cables, including under ground 3.3kV feeder cables to the plant area.

- Ancillary items normally supplied with this type of equipment inclusive of any special tools required for installation, maintenance, protection or operation.

- One (1) group of operational and maintenance spare parts.

- One (1) group of consumable spare parts.

- Connections from the power station to the plant area for telephone, PLC and data interconnections, fire alarm.

- Documentation.

- Dual redundant metering equipment to measure fuel consumption and electrical energy as supplied to the Purchaser after deduction of all transfer and parasitic loads.

- Metering equipment to measure energy produced and the fuel and oil consumption of individual units.

- Temporary test loads and connections for commissioning of the power station and for regular equipment maintenance checks.

- Removal from the site of all waste oil products and other generated waste originating from; or as a result of; the construction, operation and maintenance of the equipment.

- Arranging for the timely delivery of all fuels and oils for the equipment on behalf of the Purchaser.

- The supply and training of all staff necessary to operate and maintain the equipment.

EXCLUDED FROM SUPPLY

(a) Site preparation earthworks.

(b) Three (3) 50,000 litre interconnected diesel fuel bulk storage tanks

(c) One (1) fabricated steel elevated 2000 litre diesel fuel storage day-tank. This unit is currently available at the site if the supplier wishes to incorporate it into his design.

(d) Cost of diesel fuel and lubricating oils for the operation of the power station.

TERMINAL LIMITS

(a) Top of the earthwork. Supplier to do their own compaction testing if needed.

(b) Outlet of the diesel fuel bulk storage tanks. These tanks are located in an elevated position approximately 120 metres of the power station.

(c) Outgoing terminals of the 3.3kV plant switchboard starters.

(d) Incoming and Outgoing 3.3kV terminals of the existing 3.3kV starter to be refurbished by others.

(e) Outgoing 415V terminals of the 3.3kV / 415V plant distribution transformers.

WORK SPECIFICATIONS

GENERAL

The Supplier shall be responsible for and shall make allowances for all work and materials not specifically detailed or described but necessary to enable the work to be completed in a sound and workmanlike manner in compliance with the intent of the specification.

The Supplier shall obtain all necessary permits and pay all fees required by statutory authorities in respect of the works.

Where it is required, by the Occupational Health, Safety and Welfare Act or Regulations thereunder and/or any other act or ordinance relevant and applicable to the contract works, that personnel be certificated the Supplier shall ensure that personnel carrying out these works are certificated and shall keep copies of their certification for inspection by the Purchaser.

Approval of documents by the Purchaser does not relieve the Supplier from his responsibility to provide fit for the purpose and safe construction and to ensure the safety of all personnel during the performance of the contract works.

DESIGN

Introduction

The equipment shall form an integral part of the Benambra Copper-Zinc upgrade Project.

Duty Description

The power station and high voltage distribution system is to be considered a continuous operating system as it represents the sole source of electrical power to the project plant site.

Design Criteria and Alternative Tenders.

The Supplier may offer alternative non-conforming tenders for consideration but these must be fully detailed and the advantages to the Purchaser must be clearly identified

The equipment shall be designed such that the guaranteed availability and operational performance shall be adequate to meet the power requirement data detailed in Attachment Three.

The power station will at all times run sufficient generation to cater for the event that a single unit may become faulted and automatically be removed from the bus. The remaining units must be able to carry the increased unit loading until a replacement unit can be placed back on line.

Technical Data

The equipment shall conform to Technical Data contained in this schedule.

Total Connected Load	6,541 kW
Maximum Demand (5 min)	5,325 kW
Maximum Demand Average	4,452 kW
Annual Energy – Crushing 6,000	1,695,125 kWhr
Annual Energy – Rest of Plant	33,872,955 kWhr
Annual Energy – Total	35,568,080 kWhr
Emergency Power Demand	79 kW

Power Station Operation

Hours per year	8,760
Plant Availability (%)	100%
Design Life (years)	10

[redacted]

From: Terry, Graham SAL- OCMB1 [Graham.J.Terry@Shell.com.au]
Sent: Monday, 15 January 2001 2:10 PM
To: 'andrew mcdougall'
Subject: Fuel Pricing

Andrew

Further to our discussions this morning I list below estimated price movements to the December 2000 base price calculated on the values you provided.

These estimates are reductions to the base price of 53.06 aust cpl provided for tri ade deliveries in Dec 2000.

Crude	High $US25.00/bbl	Average$US20.20/bbl	Low$US17.90/bbl
Exchange Rate	0.7250	0.6770	0.6400
Price Redn Aust-cpl	13.30	16.20	17.40

I am currently seeking Shell's estimates and will advise when these are available.

Regards
Graham Terry
Ph 03 9666 5124
Fax 03 9666 5174
Mobile 0418 364 506
Email : Graham.J.Terry@shell.com.au

PRICE FOR DIESEL @ BENAMBRA BASED ON OPEC CRUDE OIL PRICES OF:

AUST ¢/L

$US/bbl

25 (LONG TERM-HIGH) (EXCH. RATE 72.5 ¢/A$) → 53.06 LESS 13.3 = 39.76

20.20 (LONG TERM-AV.) (EXCH. RATE 67.7 ") → 53.06 LESS 16.20 = 36.86

17.40 (LONG TERM-LOW) (EXCH. RATE 64.0 ") → 53.06 LESS 17.40 = 35.66

SHELL INDICATE $US25/bbl FOR NEXT 1 TO 2 YEARS & AS LOW AS $US18/bbl LONG TERM

LONG TERM DIESEL AUST ¢/L

39.76 (HIGH)
36.86 (AVE)
35.66 (LOW)

ELEC COST AUST ¢/KWH.

18.35 (7532 HOURS + 0.2224 KWH × 39.76)
17.56 (" " " " × 36.86)
17.23 (" " " " × 35.66)

[redacted]

From: Terry, Graham SAL- OCMB1 [Graham.J.Terry@Shell.com.au]
Sent: Monday, 15 January 2001 2:10 PM
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SHELL INDICATE $US25/bbl FOR NEXT 1 TO 2 YEARS & AS LOW AS $US18/bbl LONG TERM

LONG TERM DIESEL AUST ¢/L

39.76 (HIGH)
36.86 (AVE)
35.66 (LOW)

ELEC COST AUST ¢/KWH.

18.35 (7.532 ROUT + 0.2224 KWH × 39.76)
17.56 (" " " " × 36.86)
17.23 (" " " " × 35.66)

Generator Rentals PTY LTD
ABN 18 094 046 890

P.O. Box 290
Archerfield QLD 4108
Phone: (07) 3277 5886
Fax: (07) 3277 6227
Email: genrent@powerup.com.au

30 March 2001

Mr Andrew McDougall
Austminex N.L.
15 Queen Street
Melbourne Vic 3000

Fax: (03) 9620 2147

Dear Andrew

Re: Benambra Project Power Supply

We thank you for your facsimile yesterday regarding the abovementioned project. It is with pleasure we forward the following budget submissions for your appraisal.

OPTION 2

600,000 T.P.A. PROJECT

PARAMETERS APPLIED

Client: Austminex N.L.

Power Equipment (Peak)	4000 kw
Base Load Requirement	3585 kw
Base Charge Minimum per month	2,429m166 kw / hrs
Largest Motor Start (soft)	1500 kw
Site	Via OMEA Victoria
Annual Consumption	29.150 G/W/HR

BUDGET OFFER

Price 3.95c kw/hr	$95,985 per month
Fuel Consumption guarantee	.258 ltr/kw/hr
Base charge level per month	2,430,000 kw/hrs
Base charge rate c/kw/hr	3.95
Minimum contract period	60 months
Spare Capacity above base load	

Power Cost /KWH
0.258L × 36.94 + 3.95
= 13.5¢

EQUIPMENT AND SERVICES

8 x 925kva Cummins / Stamford Generators (QST)
8 x remote variable speed closed loop cooling systems
2 x Power Station feeder and distribution system
3 x 415v / 3.3kv Transformers
1 x Power Station building insulated
8 x day tank fuel systems
1 x Power Station waste recovery system
1 x Power Station workshop and parts store
1 x Air-conditioning plant
1 x on site maintenance technician
1 x 415 – 11kv transformer amp
1 x on site service vehicle

For 33.33¢/L on site
Diesel cost
Power Cost / KWH
0.258L × 33.33 + 3.95
= 12.55¢/KWH

Diesel Cost on site 30/05/01
87.09 Less 6.28 Less 1.0 + 3.10(Freight) Less 38.14 (rebate)
= 44.77 ¢/Litre
Power Cost = 0.258 × 44.77 + 3.95
= 15.50 ¢/KWH.

CLIENT SUPPLIED SERVICES

- Accommodation and meals for servicemen
- Diesel for on-site vehicle
- Fuel tanking and fuel supply by fuel supplier
- On-site induction courses

We trust you find our offer of interest and await your return correspondence.

Yours faithfully

Steven Clements
Managing Director

GENERATION - BID ANALYSIS

	Rise & Fall	Per Mth 0-5 yrs	Ranking	Accumulated Costs 0-5 yrs	0-8 yrs	0-10 yrs	Ranking	0-5 yrs
Contract Power Management Australia - No Automation	Fixed	$110,642	1	$6,638,499	$10,621,611	$13,277,019	1	$5,20
Contract Power Management Australia - Cummins Option	Fixed	$126,912	2	$7,614,749	$11,772,749	$14,544,749	2	$5,97
Energex - 10 year option	+ CPI	$133,337	3	$8,000,220	$12,800,352	$16,000,440	4	$6,27
Modra Electric Power - Deutz Option	+ CPI	$144,980	4	$8,698,800	$13,205,280	$15,807,000	3	$6,82
Modra Electric Power - Cummins Option	+ CPI	$158,652	5	$9,519,125	$13,646,600	$16,134,250	6	$7,46
Aggreko	+ CPI	$174,235	6	$10,454,125	$15,742,540	$19,268,150	8	$8,20
Energex - Balloon Option	+ CPI	$187,029	7	$11,221,740	$15,117,444	$16,717,524	7	$8,80
Energex - 5 + 3 + 2 year option	+ CPI	$200,047	8	$12,002,820	$14,402,940	$16,003,020	5	$9,41

Winning Bid	
2nd Place	
3rd Place	

Need to examine efficiencies to validate the above.

John Callaghan 30/7/01

SOUTHERN
Rural Water

Our Reference: 1000977

9 May 2001

11 MAY 2001

Andrew McDougall
Austminex NL
PO Box 339
Collins Street
MELBOURNE VIC 8007

Dear Mr McDougall

LICENCE TO TAKE & USE WATER – TAMBO RIVER UPPER

I refer to your application to transfer Licence No. 1000977 from Benambra Mines. I confirm that this transfer has now been effected and a new licence document is enclosed, for your records.

Please keep this licence in a safe place as it confirms your right to take and use water.

Licence No. 1000977 authorises you to take and use 30.0 megalitres of water per year from Tambo River Upper for Industrial purposes.

If you have any queries regarding this matter then please do not hesitate to call Narelle Proud, Licensing Officer, on (03) 5139 3143.

Yours sincerely

TREVOR McDEVITT
Manager Licensing Administration

d:\primary data\surface water\15 year diversion licences\letter - transfer of 1000977 (austminex nl).doc

PO Box 153 MAFFRA VIC 3860
Telephone: (03) 5139 3100
Facsimile: (03) 5139 3150

ABN: 70 801 473 421
Email: srw@srw.com.au
Website: http://www.srw.com.au

SOUTHERN RURAL WATER
WATER ACT 1989
Section 51 and 67

DIVERSION LICENCE No 1000977
(Licence to take and use water from a waterway and to operate works)

Gippsland and Southern Rural Water (The Authority) authorises:

Austminex NL
PO Box 339
Collins Street West
MELBOURNE VIC 8007

To take and use water from the waterway specified in the First Schedule and to operate works and subject to the conditions in the Second Schedule.

The licence is valid until 30 June 2015.

For the purpose of taking and using water under this licence the Licensee is authorised to install and operate works on Crown land forming the frontage to:

Lot(s)	-	Plan of subdivision No:	-
Allotment(s)	**Mining Lease 1865**	Section	**No Section**
Parish	**Thorkidaan (State Forest)**		

Trevor McDevitt
Authorising Officer
Date: 09/05/2001

FIRST SCHEDULE

1.	Waterway	**Tambo River Upper**	
2.	Type of Use	**Industrial**	
3.	Maximum Rate of Diversion	**0.10**	megalitres per day
4.	Maximum Daily Volume	**0.10**	megalitres
5.	Total Annual Volume	**30.0**	megalitres
6.	Area to be irrigated	-	hectares
7.	Land on which the water is to be used as outlined below.		

Lot(s)	-	Plan of subdivision No:	-
Allotment(s)	**Mining Lease 1865**	Section	**No Section**
Parish	**Thorkidaan (State Forest)**		

8. Annual fee at date of issue **$46.10 Licence Charge + $4.30 per megalitre of Licensed Volume**
9. Other Conditions

All communications should be addressed to: PO Box 153, MAFFRA VIC 3860

.d \primary data\surface water\15 year diversion licences\dl 1000977 (austminex nl) doc

SECOND SCHEDULE
CONDITIONS

1. The Licensee shall not -
   (a) take any water except -
      (i) by a method or methods approved by the Authority
      (ii) where a meter is provided, through that meter.
   (b) waste any water taken
   (c) irrigate an aggregate area greater than that authorised if the specified use includes irrigation and a meter is not provided
   (d) without the prior consent of the Authority interfere with, disconnect or remove any meter used for the purposes of this licence.

2. The Licensee shall -
   (a) pay charges under the licence when requested by the Authority.
   (b) maintain in efficient working order and to the satisfaction of the Authority all works and appliances required to use and take water under this licence.
   (c) when required by the Authority, keep an accurate record of the quantity of water taken under this licence and allow the Authority to inspect this record at all reasonable times and to provide a copy of the record when requested.
   (d) when required by the Authority provide a description of the area to be irrigated during any period.
   (e) when required give three days notice of the intention to take water

3. (a) Whenever the Authority is of the opinion that the taking of water under this licence may injure or adversely affect any other person, or the environment including the riverine or riparian environment, the Authority may give notice to the Licensee to -
      (i) reduce the quantity of water taken during such period or periods and to such extent as the Authority directs.
      (ii) cease taking water entirely or during such period or periods as the Authority directs.
   (b) Any notice or direction given under this clause shall be sufficiently given if it is -
      (i) forwarded by post to the Licensee at the address contained in this licence.
      (ii) given verbally to the Licensee by an Authority officer.
      (iii) published in a newspaper circulating generally within the district where the water is taken under this licence, or broadcast from a radio broadcasting station in the State.

4. The Licensee must not pollute the riverine or riparian environment through the spillage of fuel or lubricant or any other matter used in connection with works and appliances associated with this licence.

5. (a) The Authority may require the Licensee to use an approved meter or meters for the purposes of this licence and any such meter with all fittings shall be installed under the supervision of the Authority. The Licensee may be required to meet the Authority's cost of metering.
   (b) All meters used for the purposes of this licence shall be the property of the Authority.
   (c) When the taking of water is not measured by a meter, or when the Authority is satisfied that the taking of water has not been accurately measured by the meter used, the quantity of water deemed to have been taken shall be that as from time to time fixed by the Authority in a schedule of deemed depth of watering over land irrigated.

6. The Authority may from time to time authorise the taking of water to a greater extent than that specified in the First Schedule subject to the payment by the licensee of such further fee and in accordance with such conditions as the Authority specified.

7. The Licensee is not authorised by this licence to use Crown Land for any purpose other than that stated in the licence.

4svcond35

SOUTHERN RURAL WATER

OFFICIAL RECEIPT

Date: 2203401

Receipt Nr: 16105271

Received from: Alpenlakes NL

Tendered

Cash	[redacted]	Cheque	[redacted]	Other	[redacted]
		Receipt Total	[redacted]		

Detail

Amendment/transfer of ownership of a diversion licence	$36.00
Statement: 924075	$430.70

SOUTHERN

Notes

Diversion Licence Transfer from Denehurst Limited (Benambra Minis) to Austrinitx NL

[redacted]

SOUTHERN
Rural Water

14 SEP 2001

10 September 2001

AUSTMINEX NL
ATT: ANDREW MCDOUGALL
PO BOX 339, COLLINS STREET WEST
MELBOURNE VIC 8007

Dear Customer

FORMATION OF LOCAL AREA COMMITTEE - TAMBO RIVER

I am writing to you as a valued customer on the Tambo River system to inform you that Southern Rural Water (SRW) has now completed a survey of all surface water use (excluding stock and domestic) and now intends to form a Local Area Committee (LAC) for the area.

SRW is responsible for the management of surface water and groundwater within southern Victoria. We provide a range of services including licensing of both groundwater extractions and surface water diversions for agricultural, commercial and domestic interests. The formation of a LAC is an important step in our communication strategy with customers.

The aim of the committee is to provide advice on local management issues - such as rostering/restrictions, metering/monitoring and the application of general licensing policy.

Members of the committee have two key roles: to represent a group of stakeholders with an interest in the management of the waterway; and to ensure a regular transfer of information with their interest group. The committee will meet periodically until the Streamflow Management Plan Committee is formed. This is currently anticipated to be in June 2003.

If you are interested in contributing to the future planning and sustainability of surface water in your area and would like to take part in our committee and/or require more information please contact **Senior Licensing Officer Craig Parker on 0427 393109**. Applications close on 28 September 2001.

Yours sincerely

Colin McQuillen
Licensing Supervisor - East

PO Box 153 MAFFRA VIC 3860
Telephone: (03) 5139 3100
Facsimile: (03) 5139 3150

ABN: 70 801 473 421
Email: srw@srw.com.au
Website: http://www.srw.com.au

28-SEP-01 FRI 8:45 SOUTHERN RURAL WATER MAF FAX NO. 0513393150 P. 2

ATTN: ANDREW McDOUGALL

SOUTHERN
Rural Water

NOTE: WE HAVE NOMINATED 2 PEOPLE FOR YOUR RECORDS – A PRIMARY AND A SUBSTITUTE IN THE EVENT THAT THE PRIMARY IS AWAY AND NOT CONTACTABLE.

APPLICATION FORM PRIMARY: ANDREW McDOUGALL (OPERATIONS MANAGER)
SECONDARY: KEVIN TOMLINSON (MANAGING DIRECTOR)

TAMBO RIVER LOCAL AREA COMMITTEE

NAME: ANDREW McDOUGALL / KEVIN TOMLINSON PH: 03 9670 5111
AM(C): 0427 220 249
EMAIL: andrew.mcdougall@austminex.com.au MOBILE: K.T.: 0419 119 755
OR kevin.tomlinson@austminex.com.au
POSTAL ADDRESS: PO Box 339 COLLINS STREET WEST
MELBOURNE VIC POST CODE: 8007

OCCUPATION/BUSINESS AUSTMINEX NL – MINING INDUSTRY.

WATER INDUSTRY OR RELATED EXPERIENCE:
NONE

MEMBERSHIP OF RELEVANT ORGANISATIONS:
N.A.

REASON FOR WANTING TO BE ON COMMITTEE:
INTENTION TO RECOMMISSION AND OPERATE THE BENAMBRA MINE WHICH HAS A LICENCE TO DRAW WATER FROM THE UPPER TAMBO RIVER AND ITS TRIBUTARIES

PLEASE ADD ADDITIONAL PAGES IF REQUIRED.

Please direct applications to:
Colin McQuillen,
Po box 153,
MAFFRA 3860

Enquiries to:
Craig Parker
0427 393 109

PO Box 153 MAFFRA VIC 3860
Telephone (03) 5139 3100
Facsimile (03) 5139 3150

ABN: 70 801 473 421
Email: srw@srw.com.au
Website: http://www.srw.com.au

BENAMBRA PROJECT

Exploration Opportunities

Summary Report

Rod Paterson (Consulting Geologist/Geophysicist)

17 December 2001

Austminex N.L. Benambra

Contents:

Figures:

Summary:

- A review of all available data was completed by a group of three consultants with the objective of defining drillable targets wworthy of drilling for massive base metal sulphide potential.

- Peter Rea identified and prioritised 23 EM, magnetic and geochemical exploration targets and recommended work programmes.

- Llew Wynn examined all electrical geophysical data and identified anomalies and untested DHEM targets for investigation.

- Critical IP, EM, and some soil geochemical and drill hole data was captured in digital form and a GIS database constructed using ERMapper & Mapinfo/Discover. Data was validated for accuracy of both content and location where possible.

- Aeromagnetic and radiometric data from three separate surveys was levelled, merged and regridded.

- Soil geochemical data was transformed and standardised and a number of supervised and unsupervised classification techniques used to locate geochemical patterns similar to those recognised at Wilga & Currawong.

- A prospectivity analysis of the GIS data was completed using Wilga & Currawong characteristics. The ingredients given most weight were in order EM, geochemistry, magnetics and geology.

- All possible targets identified from the above work were subject to technical review on site at Benambra.

- Eight exploration targets were identified for immediate follow up.

- Ray Hazeldene prepared a costed work programme for Quarter 4.

- The Benambra tenements present a very prospective VMS terrain with many targets remaining to explore.

Introduction:

This report summarises exploration targets identified within the Benambra exploration tenements from October 2000 to December 2001 by geological and geophysical consultants and staff working for Austminex N.L..

The tasks completed were:

- A review by Llew Wynn of all electrical geophysics conducted by WMC, Macquarie Resources and Denehurst Ltd.

  Llew recommended DHEM be conducted on a number of old and more recent Austminex drill holes to assist with the identification of extensions to existing resources.

  EM37 data completed along the main Wilga-Currawong corridor and at Banksia was reinterpreted and anomalies selected and ranked (Fig. 14).

  Llew's work is summarised in various memos to Austminex N.L. (Appendix 1).

- A review of all past exploration data, reports and recommendations by previous workers was conducted by Peter Rea at the Benambra site in Feb/Mar 2001. Peter focussed on soil geochemistry, EM, magnetics and drilling.

  He recorded the coverage of WMC Moving Loop EM, compiled past IP and MLEM records for digitising and identified and prioritised targets for follow-up, Fig. 11.

  Peter Rea's recommendations are outlined in a memo to Austminex 16 April 2001, (Appendix 2).

- The author preprocessed, compiled and validated all available company and government exploration data into a MapInfo GIS database. An analysis of prospectivity was carried out using the known deposit features. A variety of image enhancements designed to highlight regional and local targets were prepared for prospectivity analysis and for company presentation purposes.

  The data compiled includes aeromagnetic-radiometric surveys by Macquarie Resources and VIMP (Victorian Government Initiative For Minerals & Petroleum); ground EM and IP by WMC, Macquarie and Denehurst; exploration drill hole data which was added to resource data,WMC and Denehurst soil and rock geochemistry (partly compiled by TerraSearch for VIMP); WMC soil geochemistry to the NW and SW of Wilga which was missing from the latter and was scanned and OCR'd from old WMC computer printouts; 1:10K geological mapping carried out by R. Allen and 1:10K topographic map sheets scanned for use as transparent overlays.

  A detailed analysis of soil geochemical data using maximum likelihood supervised and unsupervised classification techniques was carried out in an attempt to highlight other targets with a Wilga, Currawong type signature. Ratio, log & log-ratio transformations and standardisation procedures were used to prepare the data for analysis. This highlighted a number of subtle features but was not helpful in reducing the number of prospective geochemical targets. Examination of the Cu, Pb, Zn data in RGB colour space proved helpful in highlighting unique spatial patterns associated with the existing Wilga and Currawong resources, (Figs. 9, 10). These patterns are discussed in more detail below.

Minimum Target Specifications:

A massive sulphide deposit of the Wilga-Currawong style with a resource potential of greater than 3 million tonnes at Wilga grade.

And/Or

Extensions to known resources of sufficient size to allow economic exploitation.

Target Indicators:

The most important target identifiers and ranking parameters used in this study in order of priority were:

- Moving Loop or Fixed Loop surface or DHEM late time conductors or low resistivity IP anomalies, (Figs. 3, 4)
- Anomalous Cu, Pb, Zn soil geochemistry with the highest ranking given to the typical multi-element Wilga-Currawong signature, (Fig. 10).
- Discrete positive magnetic features similar to those found at Wilga and Currawong, (Fig. 1).
- Favourable stratigraphic and structural positions above the Thorkidaan rhyolitic volcanics within the Gibson's Folly dacite, andesite, sediment package or the less favourable but as yet unproductive Cowombat siltstone, (Fig. 12.)
- Proximity to existing mine infrastructure.

Target Indicators - Diagnostic Features:

Wilga and Currawong are very strong MLEM late time conductors within a mostly highly resistive 1000 ohm-m background (Gibson's Folly Formation). The Cowombat siltstone is less resistive (500 ohm-m) compared with the above and produces a moderate IP effect, (Fig. 5).

The Wilga and Currawong soil geochemical expression is high Cu, Pb and med-high Zn restricted to a 100-200m strike length at the expected outcrop position of the main lenses.

Associated with this expression both deposits show a Zn dominated anomaly to the NW above the down dip projection of the main lenses. This is clearly evident in Fig. 10 as a purple zone (Zn with subordinate Pb) which at Wilga correlates closely with the subsurface extent of the main lens. Hence a non-outcropping massive sulphide lens might have a zinc dominated surface geochemical expression.

There is a possibility that the zinc-dominated features represent hydromorphic dispersion of the main outcrop anomalies down slope from the source. However at Wilga the almost perfect fit of the zinc dominated feature with the subsurface areal extent of the Wilga lens suggests a more primary origin.

The small extent of the geochemical expression of the known massive sulphides makes target focus difficult considering the large amount of Cu, Pb, Zn anomalous geochemistry in the district.

More blanket EM coverage of anomalous geochemistry within prospective stratigraphy may lead to other discoveries (see Longer Term recommendations).

Immediate Targets:

Targets are described below in order of merit. Further target details are summarised in Table 1.

1. DHEM Anomaly - BEND 77

A WMC DHEM anomaly in drill hole BEND77 NW of Currawong is interpreted to be caused by a body ~100m beyond the end of hole. The anomaly is discussed by Llew Wynn in Appendix 2 where modelling results for the most likely source geometry are presented. A memo by Ray Hazeldene 5 October 2001 in Appendix 3 adds further background information and estimates the cost of follow up.

A separate massive sulphide lens below the end of hole and above the Cowombat siltstone member is possible. Modelling by Llew Wynn indicates that Currawong M-Lens does not make a significant contribution to this anomaly.

The surface geological, geophysical and geochemical setting for this area is illustrated by Figs.10,14..

The target is recommended for drill testing by extending B77.

2. Currawong ESE - MLEM Anomaly

A 2 station WMC MLEM anomaly that is incompletely defined and is outside the more recent EM37 coverage, (Figs. 3, 13). This is the best, untested MLEM anomaly on the property (Ch10 [5.8ms] 5-10mV/A) - except Target 8 which is inside the National Park. P.Rea, Appendix 2 mentions that one of the above readings was taken within a loop containing pumps and other metal on surface - this needs to be researched to determine what effect this might have.

Rhyolitic Thorkidaan lavas occur at surface and the faulted eastern contact of the Cowombat siltstone occurs 200m west. Dips are to the NW.

Anomalous Pb with subordinate Zn occurs in soils along this faulted contact for 2 kms, (Fig 10).

There is no associated magnetic response. However not all Currawong lenses are magnetic.

It is recommended that the anomaly be detailed with fixed loop EM37 and followed up by drilling.

3. Currawong South (Plant) - MLEM Anomaly

A single station WMC MLEM anomaly (10mV/A Ch7 [3.7ms]) occurs 70 metres south of the crusher plant (survey predates plant). The anomaly is just outside the more recent EM37 coverage, (Fig. 3).

A magnetic anomaly defined by Macquarie Resources and VIMP aeromagnetic surveys may be related. The VIMP survey is post plant construction but the relevant flight line passes 50m north of the plant. Ground magnetics by WMC indicates an anomaly in the creek near the EM feature, (Fig. 18).

Drill testing of the magnetic anomaly defined by the Macquarie Resources survey was undertaken some 500m along strike to the WSW (BEND 182). A weak to moderately magnetic dacite was intersected.

The Cowombat siltstone occurs up dip from the MLEM anomaly assuming dips of 50-70 degrees to the NW.

Weak Zn, Pb geochemistry occurs in the Cowombat siltstone in this area, (Fig. 10).

Ground magnetics and fixed loop EM37 are required to define a drillable target.

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4. WilgaWong - MLEM Anomaly

A single station WMC MLEM anomaly occurs 1 kilometre SSW of the plant outside EM37 coverage (9 mV/A Ch10 [5.8ms]), (Figs. 3, 13). The anomaly is open to the south.

A strong Cu, Pb, Zn soil anomaly is located up dip in the Kookaburra porphyry 300m to the SSE, (Fig. 10).

The magnetics is featureless over the target.

Fixed loop EM37 is recommended to define a drillable target.

5.1 Bull Ant - MLEM Anomaly

Weak to moderate WMC MLEM anomalies are located NW of Currawong along the M-Lens dip direction. EM37 detected a number of weak vertical conductors further NW (Figs. 3, 13).

A weak aeromagnetic feature may be related.

Strong Cu, Pb, Zn soil geochemistry occurs to the NW. Broader Cu, Zn halo anomalies are present (Fig. 10).

Host rocks are Gibson's Folly volcanics and a change of facing has been logged between Currawong and this anomaly. A fold repetition of the Currawong ore horizon is possible, (Fig. 12).

Drill hole BEND 72 was drilled to test the strong Cu, Pb, Zn anomaly and one of the MLEM features. No significant mineralisation was intersected and the anomalies were not adequately explained, (Fig. 10, 12, 14). DHEM was not completed in BEND 72 because of hole collapse near surface.

This prospective area within the Currawong deposit plunge corridor deserves further work.

It is recommended that BEND 72 be cleared, cased and probed with DHEM and magnetics/magnetic susceptibility.

5.2 Bull Ant West - MLEM Anomaly

A similar WMC MLEM anomaly at the limits of WMC coverage, not covered by EM37, is found 700m to the west of the above.

Anomalous Cu, Pb, Zn geochemistry with a Cu, Zn halo is associated. The elevated extensive Cu anomaly may be related to more mafic (andesitic?) lithologies within the Gibson's Folly Formation, (Fig 10).

A bulls-eye magnetic feature occurs close to the Indi fault and may be related, (Fig. 14).

This under explored area is worthy of follow up with fixed loop EM37.

6.1 Banksia EM37 (1996) - MLEM Anomaly

Alate time EM37 response at the edge of the 1996 coverage highlighted by Llew Wynn - 9900E, 6750N (5.4 mV/A Ch7) requires follow up, (Fig. 3).

The response at the nearest WMC MLEM station at 9900E, 6700N is weak.

Anomalous Cu, Pb, Zn on one line 9800E, 6600-6700N occurs 100m along strike and up dip from the target. Host rocks are Towanga Sandstone, (Fig.16).

There is no magnetic expression.

WMC IP data in this area indicates that background resistivities in these rocks are significantly lower (300-500 ohm-m) than in the Gibson's Folly Formation near Wilga and Currawong (500 - 1300 ohm-m).

This target should be followed up with fixed loop EM37 in conjunction with target 6.2.

6.2 Banksia TMI-K/Th-CuPbZn Anomaly

A coincident 600x350m TMI and potassium and potassium/thorium ratio anomaly is located 300m NW of the Banksia grid, WNW of 6.1, (Fig. 1, 2, 16).

Anomalous Cu, Pb, Zn soil geochemistry occurs on a single regional traverse across the above anomaly. Values encountered include from 20-90ppm Cu, 80-250ppm Pb & 40-160ppm Zn in six samples. There is no grid soil coverage in this area.

The anomaly occurs in an area mapped as Bluey's Creek Formation although no outcrop mapping data is recorded on 1:10K geology maps (R.Allen), (Fig. 12). This formation hosts anomalous geochemistry at the Brumby prospect to the NE.

Follow up with geological mapping and Cu, Pb, Zn, Au, Ag, As soil geochemistry on a 25x100m grid is required. Ground magnetics is recommended to accurately locate the magnetic feature. Anomalies should be surveyed with fixed loop EM37 to locate drilling targets.

7 Wombat/Fossil TMI-CuPbZn

A WNW trending aeromagnetic feature is coincident with high Pb and subordinate Zn in soils, (Figs.1, 6-9).

Geology is mapped as Cowombat siltstone overlying chlorite altered Thorkidaan flow banded rhyolite, (Fig. 12).

WMC MLEM coverage is incomplete and only weakly anomalous. However a target still exists between the two MLEM survey areas.

Geological mapping and ground magnetics is recommended to accurately define the magnetic anomaly.

Follow up fixed loop EM37 covering a zone down dip from the high Pb, Zn in soils is required.

It is possible that the anomalous geochemistry is related to fluid migration along a dyke filled structure where it cuts the prospective stratigraphy/structure.

8 Wallaby - MLEM

A strong WMC MLEM anomaly occurs on a single line (2 stations) just inside the National Park a few hundred metres outside ML1865. This anomaly has a coincident magnetic anomaly and

associated moderate Pb soil geochemistry, (Figs. 1, 3, 17). Low IP resistivity (n=4) is associated.

The host rocks are Cowombat siltstone.

The anomaly does not appear to have been followed up by WMC.

Consideration should be given to investigating this feature. Discussion with the relevant government authorities regarding the possibility of excising a small block of land from the National Park is recommended.

Longer Term Exploration Targets:

Longer term exploration targets requiring ground geophysical, geochemical surveys and geological support work are identified as follows:

- North of the existing Wilga-Currawong corridor WMC MLEM coverage for 7kms of prospective stratigraphy, anomalous soil geochemistry and magnetics requires follow up with modern fixed or moving loop EM, (Figs. 1-9, 11, 12).

- Southwest of the South Gibson's Folly prospect, anomalous Pb dominated soil geochemistry and a series of circular magnetic features within prospective stratigraphy should be followed up with EM or IP.

- Approximately 2 kms east of Wilga, a zone of incompletely defined anomalous Cu, Pb, Zn soil geochemistry trends NE through the Kookaburra prospect (Kookaburra Porphyry) for ~2 km. The geochemical signature is similar to the Wilga/Currawong outcrops and the large Wilga South soil anomaly around the porphyry at Boob Hill (Figs. 6-10).

  No EM or IP coverage exists along this zone which occurs within a late phase? of the Thorkidaan volcanic suite.

  It is recommended that grid soil geochemistry be carried out to complete anomaly definition/characterisation followed by blanket fixed or moving loop EM coverage.

- The Brumby geochemical anomaly is associated with a band of discontinuously magnetic stratigraphy which trends SW towards target 6.2 just NE from the existing Banksia grid. A few regional geochemical traverses are the only known exploration in this area (Figs. 1, 7-9, 11-12.).

  It is recommended that this zone be prospected with soil geochemistry (Cu, Pb, Zn, Au, Ag, As) and anomalies followed up with fixed or moving loop EM.

  Investigation of target 6.2 will provide information that will assist with the evaluation and rating of the prospectivity of this zone.

Exploration Targets (Peter Rea):

An earlier exploration review and exploration proposal was completed in March 2001 by consulting geologist, Peter Rea.

Recommendations are summarised in a memorandum to Austminex 14 March 2001, Appendix 2, Figs. 11.

Since then Target 1 (BEND 79) immediately NE along strike from Currawong has been tested by drilling.

Digital capture of WMC's MLEM and IP data was also completed.

Some of Peter's targets are not included with those identified above. They include:

- Nine discrete circular magnetic features (targets 15-23) recommended for follow up with soil geochemical traverses. These are still valid short to longer term targets depending on accessibility, (Fig. 11).

- A 500m zone of greater than 190ppm Zn within a 2km envelope of greater than 250ppm Cu defined by four regional reconnaissance lines on the Wallaby grid near 586100E, 5912400N, Fig. 10.

  The zone lies within the Gibson's Folly formation. The elevated Cu may be indicative of andesitic volcanic host rocks, (Fig. 12).

  Closer spaced soil sampling was recommended on the Wallaby grid lines 3400N-3700N from 7300E-8000E.

  Interesting results were to be followed up with ground EM.

  This work is indirectly covered under the longer-term targets described above.

Past Exploration Targets & Results:

Apart from Wilga & Currawong, WMC followed up a number of other geochemical and geophysical targets by drilling. Subsequent explorers reinterpreted and/or collected new data and carried out further drilling on some of these. The most significant results obtained from this exploration work are summarised below. This textual summary is mostly drawn from Austminex N.L.'s Prospectus, pages 28-30, August 2000.

- Wilga South
  This prospect is located 500m southeast of the Wilga orebody and was identified by WMC using soil geochemistry. Strong Cu,Pb,Zn base metal anomalies are present and 11 holes have tested some of these and various EM anomalies. Three drill holes intersected a stringer chalcopyrite zone in brecciated rhyolite, and one hole (BEND89) returned 16.8m at 1.02% Cu.

- Currawong North
  Diamond drilling by WMC, 250m northeast and along strike from the Currawong resource, intercepted gold and base metal mineralisation, with intersections that included:

  BEND79: 1.6m at 1.43% Cu, 10.2% Pb, 9.4% Zn, 126g/t Ag and 11.48g/t Au from 183.75m.
  BEND80: 1.95m at 3.85g/t Au from 180.25m and 2.9m at 2.9% Pb, 6.1% Zn, 45g/t Ag and 7.39g/t Au from 234.25m.

  This area may have potential for additional gold-rich base metal resources. It is significant to note that trenching of an aeromagnetic high up dip from the Currawong high-grade zone revealed gossanous mafic volcanics. Samples of mineralisation returned 9m at 2.90g/t Au,including 3m at 5.46g/t Au and 45g/t Ag.

- Banksia
  At the Banksia prospect, located approximately 8km south of Wilga, two reconnaissance soil samples returned strong anomalism in copper and lesser anomalism in lead and zinc. Follow-up
  rock chip sampling returned a maximum value of 0.58% Cu, 230g/t Ag and 0.45g/t Au from a sample of rhyolitic volcaniclastics with
  gossanous stockwork veining.

  WMC drilled DDH BEND94 to a depth of 445m, testing a weak EM anomaly and moderately strong geochemical response in the
  central felsic dome. Strong "stringer-style" mineralisation was intersected in silica-sericite altered felsic volcanics and fragmentals and was interpreted as the continuation of the mineralisation noted at surface. The best result was 3.55m at 2.3% Cu and 20g/t Ag from 199.2m.

  In 1994 Denehurst continued exploration at the prospect. A total of eight holes were drilled to test the interpreted "exhalite" position in sediments and volcanics overlying the strongest zones of "stringer" mineralisation in the central felsic volcanic unit.

  Best results were as follows:

  BANKD001: 2.0m at 3.00% Cu and 58g/t Ag from 208.0m.
  2.0m at 2.15% Cu and 40g/t Ag from 249.5m.

  34.2m at 0.57% Cu from 269.8m, including 2.5m at 3.8% Cu and 25g/t Ag from 277m.

BANKD002: 0.5m at 2.74% Cu and 2.10% Zn from 40.4m.

A thorough review of previous exploration and testing in this strongly anomalous area is recommended. The existing drilling is confined to a restricted section of the prospect and targets
remain untested ie. Immediate Targets 6.1 & 6.2.

- Peppermint
  The Peppermint prospect is located approximately 4km east-southeast of Wilga and is underlain by rhyolites, lavas, tuffs, agglomerates and lava breccias of the Upper Silurian Thorkidaan Volcanics. The area was the initial focus of WMC reconnaissance mapping, possibly due to elevated lead geochemistry in soils and the presence of a massive gossan outcrop in the central portion of the grid. WMC drilled 11 holes, focussing on the zones of elevated lead geochemistry and strong IP response, with a maximum result of 2.2m at 0.32% Cu, 1.67% Pb, 3.36% Zn and 6g/t Ag in DDH BEND3.

  Following the WMC work, exploration focussed on disseminated gold mineralisation related to felsic volcanism after recognition of elevated arsenic and gold anomalism in soils over the area. Macquarie trenched a number of soil anomalies, obtaining a maximum 6m at 3.30g/t Au close to the outcrop of the Peppermint gossan.

  Denehurst completed only limited auger soil sampling of arsenic soil anomalies during their search for gold mineralisation in the area.

- Brumby
  Exploration at the Brumby prospect, located 6km southeast of Wilga, was undertaken by WMC between 1974 and 1983 over the Ordovician volcanics and sediments of the Blueys Creek formation. Exploration involved detailed soil sampling, geological mapping, IP, ground magnetics and moving and fixed loop EM. This work continued until 1983 and strong linear geochemical anomalies were detected, along with weak and inconclusive geophysical responses.

  Four shallow diamond drill holes 65-100m deep were completed by WMC to test beneath the most significant of the geochemical responses in soils. The maximum result of 3.1m at 0.2% Cu, 0.4% Pb and 1.04% Zn from 48.0m, in DDH BEND 15, was returned from an intersection of stockwork base metal veining in tuffaceous sediments.

  Exploration by Denehurst involved moving loop Protem and the drilling of an additional four diamond holes to test beneath the strongest soil geochemical responses. All holes intersected a mixed volcano-sedimentary sequence of felsic to intermediate lavas, and volcaniclastics with fine to coarse-grained clastic sediments. Mineralisation was noted in three of the four holes and is similar in nature to that intersected in DDH BEND 15. In addition, minor stratiform pyrrhotite and sphalerite have been noted in tuffaceous sediments in all holes.

- Big Hand
  The Big Hand Prospect is located immediately northeast of the Currawong deposit and occupies the same stratigraphic position along strike. Geological mapping and rock chip sampling returned a maximum result of 16.7g/t Au from a float sample of gossanous siltstone. Follow up trenching returned a maximum value of 9m at 3.01g/t Au, 19g/t Ag, 0.29% Pb and 4200ppm As. Additional exploration by Denehurst involved soil sampling, percussion drilling and approximately 2 line kilometres of 200m moving loop Protem. Auger soil sampling on a 50 x 12.5m grid defined strong gold, arsenic, copper and lead soil anomalies coincident with the areas identified by previous explorers.

Percussion drilling of five holes returned sub-economic scattered copper and zinc values close to the contact between tuffaceous and clastic sediments and a siliceous dacitic volcanic but it is uncertain whether these holes adequately tested the horizon.

Proposed Work Programme & Budget (Ray Hazeldene):

The majority of the above short-term exploration targets and a few resource extension targets at Wilga and Currawong were nominated for investigation in a 4th quarter exploration programmeme and budget prepared by Ray Hazeldene in October 2001.

This programmeme and budget is documented in a memorandum to Austminex 5 October 2001, (Appendix 3).

Geophysical Targets (Llew Wynn):

EM37 data within the Wilga-Currawong corridor and at Banksia was re-interpreted by Llew Wynn.

Llew's Wilga-Currawong EM37 targets are illustrated with rankings and styles in Fig. 13.

An incompletely defined EM37 target was discussed under short term targets above.

A number of Llew Wynn's memoranda covering DHEM and other geophysical targets appear in Appendix 1.

Appendix 1

Memoranda On Geophysical Targets & Models

1. DHEM Models for Drill Hole Bend 77

Llew Wynn

9 July 2001

2. Benambra Exploration Targets (Pages 1 - 2)

Llew Wynn

4 December 2001

Memorandum

To: Kevin Tomlinson (03-96202147)

CC:

From: Llew Wynn

Date: 27/04/2006

Re: DHEM Models for Drill Hole Bend 77

Kevin,

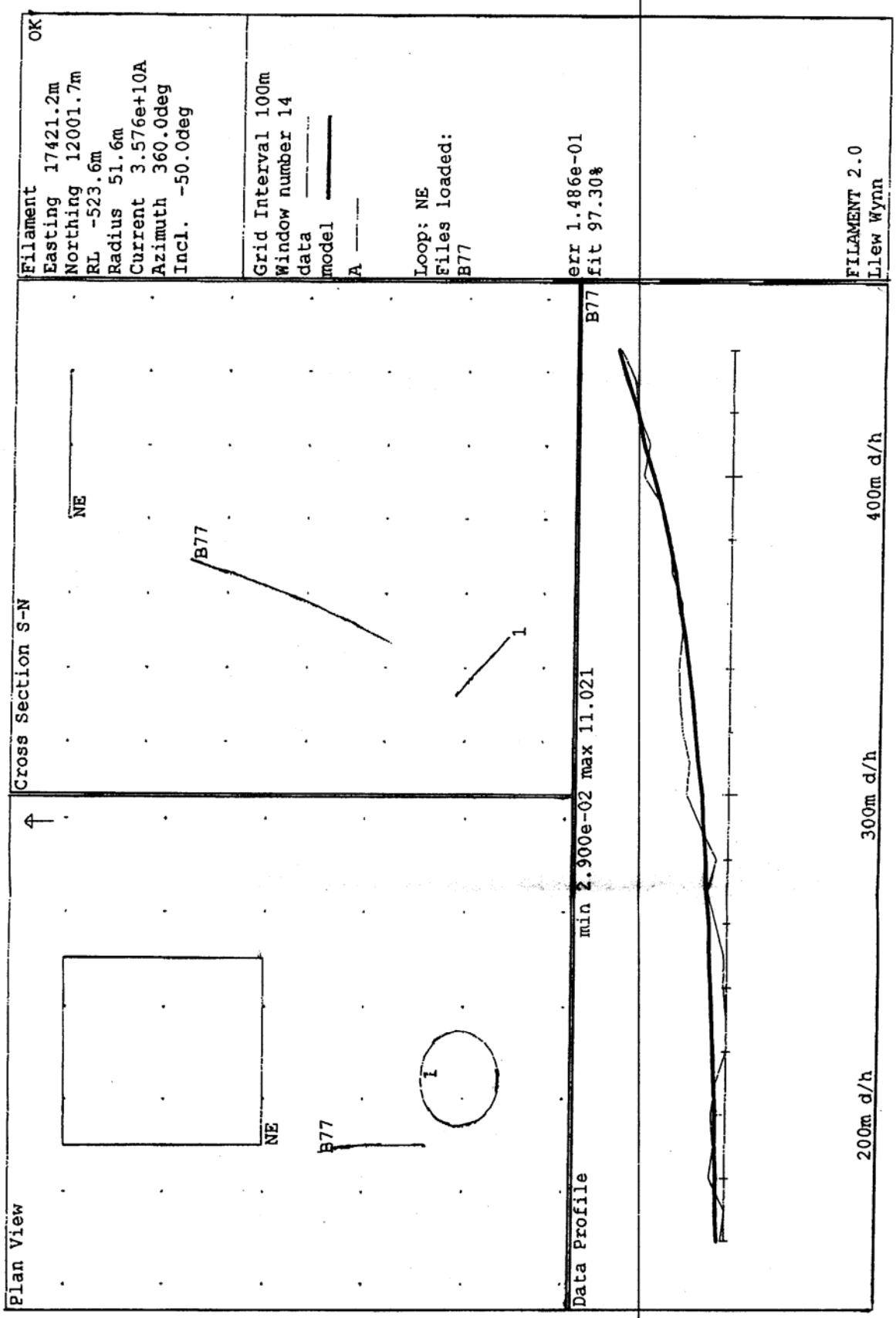
Appended are three figures showing down-hole EM models for Bend 77.

Figure 1 Model with one conductor, Conductor (1)

Figure 2 Model with Conductor (1) and Currawong M-Lens

Figure 3 Model with Currawong M-Lens alone - Conductor (1) turned off.

Regards Llew



Bend 77

Figure 1.

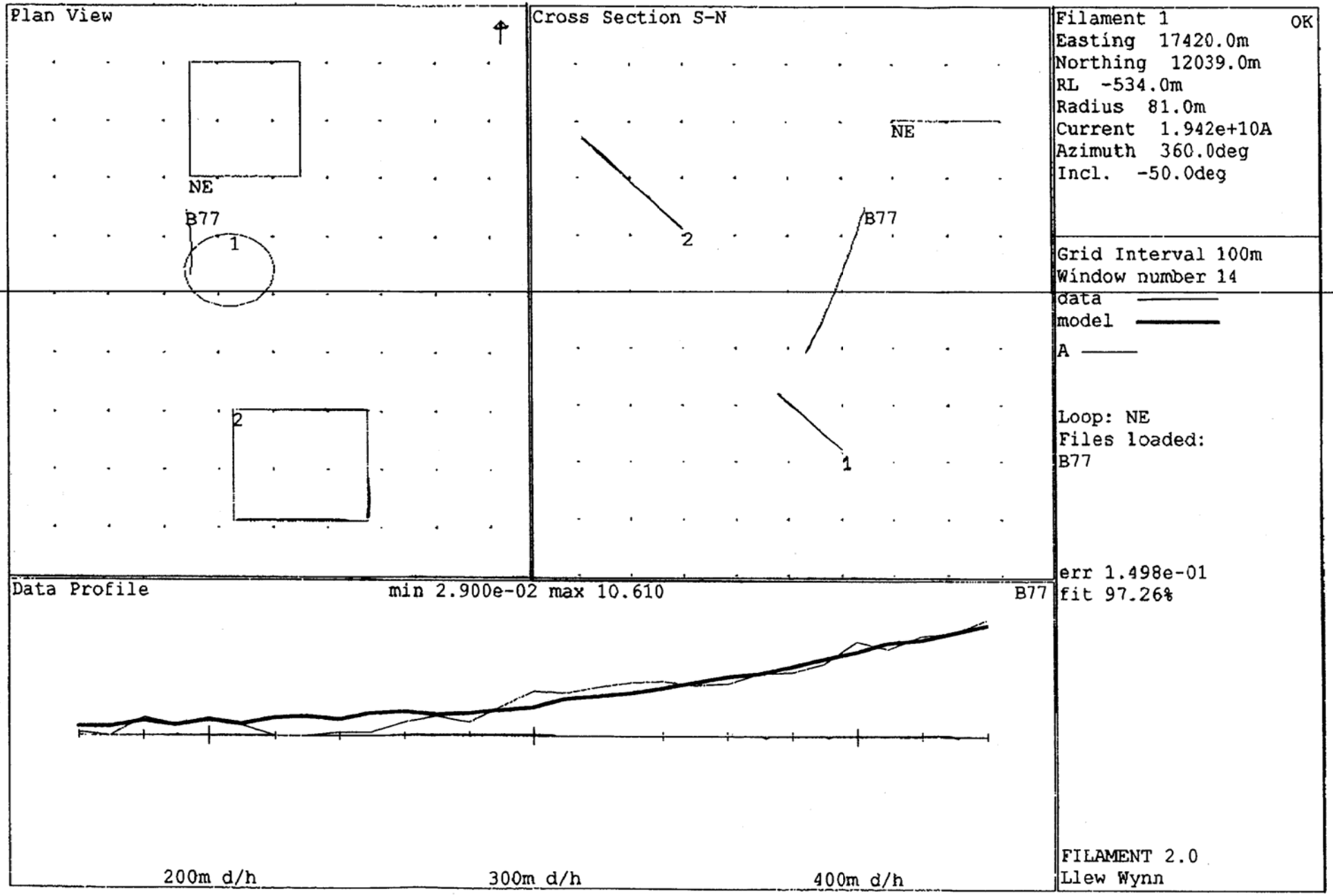


Figure 2

Bend 77 (M lens - conductor 2)

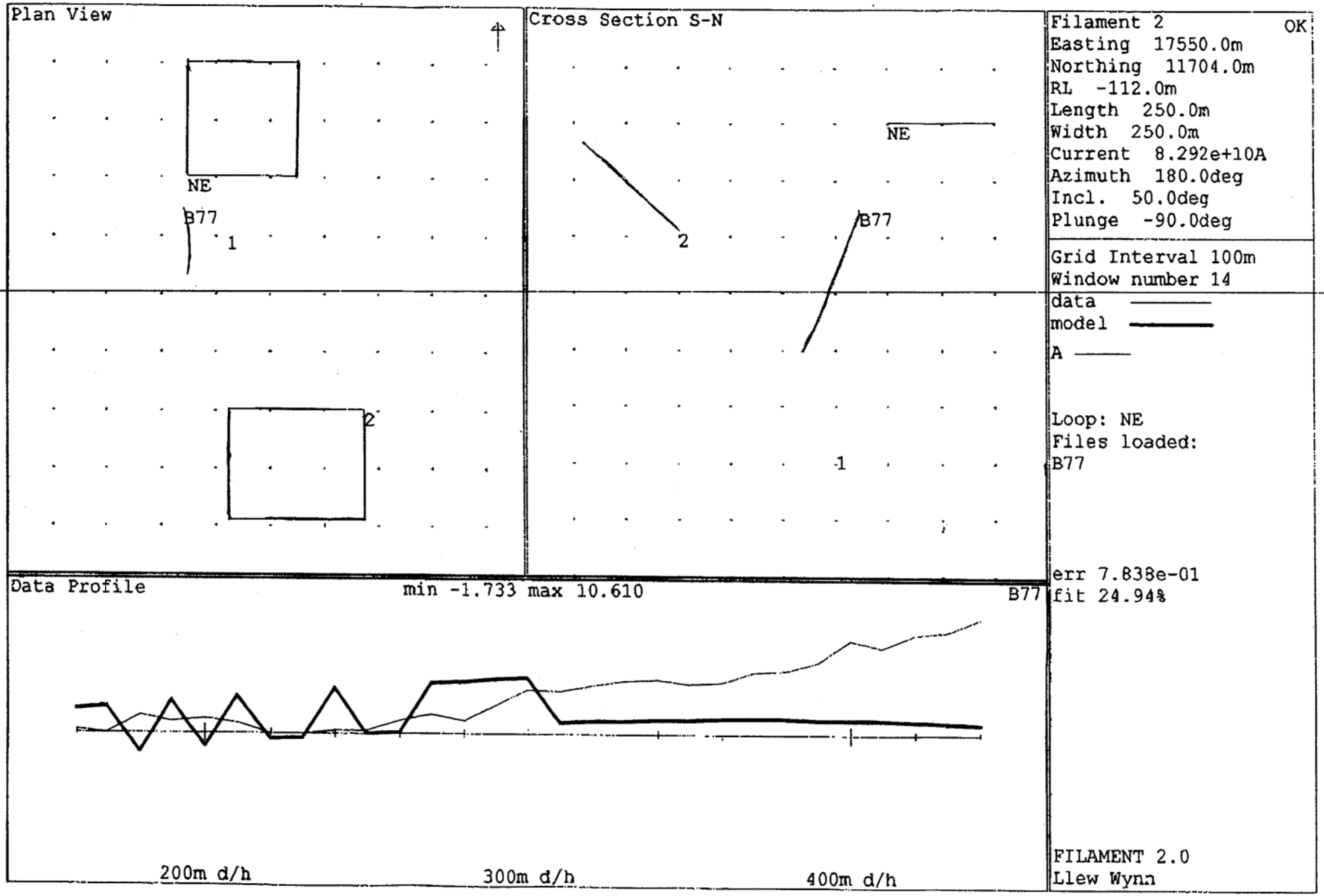


Figure 3

bend 77 (Conductor 1 disabled)

Memo

Unit 4 49 COOK STREET
NORTH WARD, QLD 4810

phone 07-47712662 fax 07-47712662
mobile 0438712662

To: Kevin Tomlinson

From: Llew Wynn

CC: R. Hazeldene, R. Paterson

Date: 27/04/2006

Re: Benambra Exploration Targets

A review of geochemical and geophysical data has been completed and 7 exploration targets have been identified and ranked on merit.

Target 1.

Bend 77. (Currawong).

WMC detected a DHEM anomaly in Bend 77. The DHEM responses increase towards the end of the hole (420m). The DHEM data indicates a subsurface conductor occurs beyond the drill hole. At the time of the survey it was standard procedure to utilise several energising loops positioned around the drill hole to accurately locate the position of subsurface conductors. WMC used four loops to energise the ground around Bend 77. Anomalous DHEM responses were detected in three of the four loops indicating the source of the conductive response was detected with three different loop positions. Initial interpretation positioned the source of the DHEM anomaly beyond the end of the hole and grid east of the hole. Subsequent down hole EM modelling confirmed this interpretation. These responses are very similar to DHEM responses detected above The Dry River South VHMS deposit and above massive sulphides at the Boyds 5 Prospect (Balcooma, North Queensland).

Modelling also indicated the DHEM anomaly is not the response from the M lens at Currawong. DHEM modelling indicates the M lens contributes very little to the anomaly. The very weak contribution from the M lens is caused by;

(a) The distance between the M lens and Bend 77. (The bulk of the M lens is approximately 350m grid south of Bend 77.

(b) The M lens was 350-400m from the edge of the transmitter loop which produced the strongest DHEM response. In addition, the energising field from this loop was poorly coupled with the M lens.

The following observations make this anomaly a high priority.

1. The DHEM anomaly is evident on three WMC loops indicating the subsurface conductor was detected from three different loop positions.

2. Geophysical modelling confirms the initial DHEM interpretation.

3. Geophysical modelling indicates the DHEM anomalous responses are not responses from the Currawong M lens.

Recommendations.

1. Confirm the WMC DHEM anomaly by re-surveying Bend 77 with DHEM.

2. If Bend 77 is blocked, remove the poly in Bend 179, extend the drill hole (250m) and survey with DHEM.

3. Use these DHEM results to site a drill hole to intersect the conductor.

If Bend 179 cannot be extended, arrange to have the DHEM data from Bend 77 modelled by the author of the modelling package to confirm the current model prior to drilling the target.

Target 2.

Currawong Southeast.

WMC detected MLEM anomalies at 18400E-10800N (582510E – 5906160N) and also at

18600E-10800N. These anomalies are associated with anomalous Pb (soils).

Recommendations.

Confirm the MLEM anomalies with a fixed loop EM survey

Target 3.

Currawong South (near Plant).

WMC detected a MLEM anomaly near the plant site. The anomalous MLEM values are noisy and require re-surveying to confirm the WMC results.

A Pb geochemical anomaly (soils) and an aeromagnetic anomaly (detected prior to Plant construction) occur in the vicinity of the MLEM anomaly.

Recommendations.

1.Confirm the MLEM anomaly with a small fixed loop survey (position the **transmitter loop across the gully, away from the Plant).**

2. Conduct ground magnetic traverses to locate the aeromagnetic anomaly.

Appendix 2

Exploration Review & Proposal Memorandum

Peter Rea

16 April 2001

Appendix 3

Benambra Project - 4th Quarter Exploration Programme

Ray Hazeldene

5 October 2001

Appendix 4

Notes To Accompany Benambra GIS CD

Rod Paterson

17 December 2001

Notes To Accompany AUSTMINEX N.L. Benambra GIS CD
Provides a general description of the contents of each directory
And comments on the required software configuration.
11 December 2001

CONFIGURATION & SETUP

1. GIS was setup using MapInfo 6.0 and ERMapper 6.0-6.21
   Discover3.097 was used for location transformations and drill hole trace generation.

2. Extensive use is made of the ERMapper plugin for MapInfo.This plugin allows ERMapper algorithms to be displayed in MapInfo workspaces. The plugin is called MapImagery and is produced by GID. An installable free version is provided on the CD.

3. MapInfo workspaces contain some absolute pathnames ie: D:\aberfoyl\jobs\benambra which are not valid for the CD. To avoid problems with these absolute pathnames go to Options/Preferences/Directories/Search Directories For Tables in MapInfo and add the drive letter of your CD drive to the existing set of search directories.

   Some algorithms and workspaces contain reference to large multilayer datasets and involve some complex internal processing. Consequently their display may be very slow depending on the CDROM drive speed and cache size. If space is available then copying the whole CD to your hard drive will improve display speed. If you do this then the ***Search Directories For Tables*** in MapInfo will need to be set to the appropriate hard drive letter ie. C:\

4. Where possible all located data files contain both local grid and AMG55/AGD66 coordinates within the tables themselves. This allows reference to local grid coords referenced in most hardcopy reports and geochemical & ground geophysical surveys.

   Grid transforms (planar) have been set up in the MapInfo/Discover environment using relationships (local coord/AMG pairs) calculated from the grid tables in the ***grid*** directory and from recent survey information at Wilga & Currawong.

5. The original soil data capture work done by TerraSearch who digitised most of WMC's soil Geochemistry for NRE (Victorian Geological Survey) was found to be incomplete. A significant portion of WMC's database was available as computer printouts only - the missing data was captured using scanning and OCR. The grid transform used by TerraSearch is non planar and does not exactly match the WMC planar transform used for all the other ground data. The captured missing geochemical data was matched to the TerraSearch transform for consistency - however as a result there is a small misfit between all other grid and drillhole datasets and the soil geochemistry in the vicinity of Currawong.

   Some of WMC's regional soil traverses are only roughly located because of terrain correction problems etc. This is apparent where they cross some of the later soil grids (errors of 100m or more). There are comments in WMC monthly reports regarding this problem.

6. The best way to obtain a quick overview of the most prospective areas and examine the critical datasets on the CD is by displaying the workspaces in the ***presentation*** and ***prospectivity*** directories. These workspaces contain multiple datasets which can be turned on and off to observe their relationships to each other and various prospective targets. Transparent overlays of geology and dot plots of geochemical data are also available for comparison.

DIRECTORY DESCRIPTIONS

Cadast

- Tenements (current & expired)
  Parks
  Roads
  Towns
  Restricted/unrestricted Crown land

Drilling

- All WMC collars, surveys and assays
  All post WMC collars, surveys and assays
  Setup in AMG grid for MapInfo/Discover.
  In Excel and MapInfo Tab format.

- All Currawong and Wilga resource data.
  All collars, surveys, assays, alteration, faults, litho in Excel format ex Datamine.
  All coords in Local Grid.

- MapInfo tab files containing collars and hole traces (surface plan) are stored in ***database*** sub directory.

EM_Data

- Root directory contains MLEM coverage diagrams by grid in Mapinfo tab format.

- ***Located_data*** subdirectory contains:
  All WMC MLEM (Moving Loop EM)
  located data (txt, xls, tab),
  images and grids [Chan7 & 10] (ers, bil, alg - ermapper)

- ***Banksia_EM_Dec95***
  BigHand_EM_Feb96
  Brumby_EM_Dec95
  Thes above subdirectories contain Macquarie Resources/Denehurst located ascii EM data and MapInfo tab files with survey station locations etc.

- Digital data is not available for the Mine corridor EM37 Fixed Loop EM survey - see later under Scans directory.

- The ***DHEM*** subdirectory contains all Austminex DHEM raw data files and models/reports by Llew Wynn.

- All the above data in MapInfo tables is in both AMG and Local Grid coordinates.

Geochem

- All WMC and Denehurst soil geochem data (located and gridded data as tab, bil, ers, alg - ERMapper). All soil grids were generated using Discover's Inverse Distance algorithm using 1/D**2 and an elliptical 250mx100m search with the long axis oriented at 40 degrees AMG. All soil grids have been clipped back to the data bounds to remove any possible artifacts
  Internal WMC assay method (Nitric/Perchloric/AAS) and Denehurst (ALS/ICP588) analytical techniques produce different results and ICP588 has lower detection limits The data sets are best dealt with separately - Pb is

worst effected (survey overlap occurs at Banksia East, Peppermint and Big Hand).

- Subdirectory ***Classify*** contains the output of a number of Maximum Likelihood supervised and unsupervised classifications of soil geochemical data (logs, ratios, logratios - raw and standardised).

  This was done to identify geochemical anomalies with features similar to Wilga and Currawong. The coverage of only 3 elements and the relatively high detection limits reduce the effectiveness of the results from a prospectivity and litho-geochemical viewpoint. Results are discussed more fully in the prospectivity report.

Geology

- Contains Victorian government (VIMP) Bairnsdale and Tallangatta 1:100K geology in MapInfo tab format which has been partly modified to remove map sheet boundary problems etc.

- Also contains Bairnsdale and Tallangatta 1:250K geology in tab format.

- Not all VIMP themes are included with the above data because of space limitations on the CD. The full data is available from NRE on CD for no charge.

- The majority of the 1:100K geology near Benambra is based on R.Allen's 1:10K mapping completed as part of his PhD thesis during WMC's tenure. The original 1:10K maps have been scanned as B&W line drawings and can be used as overlays on images, vectors etc. These are stored under the ***Scans*** directory. They contain much more detailed information than the Government maps - including sample and thin section location reference numbers etc.

Gravity

- VIMP gravity stations in MapInfo tab format. More detailed data has just been released.

Grids
Wmcgrids

- Local grid layouts in tab format in AMG55/AGD66. These two directories contain mostly the same information. They need to be combined to remove confusion. Some grids are still not well located with respect to AMG and this could be resolved using GPS control if old grid points can still be found.

IP

- All WMC (Frequency Domain) and Denehurst (Time Domain) dipole/dipole IP located and gridded (n=4) as xls, tab, bil, ers, alg (ermapper) files.

- Also included are the more recent Mine corridor gradient array survey data as images and spreadsheets - original digital data was not located.

MineInfraStruct

- All mine site related MapInfo tab files - plant, tailings dam, roads etc.
- Subdirectory ***survey*** contains files produced by engineering contractor who built access road and taings dam etc. Includes some detailed contour info for roads and tailings dam environs.

Presentation

- Various MapInfo workspaces, bmp, tiff, jpeg, powerpoint files produced for KT's board meetings etc.

Prospectivity

- As above; produced to demonstrate prospective targets after on-site meeting early July 2001. Base directory includes Peter Rea's targets as tab files.

- Peter Rea's final memo/report on exploration targets is stored in subdirectory ***Peter_Rea.***

- Ray Hazeldene's budget report on the follow up of prospective targets identified from the above is stored in subdirectory ***Ray_Hazeldene.***

- Additional reports by Llew Wynn referred to by P.Rea are stored in subdirectory Llew_Wynn.

- Subdirectory ***FinalReport*** contains all workspaces, tiffs, xls and doc files for the final prospectivity report written by the author. There is a tiff file at 96dpi resolution for each figure in the report.

Magnetics

- VIMP and Macquarie Resources aeromagnetic images (ers, alg, erv - ERMapper format).

- Located data in MapInfo tab format can be used to locate flight lines.

- Three separate VIMP aeromagnetic surveys have been levelled with each other using the located data then merged and regridded. The first vertical derivative was computed using located data and then gridded.

Radiometrics

- VIMP radiometrics as for the above. Levelling of this data proved more difficult; levelling was completed using regression on sections of overlapping located data. Uranium data proved too noisy to merge reliably - overlap between Murrindal and Omeo surveys was the main area of difficulty.
  Data was merged and regridded. K/Th ratio appears to be the most useful for mapping and prospectivity analysis.

Scans

- ***Geology***

  This subdirectory contains Rod Allen's 1:100K reduction/compilation of his 1:10K mapping. Scanned from B&W line drawing transparency and imported and registered in MapInfo. Useful as transparent overlay

- ***Geology10k***

  Rod Allen's 1:10K mapping scanned, imported and registered in MapInfo as above.
  Most detailed summary geology available - contains rock sample and thin section location identifiers.
- ***LlewsTargets***

  EM37 anomaly map and survey coverage for mine corridor; scanned Lllew Wynns hand drawn map on tracing paper and imported into MapInfo as a raster overlay; digitised and entered anomalies in attributed MapInfo table.

- ***Topo10K***

  WMC 1:10k topographic contours, tracks and streams (produced by Photec Air Surveys form 1:80K aerial photography). Transparencies scanned and imported into MapInfo as transparent raster overlays.

Rod Paterson
Consulting Geologist/Geophysicist
10 Howitt Street
Glen Iris Vic 3146
Australia

Phone: **61-3-98892793**
Fax: **61-3-86100373**
Email: **rpaterso@bigpond.net.au**

9 December 2001